THE MINERAL INDUSTRY

ITS

STATISTICS, TECHNOLOGY AND TRADE

DURING

1907

FOUNDED BY RICHARD P. ROTHWELL

EDITED BY

WALTER RENTON INGALLS

Editor of the Engineering and Mining Journal; Member American Institute of Mining Engineers; Member Institution of Mining and Metallurgy; Member Mining and Metallurgical Society of America; Member American Chemical Society; Member Society of Chemical Industry, Etc.

VOLUME XVI

SUPPLEMENTING VOLUMES I TO XV

HILL PUBLISHING COMPANY

505 PEARL STREET, NEW YORK

6 BOUVERIE STREET, LONDON, E.C.

The Engineering and Mining Journal—Power—American Machinist

1908

Hill Publishing Company, New York, U. S. A.

CONTRIBUTORS TO VOLUME XVI.

ATWATER, C. G.
Engineer, United Coke and Gas Company, New York.
Ammonia and Ammonium Sulphate.
Plants for Making Sulphate of Ammonia.
By-Product Coke Ovens.

AUERBACH, HERBERT S.
General Manager, United Golden Chest Mining and Milling Company, Murray, Ida.
Tungsten Mining in Idaho.

AUSTIN, L. S.
Professor of Metallurgy, Michigan College of Mines, Houghton, Mich.
Progress in the Metallurgy of Copper.

BAIN, H. FOSTER
Director, State Geological Survey, Urbana, Ill.
Cement in Illinois.
Fluorspar in Illinois.
Petroleum in Illinois.
Zinc Mining in Illinois.

BARBOUR, ERWIN H.
State Geologist, Lincoln, Neb.
Coal Mining in Nebraska.
Silica Production of Nebraska.

BEASON, L. H.
Correspondent of the *Engineering and Mining Journal*, Salt Lake City, Utah.
Coal Mining in Utah.
Copper Mining in Utah.
Gold Mining in Utah.
Salt Mining in Utah.

BEELER, H. C.
State Geologist, Cheyenne, Wyo.
Asbestos Mining in Wyoming.
Coal Mining in Wyoming.
Copper Mining in Wyoming.
Gold and Silver Mining in Wyoming.
Graphite Mining in Wyoming.
Gypsum in Wyoming.
Iron Ore Production of Wyoming.
Limestone in Wyoming.

BELL, ROBERT N.
State Mining Inspector, Boise, Ida.
Antimony Mining in Idaho.
Copper Mining in Idaho.
Gold and Silver Mining in Idaho.
Lead Mining in Idaho.
Zinc Mining in Idaho.

BLAKE, WILLIAM P.
Territorial Geologist, Tucson, Ariz.
Bismuth in Arizona.
Molybdenum in Arizona.
Tungsten Mining in Arizona.
Vanadium Mining in Arizona.

BLATCHLEY, W. S.
State Geologist, Indianapolis, Ind.
Iron Ore Deposits of Indiana.

BOHN, C. A.
Mining Engineer, Mexico City, Mexico.
Petroleum Industry in Mexico.

BRINSMADE, ROBERT B.
Director, Wisconsin State Mining Trade School, Platteville, Wis.
Copper Mining in Arizona.

BUCKLEY, E. R.
Late State Geologist of Missouri; now Geologist for the Federal Lead Co., Flat River, Mo.
Barytes Mining in Missouri.
Iron Ores of Missouri.
Limestone in Missouri.
Nickel and Cobalt in Missouri.
Silver Production of Missouri.
Zinc Mining in Missouri.

CAMERON, WILLIAM
United States Mine Inspector, So. McAlester, Oklahoma.
Coal Mining in Oklahoma.

CHANNING, J. PARKE
Consulting Engineer; President of the Tennessee Copper Company, New York.
Copper Mining and Smelting by the Tennessee Copper Company.

CIRKEL, FRITZ
Mining Engineer, Montreal, Canada.
Mica Mining in Ontario.

CLARK, WILLIAM BULLOCK
State Geologist, Baltimore, Md.
Coal Mining in Maryland.
Iron Mining in Maryland.

COLCORD, F. F.
Cashier and Chief Clerk, U. S. Metals Refining Company, Chrome, N. J.
Sampling and Assaying.

COLLINS, GEORGE E.
Mining Engineer, Denver, Colo.
Gold and Silver Mining in Colorado.

COLVOCORESSES, G. M.
Mining Engineer, 43 Exchange Pl., New York.
Nickel and Cobalt Mining in New Caledonia.

DARROCH, JAMES
Metallurgist, Glasgow, Scotland.
The Occurrence and Metallurgy of Bismuth.

DENNIS, WILLIAM B.
General Manager, Black Butte Quicksilver Company, Black Butte, Ore.
Quicksilver Mining in Oregon.

DOUGLAS, JAMES
President, Copper Queen Consolidated Mining Company, New York.
Copper in Arizona in 1907.

DWIGHT, ARTHUR S.
Mining Engineer, 25 Broad St., New York.
The Dwight and Lloyd Sintering Process.

EAMES, JR., RICHARD
Mining Engineer, Salisbury, N. C.
Copper Mining in North Carolina.

EASTON, STANLEY A.
General Manager, Bunker Hill & Sullivan Mining and Concentrating Company, Kellogg, Ida.
Coeur d'Alene District of Idaho.

FARRELL, JOHN R.
Mining Engineer, San Francisco, Cal.; late Chief Engineer for Tanganyika Concessions, Ltd.
Copper Mining in the Congo Free State
Tin Mining in the Congo Free State

FICHTEL, C. L. C.
Correspondent of the *Engineering and Mining Journal*, Calumet, Mich.
Copper Mining in Lake Superior District.

FITTS, R. F.
Greeley, Colo.
The Preparation of Mica in the United States.

FRASER, LEE
Saginaw, Mich.
Coal Mining in Michigan.

FRIEDMAN, L. W.
Correspondent of the *Engineering and Mining Journal*, Birmingham, Ala.
Coal Mining in Alabama.
Iron and Steel in Alabama.

FRY, SYDNEY
Director, Westport School of Mines, Westport, New Zealand.
Copper at Mount Radiant, New Zealand.

FULTON, CHAS. H.
President, South Dakota State School of Mines, Rapid City, S. D.
Cyanidation during 1907.

GEORGE, HAROLD C.
Instructor in Surveying and Mechanics, Wisconsin State Mining Trade School, and Mining Engineer for the Hodge, Empire and Acme mines, Platteville, Wisconsin.
Petroleum.

GIBSON, THOMAS W.
Deputy Minister of Mines, Toronto, Ont.
Nickel and Cobalt in Ontario.

GOULD, CHARLES N.
Professor of Geology, State University of Oklahoma, Norman, Okla.
Asphalt in Oklahoma.
Coal Mining in Oklahoma.
Gypsum Mining in Oklahoma.

GRIMSLEY, G. P.
Assistant State Geologist, Morgantown, W. Va.
Glass Sand Industry in West Virginia.
Limestone in West Virginia.

HAVARD, F. T.
Metallurgical Engineer, New York.
The French Process for the Treatment of Antimony Ores.

HAWORTH, ERASMUS
State Geologist, Lawrence, Kan.
Natural Gas in Kansas and Oklahoma.
Petroleum in Kansas and Oklahoma.

HIGGINS, EDWIN
Assistant Editor of the *Engineering and Mining Journal*.
Notes on the Milling of Barytes and the Manufacture of Barium Products.
Borax.
Chrome Ore in Foreign Countries.
Copper Mining in Montana.
Gold and Silver Mining in Montana.
Also many unsigned contributions.

HOBART, FREDERICK
Associate Editor, *Engineering and Mining Journal*.
Gold and Silver.
Iron and Steel.

HOFMAN, H. O.
Professor of Metallurgy, Massachusetts Institute of Technology, Boston, Mass.
Recent Improvements in Lead Smelting.

HOHAGEN, GEORG
Director of the Casa Nacional de Moneda and Director of the School of Mines, Potosi, Bolivia.
Review of Tin Mining Progress in Bolivia.

HOTCHKISS, W. O.
Assistant State Geologist, Madison, Wis.
Graphite Mining in Wisconsin.
Manganese Ore in Wisconsin.

HUTCHINS, JOHN POWER
Mining Engineer, New York City.
Gold Mining on Seward Peninsula, Alaska.
Gold Mining in Yukon Territory.
Gold Mining in South America.

HUTCHINSON, W. SPENCER
Consulting Mining Engineer, Boston, Mass.
Barytes in Nova Scotia.

HYDE, FREDERIC S.
Consulting Chemist, Brooklyn, N. Y.
Some Characteristics of Natural Graphite.

INGALLS, WALTER RENTON
Editor of the *Engineering and Mining Journal.*
Valuation of Antimony Ore.
Copper.
Lead.
Tin.
Zinc.
Also some unsigned contributions.

JACOBS, E.
Editor, *British Columbia Mining Record,* Victoria, B. C., and Correspondent of *Engineering and Mining Journal.*
Copper Mining in British Columbia.
Gold and Silver Mining in British Columbia.

JAMES, ALFRED
President, Institution of Mining and Metallurgy; Consulting Mining Engineer, London, England.
Progress in Gold Ore Treatment during 1907.

JONES, JOHN D.
Chief Mine Inspector, Denver, Colo.
Coal Mining in Colorado.

JUDD, E. K.
Instructor in Mining, Columbia University, New York.
Arsenic in New York.
Asphalt.
Barytes.
Vanadium.

KEMP, JAMES FURMAN
Professor of Geology, Columbia University, New York.
A Review of the Literature on Ore Deposits in 1907.

KENNEDY, J. E.
Correspondent of the *Engineering and Mining Journal,* Platteville, Wis.
Zinc Mining in Wisconsin.

KEYES, CHARLES R.
Consulting Mining Engineer, Socorro, N. M.
Copper Mining in New Mexico.
Gold and Silver Mining in New Mexico.
Iron Ore in New Mexico.
Zinc Mining in New Mexico.

KÜMMEL, HENRY B.
State Geologist, Trenton, N. J.
Cement Industry of New Jersey.
Graphite Mining in New Jersey.
Iron Ore Production in New Jersey.

KUNZ, GEORGE FREDERICK
Gem Expert of Tiffany & Co., New York.
Precious Stones.

LANE, ALFRED C.
State Geologist, Lansing, Mich.
Bromine Industry in Michigan.
Cement in Michigan.
Coal Mining in Michigan.
Limestone in Michigan.
Salt in Michigan.

LEES, JAMES H.
Assistant State Geologist, Des Moines, Iowa.
Coal Mining in Iowa.
Iron Ore in Iowa.
Lead Mining in Iowa.
Zinc Mining in Iowa.

LEONARD, A. G.
State Geologist, Grand Forks, N. D.
Cement Industry of North Dakota.
Coal Mining in North Dakota.

LESLEY, ROBERT W.
Vice President, American Society for Testing Materials, Philadelphia, Penn.
Cement.

LEWIS, JAMES B.
General Manager, Anchor Tin Mining Company, Lottah, Tasmania.
Tin Mining in Tasmania.

LOCKE, CHARLES E.
Assistant Professor of Mining, Massachusetts Institute of Technology, Boston, Mass.
Progress in Gold Milling in 1907.
Progress in Ore Dressing and Coal Washing in 1907.

LOZE, ED.
Mining Engineer, Arras (Pas-de-Calais), France.
Coal Industry of France.

LUTY, S. F.
Correspondent, *Engineering and Mining Journal,* Pittsburg, Penn.
Pittsburg District Coal in 1907.
Iron and Steel in the Pittsburg District in 1907.

MANCE, F. S.
Dept. of Mines, Sydney, N. S. W., Australia.
Coal Mining in New South Wales.
Copper Mining in Australia.
Gold and Silver Mining in Australasia.

MARTIN, W. G.
Electrical and Metallurgical Engineer, Denver, Colo.
Valuation of Western Zinc Ores.

MEEKS, REGINALD
Editorial Staff, *Engineering and Mining Journal*, New York.
Arsenic.
Molybdenum.
Nickel and Cobalt.
Nitrate of Soda.
Potassium Salts.

MORRISON, A. C.
Chicago, Ill.
Calcium Carbide and Acetylene.

MORRISON, E.
Correspondent, *Engineering and Mining Journal*, Chicago, Ill.
Chicago Coal Market in 1907.
Chicago Iron and Steel Market in 1907.

NEWLAND, D. H.
Assistant State Geologist, Albany, N. Y.
Cement Industry of New York.
Feldspar Mining in New York.
Garnet in New York.
Graphite in New York.
Gypsum in New York.
Iron Mining in New York.
Limestone in New York.
Pyrite Mining in New York.
Salt Mining in New York.
Talc in New York.

PACKARD, GEORGE A.
Mining Engineer, Boston, Mass.
Copper Mining in Vermont.

PALMER, LEROY A.
Stockton, Utah.
Gypsum Mining in Michigan.

PARSONS, FLOYD W.
Associate Editor, *Engineering and Mining Journal*, New York.
Coal and Coke.

PAUTRAT, G.
Chemical Engineer and Metallurgist, Antimoine-Collet, Lozère, France.
Notes on Antimony Smelting.

PERKINS, G. H.
State Geologist, Burlington, Vt.
Talc and Soapstone in Vermont.

PRATT, JOS. HYDE
State Geologist, Chapel Hill, N. C.
Monazite in the Carolinas.

READ, T. T.
Professor of Mining and Metallurgy, Pei-Yang University, Tientsin, China.
Coal Mining in China.
Copper Mining in China.
Tin Mining in China.

RICE, CLAUDE T.
Editorial Staff, *Engineering and Mining Journal*, New York.
Corundum and Emery.
Talc and Soapstone.

RICHARDS, JOSEPH W.
Professor of Metallurgy, Lehigh University, South Bethlehem, Penn.
Aluminum.

RICHARDS, ROBERT H.
Professor of Mining, Massachusetts Institute of Technology, Boston, Mass.
Progress in Gold Milling in 1907.
Progress in Ore Dressing and Coal Washing in 1907.

ROGOVIN, I. I.
Mining Engineer, St. Petersburg, Russia.
Coal and Coke in Russia.
Copper Mining in Russia.
Gold Mining in Russia.
Petroleum in Russia.
Zinc Mining in Russia.

ROSS, DAVID
Secretary, Illinois Bureau of Labor Statistics, Springfield, Ill.
Coal Mining in Illinois.

ROWE, J. P. W.
Carbonado.

RUHM, H. D.
Civil and Mining Engineer, Mt. Pleasant, Tenn.
Phosphate Mining in Tennessee.

RUTLEDGE, J. J.
Mining Engineer, Baltimore, Md.
Pyrites.

SELLARDS, E. H.
State Geologist, Tallahassee, Fla.
Fuller's Earth in Florida.

SHERIDAN, J. E.
United States Mine Inspector, Silver City, N. M.
Coal Mining in New Mexico.

SMITH, EUGENE A.
State Geologist, University, Ala.
Bauxite Mining in Alabama.
Cement in Alabama.
Coal Mining in Alabama.
Gold Mining in Alabama.
Graphite Mining in Alabama.
Iron Ores of Alabama.
Mica Mining in Alabama.
Pyrite Mining in Alabama.

SMITH, R. V.
Professor of Mining and Metallurgy, New Mexico School of Mines, Socorro, N. M.
Copper Production of New Mexico.
Zinc Mining in New Mexico.

SPEIER, PAUL
Metal Merchant, Breslau, Germany.
The Metallurgy and Uses of Cadmium.

STICHT, ROBERT
General Manager, Mount Lyell Mining and Railway Company, Queenstown, Tasmania.
Mining and Smelting at Mount Lyell, Tasmania.

Strangways, H. F.
Chrome Ore Mining and Milling in Canada.

Tarr, R. P.
Mining Engineer, Northwestern Improvement Company, Tacoma, Wash.
Coal Mining in Washington.

Tone, F. J.
Works Manager, Carborundum Company, Niagara Falls, N. Y.
Carborundum.

Van Zwaluwenburg, A.
Editorial Staff, *Engineering and Mining Journal*, New York.
Glass.
Gold and Silver Mining in Mexico.
Gypsum.

Veatch, A. C.
Geologist, U. S. Geological Survey, Washington, D. C.
Notes on the Mining Laws of Western Australia.

Veatch, Otto
Assistant State Geologist, Atlanta, Ga.
Bauxite in Georgia.

Vogel, Felix A.
Mining Engineer, New York.
Pyrites Mining and Milling in St. Lawrence County, N. Y.

Weed, Walter Harvey
Mining Geologist, New York.
Copper Mining in Servia.

Wheeler, H. A.
Consulting Mining Engineer, St. Louis, Mo.
Southeast Missouri Lead District.

Wilkinson, W. Fischer
Consulting Mining Engineer, London, Eng.
Gold Mining in the Transvaal.
Gold Mining in Rhodesia.
Gold Mining in West Africa.

Woodbridge, Dwight E.
Mining Engineer, Duluth, Minn.
Lake Superior Iron Ore.

Yale, Charles G.
Associate Editor, *Engineering and Mining Journal*, San Francisco, Cal.
Chromium and Chrome Ore.
Quicksilver.

Zook, Jesse A.
Correspondent of the *Engineering and Mining Journal*, Joplin, Mo.
Lead Production of the Joplin District.
Zinc Production of the Joplin District.

CONTENTS OF VOLUME XVI.

VALUES OF FOREIGN COINS.

ESTIMATE BY DIRECTOR OF THE MINT, JAN. 1, 1908.

COUNTRY.	Standard	Monetary Unit.	Value in Terms of U.S. Gold Dollar.	Coins.
Argentine Republic	Gold	Peso	$0.965	Gold: argentine ($4.824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	.203	Gold: 10 and 20 crowns. Silver: 1 and 5 crowns.
Belgium	Gold	Franc	.193	Gold: 10 and 20 francs. Silver. 5 francs
Bolivia	Silver	Boliviano	.429	Silver: boliviano and divisions.
Brazil	Gold	Milreis	.546	Gold: 5, 10, and 20 milreis. Silver: ½, 1, and 2 milreis.
British Possessions, N. A. (except Newf'nd).	Gold	Dollar	1.000	
Central Amer. States—				
Costa Rica	Gold	Colon	.465	Gold: 2, 5, 10, and 20 colons ($9.307). Silver: 5, 10, 25, and 50 centimos.
British Honduras	Gold	Dollar	1.000	
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	.429	Silver: peso and divisions.
Chile	Gold	Peso	.365	Gold: escudo ($1.825), doubloon ($3.650), and condor ($7.300). Silver: peso and divisions.
China	Silver	Tael: Amoy	.704	
		Tael: Canton	.702	
		Tael: Cheefoo	.673	
		Tael: Chin Kiang	.688	
		Tael: Fuchau	.651	
		Tael: Haikwan (customs)	.716	
		Tael: Hankow	.659	
		Tael: Kiaochow	.682	
		Tael: Nankin	.697	
		Tael: Niuchwang	.660	
		Tael: Ningpo	.677	
		Tael: Peking	.686	
		Tael: Shanghai	.643	
		Tael: Swatow	.650	
		Tael: Takau	.708	
		Tael: Tientsin	.682	
		Dollar: Hongkong	.463	
		Dollar: British	.463	
		Dollar: Mexican	.466	
Colombia	Gold	Dollar	1.000	Gold: condor ($9.647) and double condor. Silver: peso.
Denmark	Gold	Crown	.268	Gold: 10 and 20 crowns.
Ecuador	Gold	Sucre	.487	Gold: 10 sucres ($4.8665). Silver: sucre and divisions.
Egypt	Gold	Pound (100 piasters)	4.943	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters. Silver: 1, 2, 5, 10, and 20 piasters.
Finland	Gold	Mark	.193	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	.238	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound sterling	4.866½	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Gold	Drachma	.193	Gold: 5, 10, 20, 50, and 100 drachmas. Silver: 5 drachmas.
Haiti	Gold	Gourde	.965	Gold: 1, 2, 5, and 10 gourdes. Silver: gourde and divisions.
India (British)	Gold	Pound sterling*	4.866½	Gold: sovereign (pound sterling). Silver: rupee and divisions.
Italy	Gold	Lira	.193	Gold: 5, 10, 20, 50, and 100 lire. Silver: 5 lire.
Japan	Gold	Yen	.498	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.

NOTE.—The coins of silver-standard countries are valued by their pure silver contents, at the average market price of silver for the three months preceding January 1, 1908.

* The sovereign is the standard coin of India, but the rupee ($0.3244⅓) is the current coin, valued at 15 to the sovereign.

COUNTRY.	Standard	Monetary Unit.	Value in Terms of U.S. Gold Dollar.	Coins.
Liberia	Gold	Dollar	1.000	
Mexico	Gold	Peso†	.498	Gold: 5 and 10 pesos. Silver: dollar‡ (or peso) and divisions.
Netherlands	Gold	Florin	.402	Gold: 10 florins. Silver: 2½, 1 florin, and divisions.
Newfoundland	Gold	Dollar	1.014	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	.268	Gold: 10 and 20 crowns.
Panama	Gold	Balboa	1.000	Gold: 1, 2½, 5, 10, and 20 balboas. Silver: peso and divisions.
Persia	Silver	Kran	.079	Gold: ¼, 1, and 2 tomans ($3.409). Silver: ¼, ½, 1, 2, and 5 krans.
Peru	Gold	Libra	4.866½	Gold: ½ and 1 libra. Silver: sol and divisions.
Philippine Islands	Gold	Peso	.500	Silver peso: 10, 20, and 50 centavos.
Portugal	Gold	Milreis	1.080	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	.515	Gold: 5, 7½, 10, and 15 rubles. Silver: 5, 10, 15, 20, 25, 50, and 100 copecks.
Spain	Gold	Peseta	.193	Gold: 25 pesetas. Silver: 5 pesetas.
Straits Settlements	Gold	Pound sterling§	4.866½	Gold: sovereign (pound sterling). Silver: dollar and divisions.
Sweden	Gold	Crown	.268	Gold: 10 and 20 crowns.
Switzerland	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	.044	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	1.034	Gold: peso. Silver: peso and divisions.
Venezuela	Gold	Bolivar	.193	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

† Seventy-five centigrams fine gold.
‡ Value in Mexico, $0.498.
§ The current coin of the Straits Settlements is the silver dollar issued on Government account, and which has been given a tentative value of $0.567758¼.

INTRODUCTION.

In the preparation of the statistics for this volume, the figures previously reported for 1906, and in some cases for earlier years, have been revised in the light of later and more minute investigation, in accordance with our regular practice; therefore it is important for all who have occasion to refer to them to observe the caution to *use always the figures in the latest volume of* THE MINERAL INDUSTRY. There are no statistical reports of this nature which are absolutely correct, owing to the practical impossibility of obtaining accurate reports from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers, which prevents them from making returns, the unwillingness of a few to give their figures, and the confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our own statistics, but also in the statistics reported by various governments. In our own work, however, we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation. In estimating values, we are disposed to use actual market prices rather than the values reported by the producers themselves, which are apt to be misleading for the reasons mentioned above.

For many of the statistics relating to the mineral production of the United States in 1907 and previous years, we are indebted to the United States Geological Survey, and for the production of gold and silver in the United States to Frank E. Leach, Director of the Mint. Acknowledgment is due, also, to various State geological surveys and statistical bureaus for information incorporated in this volume. In the text and footnotes to the various tables, we have generally credited such information to the proper sources, but this acknowledgment may stand for any unintentional oversights. The same acknowledgment is due with respect to the foreign statistics, which we state always as officially reported by the respective governments, when such reports are available.

In publishing this volume of THE MINERAL INDUSTRY at a comparatively

early date in the year following that to which it especially relates, the editorial work upon it having been completed at the end of June, and the last proof-reading having been done in July, it has been impossible to collect statistics for all the substances of mineral production in the United States, but the omissions are generally in the cases of substances of minor importance. In many instances it has been possible to make use of the statistics collected by the U. S. Geological Survey in the case of substances in which an independent investigation in behalf of The Mineral Industry was not made. Many of the statistics of the U. S. Geological Survey are now published with commendable promptness and with an accuracy that makes it unnecessary to enter into the same duplication as formerly. The statistics for foreign countries are given in all cases for the latest year available. In several cases our work has been greatly facilitated by the courtesy of the statisticians of foreign governments in sending us their reports for 1907 in manuscript in advance of the regular publication in print. This has been highly helpful in enabling us to present this complete summary for 1907 so soon after the close of the year. In the preparation of The Mineral Industry there is always a temptation to postpone the conclusion of the editorial work pending the receipt of missing figures, but it has been considered that May 31 is a reasonable date at which to draw the line, with a view to combining the maximum of completeness with the maximum of promptness. Because of a variety of circumstances it was impossible to conclude the work for Vol. XVI until June 30, but it is hoped that the next volume will be finished about a month earlier which as above remarked is the irreducible minimum.

Some of the statistics reported in this volume are preliminary, and subject to revision. It is our belief, however, that statistics of reasonable commercial accuracy, promptly published, are of much greater value to technology and trade than are statistics, corrected to the last unit, which are published a year or two late. However, it has been necessary to use these approximations in only a few instances, and it is hoped that most of the figures which are to be found in this book will prove to require only slight, if any, revision.

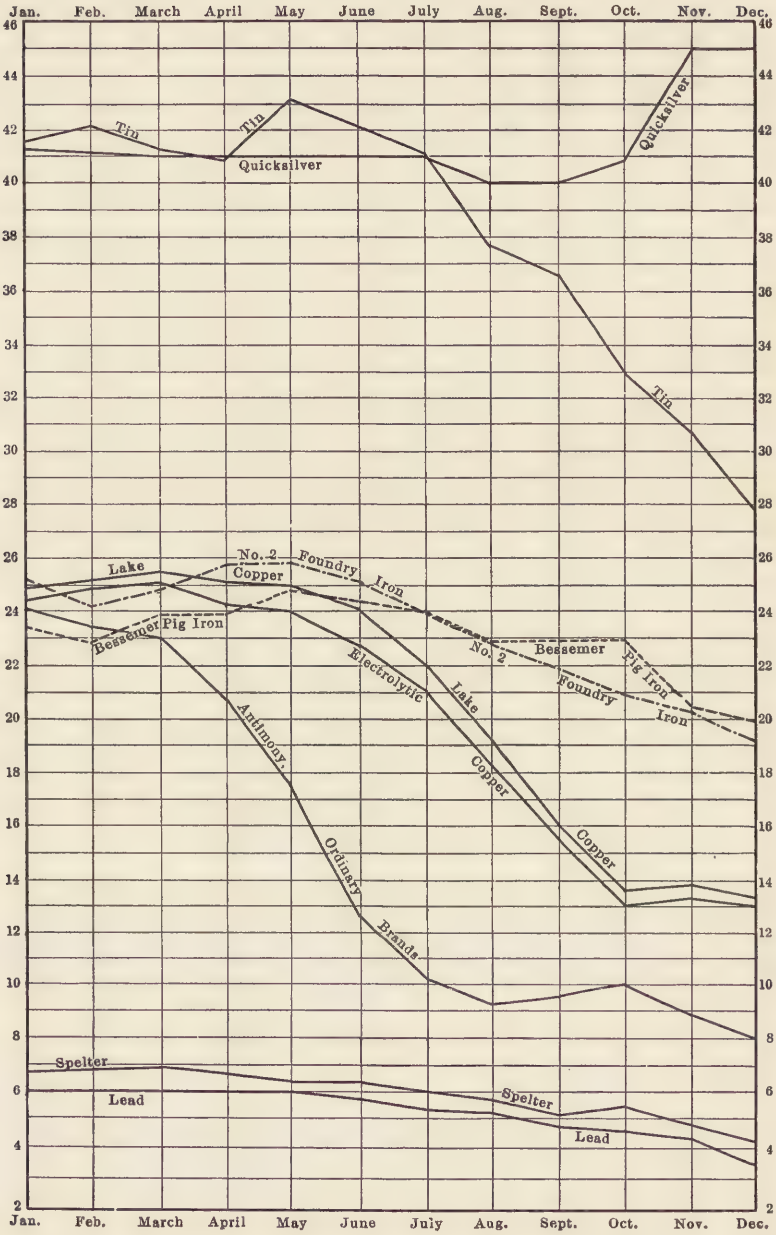
As in Vol. XV, we have divided the large table, which previously followed this introduction, into three sections, one representing the production of ores and minerals in the United States; the second representing the production of metals of domestic origin; and the third representing the manufacture of chemical products, or other articles of commerce that are derived from the ores or minerals directly mined. It is conceived that this arrangement is more logical, and more valuable than one in which all the substances are heterogeneously assembled. We have also added in the summary data as to the production of some of the rarer metals, the trade in which is newly developing in the United States. Thus in 1907 there was

for the first time a production of bismuth and cadmium. In all of the tables values are reported merely as an indication of the relative magnitude of the various industries, from the commercial standpoint. As previously noted, comparatively little weight is to be placed upon value figures. For example, as a matter of form, all the copper produced in the United States is computed on the basis of the average price for Lake copper, in conformity with previously existing custom, although, as it appears in the detailed statistics, by far the major portion of the copper produced in the United States is sold as electrolytic copper, which fetches a slightly lower price than Lake; there are similar differences with regard to spelter, and other metals.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES.

Products.	Measures.	1906		1907	
		Quantity.	Value.	Quantity.	Value.
Antimony ore	Sh. T.	295	$ 44,250	210	$ 28,432
Asbestos	Sh. T.	1,695	20,565	950	11,700
Asphaltum (*u*)	Sh. T.	116,653	1,066,019	223,861	2,826,489
Barytes	Sh. T.	63,486	252,719	65,579	251,308
Bauxite	Lg. T.	78,331	352,490	(*u*)97,776	(*u*)480,330
Borax	Sh. T.	58,173	1,182,410	....	....
Chrome ore	Lg T.	180	1,800	335	5,620
Coal, anthracite	Sh. T.	72,209,566	166,307,002	86,341,832	193,653,218
Coal, bituminous	Sh. T.	341,612,837	401,717,090	388,222,868	463,654,776
Diatomaceous earth (*u*)	Sh. T.	8,099	72,108	....	104,406
Emery	Sh. T.	2,147	22,780	(*u*)1,069	(*u*)12,294
Feldspar (*u*)	Sh. T.	72,656	401,531	92,799	558,944
Flint	Sh. T.	(*u*)66,697	(*u*)243,012	75,561	407,699
Fluorspar	Sh T.	34,683	201,481	36,350	202,736
Fuller's earth	Sh, T.	28,000	237,950	34,039	323,275
Garnet	Sh. T.	5,404	179,548	6,723	209,895
Graphite, amorphous (*u*)	Sh. T.	16,853	102,175	26,962	138,381
Graphite, crystalline	℔.	4,894,483	170,866	4,586,149	149,548
Gypsum (*u*)	Sh. T.	1,540,585	3,837,975	1,751,748	4,942,264
Iron ore	Lg. T.	49,237,129	107,091,574	52,955,070	117,560,255
Limestone flux	Lg. T.	15,486,139	7,339,125	15,722,801	7,480,121
Magnesite	Sh. T.	(*e*)4,000	240,000	....	....
Manganese ore (*u*) (*d*)	Lg. T.	6,921	88,132	5,604	63,369
Mica, sheet (*u*)	℔.	1,423,100	252,248	1,060,182	349,311
Mica, scrap (*u*)	Sh. T.	1,489	22,742	3,025	42,800
Monazite (*u*)	℔.	846,175	152,312	547,948	65,754
Petroleum, crude	Bbl. (*i*)	131,771,505	80,277,279	164,347,930	123,260,948
Phosphate rock	Lg. T.	2,052,742	8,464,535	2,251,459	10,450,522
Pumice (*u*)	Sh. T.	12,200	16,750	8,112	33,818
Pyrites	Lg. T.	225,045	767,866	261,871	851,346
Quartz, crystalline (*u*)	Sh. T.	24,082	121,671	22,977	157,094
Salt (*u*)	Bbl. (*k*)	28,172,380	6,658,350	29,704,128	7,439,551
Sand, glass	Sh. T.	1,089,430	1,208,788	1,187,296	1,250,067
Slate, roofing (*u*)	Squares (*f*)	....	5,668,346	1,277,554	4,817,769
Sulphur	Lg. T.	294,000	6,247,500	307,806	6,427,025
Talc, common (*u*)	Sh. T.	58,972	874,356	....	....
Talc, fibrous	Sh. T.	64,200	541,600	59,000	501,500
Tin ore	Sh. T.	10	3,044	63	15,209
Tungsten, ore	Sh. T.	1,096	442,784	1,468	715,031
Whetstones and oilstones (*u*)	....	....	268,070	....	264,188
Zinc ore	Sh. T.	905,175	17,250,420	902,923	....

Additional details will be found under the respective captions farther on in this volume. (*c*) Includes sulphate made from metallic copper. (*d*) Does not include manganiferous iron ore. (*e*) Estimated. (*f*) One square covers 100 square feet. (*g*) Barrels of 265 ℔. (*h*) Barrels of 380 ℔. (*i*) Barrels of 42 gallons. (*k*) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 ℔. (*m*) Includes a small quantity made from spelter. (*o*) Flasks of 75 ℔. (*q*) Includes spiegeleisen, although the value is given as for ferromanganese. (*s*) Includes nickel from Canadian ores smelted in the United States. (*t*) Barrels of 330 ℔. (*u*) Figures reported by the United States Geological Survey. (*v*) Recovered from scrap metal. (*w*) Statistics not collected.



PRICES OF METALS AT NEW YORK IN 1907.
(Plotted from monthly averages.)

PRODUCTION OF SECONDARY MINERALS AND CHEMICALS IN THE UNITED STATES.

Product.	Measures.	1906		1907	
		Quantity.	Value.	Quantity	Value.
Alundum	℔.	4,331,233	$ 303,186	6,751,444	$ 405,086
Ammonium Sulphate	Sh. T.	75,000	4,674,750	89,000	5,511,770
Arsenic	℔.	1,663,000	83,150	2,020,000	101,000
Bromine	℔.	1,229,000	,184,350	1,062,000	138,060
Calcium chloride	Sh. T.	(w)	....	45,000	450,000
Carborundum	℔.	6,225,280	435,770	7,532,670	451,960
Cement, nat. hyd. (u)	Bbl. (g)	3,935,151	2,362,140	2,887,700	1,467,302
Cement, portland (u)	Bbl. (h)	46,610,822	51,240,652	48,785,390	53,992,551
Cement, slag (u)	Bbl. (t)	481,224	412,912	557,252	443,998
Coke	Sh. T.	33,333,039	88,582,079	36,993,622	99,055,150
Copper sulphate (c)	℔.	50,925,932	3,157,408	44,867,650	2,804,228
Copperas	Sh. T.	22,839	228,390	26,771	294,481
Crushed steel	℔.	837,000	58,590	840,000	58,800
Graphite, artificial	℔.	4,868,000	312,764	6,924,000	483,717
Lead, white	Sh. T.	123,640	15,234,297	111,409	12,254,990
Lead, sublime, white	Sh. T.	7,988	798,880	8,700	1,026,600
Lead, red	Sh. T.	13,693	1,874,448	13,370	1,778,717
Lead, orange mineral	Sh. T.	2,927	421,488	815	123,917
Litharge	Sh. T.	13,816	1,890,050	14,769	1,624,553
Mineral wool	Sh. T.	5,357	55,550	9,008	81,769
Zinc-oxide (m)	Sh. T.	77,800	6,257,361	85,390	7,731,100

Additional details will be found under the respective captions farther on in this volume. (c) Includes sulphate made from metallic coppr. (d) Does not include manganiferous iron ore. (e) Estimated. (f) One square covers 100 square feet. (g) Barrels of 265℔. (h) Barrels of 380℔. (i) Barrels of 42 gallons. (k) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 ℔. (m) Includes a small quantity made from spelter. (o) Flasks of 75 ℔. (q) Includes spiegeleisen, although the value is given as for ferromanganese. (s) Includes nickel from Canadian ores smelted in the United States. (t) Barrels of 330℔. (u) Figures reported by the United States Geological Survey. (v) Recovered from scrap metal (w) Statistics not collected

PRODUCTION OF METALS IN THE UNITED STATES.

Products.	Measures.	1906		1907	
		Quantity.	Value.	Quantity.	Value.
Aluminum	℔.	14,350,000	5,166,000	26,000,000	$ 10,920,000
Antimony	℔.	5,916,000	1,283,772	5,794,000	859,830
Copper	℔.	917,620,000	180,000,339	879,241,766	181,660,141
Ferromanganese (q)	Lg. T.	305,642	16,810,310	339,348	21,887,946
Gold (fine)	Troy oz.	4,565,333	94,373,800	4,314,742	89,191,726
Iron (pig)	Lg. T.	25,001,549	480,279,756	25,442,013	580,077,896
Lead	Sh. T.	345,529	39,093,151	350,130	37,288,845
Nickel (s)	Sh. T.	7,150	6,360,640	8,750	7,875,000
Platinum	Troy oz.	(e)400	8,800	....	....
Quicksilver	Flasks (o)	28,293	1,157,184	20,932	780,506
Silver (fine)	Troy oz.	56,517,900	37,748,757	58,850,530	38,445,181
Tin (u)	Sh. T.	(w)	....	(v)1,662	914,404
Zinc	Sh. T.	225,494	27,961,256	249,612	29,763,735

Additional details will be found under the respective captions farther on in this volume. (c) Includes sulphate made from metallic copper. (d) Does not include manganiferous iron ore. (e) Estimated. (f) One square covers 100 square feet. (g) Barrels of 265℔. (h) Barrels of 380℔. (i) Barrels of 42 gallons. (k) Includes salt used in manufacture of alkali; the barrel of salt weighs 280℔. (m) Includes a small quantity made from spelter. (o) Flasks of 75℔. (q) Includes spiegeleisen, although the value is given as for ferromanganese. (s) Includes nickel from Canadian ores smelted in the United States. (t) Barrels of 330℔. (u) Figures reported by the United States Geological Survey. (v) Recovered from scrap metal (w) Statistics not collected.

Summary of the Production.

Aluminum.—The production of aluminum in the United States in 1907 is estimated at 26,000,000 ℔., against 14,350,000 ℔. in 1906. These statistics are estimated upon the basis of known furnace capacity. The average price of aluminum in 1907 was 42c. per ℔., against 36c. per ℔. in 1906.

Alundum.—In 1907 there were 6,751,444 lb. produced, valued at $405,086, against 4,331,233 lb. valued at $303,186 in 1906. This abrasive substance is made only at Niagara Falls, N. Y., by the Norton Emery Wheel Company of Worcester, Mass. It is used in the manufacture of artificial corundum grinding wheels.

Ammonia and Ammonium Sulphate.—The ammonia production during 1907 is estimated at 89,746 short tons as against 74,922 tons produced in 1906. These figures include the production of all forms of ammonia expressed in equivalent sulphate of ammonia.

Antimony.—In 1907 there were produced 210 short tons of antimony ore, valued at $28,432, as compared with 295 short tons valued at $44,250 in 1906. Of antimony metal, 2897 short tons were produced in 1907, as compared with 2958 tons in 1906. These figures include the metal in imported ores, as well as that in hard lead, which latter represents by far the greater proportion of the total production. The average price of ordinary antimony in 1907 was 14.84c. per lb., against 21.73c. per lb., in 1906.

Arsenic.—The production of arsenious acid in 1907 was 2,020,000 lb., valued at $101,000, against 1,663,000 lb., worth $83,150, in 1906. The Washoe smelter at Anaconda, Mont., and the Everett smelter on the Pacific Coast are the chief producers of this substance. During 1907 there was imported 9,922,870 lb. of arsenious acid, valued at $553,440, or 5½c. per lb.; as against 7,639,507 lb., valued at $336,609, or 4½c. per lb. in 1906.

Asbestos.—In 1907 the output of asbestos was 950 short tons, valued at $11,700, against 1695 tons, valued at $20,565, in 1906. The greatest part of domestic demand for asbestos is still supplied by imports from Canada. The imports during 1907 consisted of manufactured asbestos to the value of $200,371, and unmanufactured to the value of $1,104,109, which was an increase of $197,865 over the total imports for 1906.

Barytes.—The output of barytes in 1907 amounted to 65,579 short tons ($251,308), as compared with 63,486 tons ($252,719) in 1906. Missouri continues to be the largest individual producer but Tennessee is now an important source of supply.

Bauxite.—The output of bauxite in the United States in 1907 is estimated at 97,776 long tons ($480,330), as compared with 78,331 long tons ($352,490) in 1906. The productive areas of the United States are confined to Arkansas, Georgia, Alabama and Tennessee. Arkansas still leads in total production.

Bismuth.—The production of metallic bismuth in 1907 amounted to about 10,000 lb. This is the first time that this metal has been reported and the statistics represent the inauguration of a new industry. There was no production of bismuth ore in the United States in 1907.

Borax.—The output of borax in 1906 was 58,173 short tons, worth

$1,182,410, as compared with 46,334 tons, worth $1,019,158, in 1905. Practically all of the crude borax is produced in California, the statistics of which State for 1907 are not yet available.

Bromine.—In 1907 there were produced 1,062,000 ℔. of bromine ($138,060), against 1,229,000 ℔. ($184,350) in 1906. There was a decrease in the production of Michigan and an increase in that of West Virginia. The other producing States are Ohio and Pennsylvania. A large part of the total bromine produced is recovered in the form of bromides.

Cadmium.—The production of cadmium in 1907 amounted to 15,000 ℔., worth $18,750. This is the first production of cadmium in the United States.

Calcium.—In 1907 the production of metallic calcium was 350 ℔., valued at $613. This is the first production of this metal in the United States.

Calcium Chloride.—The production of calcium chloride in 1907 was 45,000 short tons, worth $450,000. No statistics were collected for 1906.

Carborundum.—The output of carborundum in the United States in 1907 was 7,532,670 ℔. ($451,960), against 6,225,280 ℔. ($435,770) in 1906. The only producer of this substance is the Carborundum Company of Niagara Falls, N. Y. The material is used both as an abrasive and as a refractory lining for furnaces.

Cement.—The production of cement of all kinds in the United States in 1907 amounted to 52,230,342 bbl., valued at $55,903,851. This output was divided as follows: Portland, 48,785,390 bbl. ($53,992,551); natural, 2,887,700 bbl. ($1,467,302); slag, 557,252 bbl. ($443,998). For comparison the 1906 production is given: Portland, 46,610,822 bbl. ($51,240,652); natural, 3,935,151 bbl. ($2,362,140); slag, 481,224 bbl. ($412,912). Although the total production of all kinds of cement showed an increase over that of 1906, there was a decrease of more than 1,000,000 bbl. in natural hydraulic cement.

Chrome Ore.—In 1907 the production of chrome ore was 335 long tons, valued at $5620 against 180 tons ($1800) in 1906. California is the only State producing chrome ore, the rest of the requirements of the country being supplied by imports from New Caledonia and Turkey. The material is used as a source of chromium salts and also as a refractory lining for metallurgical furnaces.

Coal.—The total production of coal in the United States for 1907 was 474,564,700 short tons, valued at $662,307,994, or an average of $1.40 per ton. The corresponding output in 1906 was 413,822,402 short tons, valued at $568,024,092 or $1.37 per ton. In 1907 the anthracite production was 86,341,832 short tons ($198,653,218); bituminous, 328,222,868

short tons ($463,654,776). In 1906 the total output was divided into anthracite, 72,209,566 short tons ($166,307,002), and bituminous 341,612,837 short tons ($401,717,090). The production of anthracite coal in Pennsylvania showed a remarkable increase in 1907, reaching the highest point yet recorded in the industry. The production of bituminous coal showed an increase over that of 1906; the increase would have been much more marked had it not been for the set-back received by the industry during the last half of the year.

Coke.—The output of coke in 1907 was 36,993,622 short tons, as against 33,333,039 short tons in 1906. The total values of these outputs are, respectively, $99,055,150 and $88,582,079, giving an average of $2.68 per ton in 1907 and $2.66 per ton in 1906. The increase in production was not great, due to the depression in the iron and steel industries late in the year.

Copper.—The production of copper of domestic origin in 1907 was 879,241,766 lb. as against 917,620,000 lb. in 1906. The total production of the refineries, plus the pig copper exported, was 1,152,747,890 lb. in 1907. The consumption of copper in the United States in 1907 was 537,818,489 lb. The average price of electrolytic copper was 20.004c. per lb., against 19.278c. in 1906.

Copper Sulphate.—The production in 1907 was 40,138,117 lb. against 50,925,932 lb. in 1906. These figures include the product obtained directly from matte, that obtained as a by-product in lead and copper refinery, and that which is produced by the treatment of metallic copper.

Copperas.—In 1907 the output of copperas in the United States was 26,771 short tons, worth $294,481 as compared with 22,839 short tons ($228,390) in 1906. The greatest part of the copperas produced is obtained as a by-product of iron and steel sheet and wire plants. In 1906 the United States Steel Corporation furnished more than 90 per cent. of all the copperas made in this country.

Cryolite.—There is no production of cryolite in the United States, the entire consumption being supplied by imports from Greenland. The imports of 1907 amounted to 1238 long tons, worth $28,902 as against 1505 long tons, worth $29,683 in 1906.

Crushed Steel.—In 1907 the production of crushed steel was 840,000 lb. ($58,800), against 837,000 lb. ($58,590) in 1906. Crushed steel is used as an abrasive. It is produced by only one concern in the United States.

Emery.—The production of emery in the United States in 1907 was 1069 short tons, valued at $12,294, as compared with 2147 tons ($22,780) in 1906. Practically all of the emery is produced in the region around Peekskill, N. Y., with a smaller amount from Chester, Mass. During 1907 the United States imported 4,282,228 lb. of corundum and emery grains,

valued at $186,156; 11,235 long tons of ore and rock, valued at $211,184; and manufactures to the value of $15,282.

Feldspar.—The United States in 1907 produced 92,799 short tons of feldspar worth $558,944, against 72,656 tons ($401,531) in 1906. Pennsylvania, Maryland, Connecticut, Maine and New York are the chief producing States.

Ferromanganese and Spiegeleisen.—The output of ferromanganese and spiegeleisen in 1907 was 339,348 long tons against 305,642 in 1906. The total values of these products for 1907 and 1906 respectively, are $21,887,946 and $16,810,310.

Fluorspar.—The output of fluorspar in 1907 was 36,350 short tons, valued at $202,736, which is to be compared with a production of 34,683 tons, valued at $201,481, in 1906. The average value of the product was $5.59 per ton.

Fuller's Earth.—The 1907 output of fuller's earth was 34,039 short tons, valued at $323,275 against 28,000 short tons, worth $237,950 in 1906. The chief use of fuller's earth is in clarifying oils and fats. The chief producing districts are in Florida. The imports in 1907 were 14,648 short tons, valued at $122,221.

Garnet.—The production of garnet in 1907 amounted to 6723 short tons with a value of $209,895 as compared with 5404 tons, valued at $179,548 in 1906. By far the greatest part of this production comes from New York. The increasing use of artificial abrasives apparently does not greatly restrict the production of garnet.

Gold.—In 1907 the output of the United States was 4,314,742 oz. fine worth $89,191,726, as against 4,565,333 oz. fine ($94,373,800) in 1906. Almost every State reported a diminished production: the only important gain being $5,426,058 in Nevada. Colorado was the largest producer, with 990,398 oz. fine, valued at $20,471,527.

Graphite.—The production of crystalline graphite in the United States during 1907 amounted to 4,586,149 lb., worth $149,548, as against 4,894,483 lb. ($170,866) in 1906. New York continues to be the largest producer of crystalline graphite. The output of artificial graphite in 1907 was 6,924,000 lb. (worth $483,717) against 4,868,000 lb. ($312,764) in 1906. The demand for artificial graphite is growing rapidly, and it will be increased still more in the future as a result of the successful production of soft artificial graphite, which comes into direct competition with natural graphite as a lubricant, and other uses for which ordinary artificial graphite can find no application.

Gypsum.—The output of gypsum in the United States in 1907 was 1,751,748 short tons, valued at $4,942,264, against 1,540,585 tons ($3,837,975) in 1906. The United States Gypsum Company is the largest operator. Of the 1907 production, 232,546 tons were sold crude; 46,851 tons sold

crude, ground, as land plaster; and 1,125,301 tons sold as calcined plaster.

Iron.—The production of pig iron in 1907 amounted to 25,442,013 long tons valued at $580,077,896, against 25,001,549 tons valued at $480,279,756 in 1906. The production of bessemer pig in 1907 showed a slight decrease as compared with 1906. Foundry and forge pig increased slightly, while basic pig show a decrease.

Iron Ore.—In 1907 the United States produced 52,955,070 long tons of iron ore, valued at $117,560,255. The three producing districts contributed their shares as follows: Lake Superior, 42,245,070 tons; Southern States, 7,585,000 tons; other States, chiefly New York and Central States, 3,125,000 tons. The total output in 1906 was 49,237,129 long tons, having a value of $107,091,574.

Lead.—The production of lead of domestic origin in the United States in 1907 amounted to 350,130 ($37,288,845) short tons. The production in 1906 was 345,529 ($39,093,151) tons.

Lead Products.—The domestic output of lead products during 1907 was as follows: White lead, 111,409 short tons ($12,254,990) against 123,640 tons ($15,234,297) in 1906; red lead, 13,370 short tons ($1,778,717) against 13,693 tons ($1,874,448) in 1906; litharge, 14,769 short tons ($1,624,553), against 13,816 tons ($1,890,050) in 1906; orange mineral, 815 short tons ($123,917) against 2927 tons ($421,488) in 1906; sublimed white lead, 8700 short tons ($1,026,600) against 7988 tons ($798,880) in 1906.

Limestone.—The consumption of limestone in 1907 for metallurgical purposes was 15,722,801 short tons, having a value of $7,480,121, which is to be compared with a consumption in 1906 amounting to 15,486,139 short tons, valued at $7,339,125. Iron smelting demanded the largest percentage of the above amounts.

Lithia.—The 1907 output of lithia minerals is not yet available. The production in 1906 was 383 tons worth $7411 as compared with 21 tons worth $252, in 1905. The industry suffered during the latter part of 1907 by reason of the drop in prices of lithium salts.

Mica.—In 1907 the output of sheet mica amounted to 1,060,182 lb. ($349,311), against 1,423,100 lb. ($252,248) produced in 1906. The output of scrap mica in 1907 was 3025 short tons valued at $42,800 as compared with 1489 tons produced in 1906 and worth $22,742. The domestic production of mica is but a small part of the consumption and considerable quantities are imported from Canada and India.

Mineral Wool.—The production of mineral wool in 1907 was 9008 short tons, worth $81,769, as compared with 5357 tons, worth $55,550 in 1906. The chief producers are the U. S. Mineral Wool Company of New York and the Columbia Mineral Wool Company of Chicago.

Monazite.—The output of monazite in 1907 was 547,948 ℔., valued at $65,754, against 846,175 ℔., valued at $152,312, in 1906. The decreased production in 1907 was due chiefly to the fall in the price of thorium nitrate, the financial depression, and the exhaustion of some of the richer deposits of Brazil. The chief production continues to come from the Carolinas.

Nickel.—The output of metallic nickel, almost entirely obtained from Canada, was 8750 short tons, valued at $7,875,000, in 1907, against 7150 tons valued at $6,360,640, in 1906.

Petroleum.—The production of petroleum in 1907 was 164,347,930 bbl. (of 42 gal.), valued at $123,260,948, while the 1906 production was 131,771,505 bbl., valued at $80,277,279. A decline in production was again recorded in the Appalachian field and a rapid decline in the Texas and Louisiana and Lima fields, but the Mid-continental and Illinois fields showed remarkable increases in production over 1906.

Phosphate Rock.—The 1907 output of phosphate rock was 2,251,459 long tons, valued at $10,450,522, as against 2,052,742 tons, valued at $8,464,535, in 1906. A feature of the industry in 1907 was the development, at several points, of a new field covering parts of Idaho, southwestern Wyoming and northeastern Utah.

Platinum.—The figures for the 1907 production are not yet available, but there was probably a small increase over 1906. In 1906 the production was 400 oz. troy. The increase results from awakened attention to the black sand concentrates from Pacific coast placer workings. Imports from Russia supply the major part of the domestic demand.

Potassium Salts.—Domestic consumption is supplied entirely by imports which in 1907 amounted to 364,053,858 ℔. valued at $7,011,198, as against 331,220,320 ℔., valued at $6,139,597, in 1906.

Pyrite.—The production of pyrite was 261,871 long tons in 1907 and 225,045 tons in 1906. These amounts are valued, respectively, at $851,346 and $767,866. The chief producing State is Virginia, but the production in Alabama is becoming of increasing importance.

Quicksilver.—The 1907 output of quicksilver was 20,932 flasks of 75 ℔., worth $780,506, as compared with 28,293 flasks, worth $1,157,184, in 1906. These values are calculated on the basis of the average value of quicksilver at New York for the years in question. California is still the chief source of this metal, but small amounts are produced in Texas and Utah.

Salt.—In 1907 there were produced 29,704,128 bbl. of salt of 280 ℔. each, valued at $7,439,551. The output in 1906 was 28,172,380 bbl., valued at $6,658,350. Michigan and New York together produce approximately two-thirds of the total output.

Silver.—The production of silver in 1907 was 58,850,530 oz. fine, worth

$38,445,181, while in 1906 the production was 56,517,900 oz. fine, valued at $37,748,757.

Sodium.—The 1907 production of sodium amounted to 2000 short tons worth $1,000,000.

Sulphur.—The production of sulphur during 1907 is estimated at 307,806 long tons, valued at $6,427,025, against 294,000 tons, valued at $6,247,500, in 1906. Louisiana is the chief source of the domestic sulphur in the United States, the small amount which is produced in the West being consumed locally.

Talc and Soapstone.—The production of fibrous talc in 1907 amounted to 59,000 short tons, valued at $501,500, against 64,200 tons ($541,600) in 1906. New York is the principal source of the fibrous talc produced in this country.

Tungsten Ore.—The production of tungsten ore in 1907 was 1468 short tons, valued at $715,031, as compared with 1096 tons ($442,784), in 1906. Boulder county, Colo., supplies over 85 per cent. of the domestic production in the United States.

Zinc.—The output of virgin spelter in 1907 was 249,612 short tons, against 225,494 tons in 1906. During 1907 there was another large increase of smelting capacity in the United States, which now appears to be in excess of the immediate requirements. The average price of spelter at New York was 5.962c. per lb. in 1907, against 6.198c. per lb. in 1906.

Zinc Ore.—The output of zinc ore amounted to 902,923 short tons in 1907, as compared with 905,175 tons in 1906.

Zinc Oxide.—In 1907 the production of zinc oxide, including zinc-lead pigments, was 85,390 short tons, valued at $7,731,100, as compared with 77,800 tons, valued at $6,257,361, in 1906. There is a small production of zinc oxide by the combustion of spelter, but the great bulk of the product is made directly from ore.

ALUMINUM[1].

By Joseph W. Richards.

The capacity of the plants of the Aluminum Company of America was greatly increased in 1907. In the early part of the year the works at Niagara Falls, N. Y., had 12,000 h.p., the works at Massena, N. Y., 12,000, and at Shawinegan Falls, Canada, 5000, a total of 29,000 h.p. At the end of the year Niagara Falls had 40,000, Massena, 20,000, and Shawinegan Falls, 15,000, giving a total of 75,000. The business depression of the close of 1907 affected the aluminium industry as it did all others, and it is best to assume that the year's output did not average over 75 per cent. of the total capacity of the existing plants. With 60,000 h.p. in the United States and 15,000 in Canada, this would correspond to an output of 13,000 tons in the United States and 5000 tons in Canada, a total of 18,000 tons of 2000 ℔. Since the company is unwilling to give the exact figures of its output, approximations such as the above based on known power available and its possible utilization, are the only resource for statistics, unsatisfactory as they may appear.

PRODUCTION, IMPORTS AND CONSUMPTION OF ALUMINIUM IN THE UNITED STATES.

Year.	Production.			Imports.			Exports.	Consumption.
				Crude.		Mfrs.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Value.	Value.	Value.
1897	4,000,000	$1,400,000	$0.35	1,822	$1,082	$3,647	(a)	$1,404,729
1898	5,200,000	1,690,000	0.33	60	30	13,840	$238,997	1,474,268
1899	6,500,000	2,112,500	0.33	53,622	9,425	7,828	291,515	1,838,238
1900	7,150,000	2,288,000	0.32	256,559	44,455	5,989	281,821	2,056,623
1901	7,150,000	2,238,000	0.31	564,803	104,168	5,580	183,579	2,164,169
1902	7,300,000	2,284,590	0.31	745,217	215,032	3,819	116,052	2,387,389
1903	7,500,000	2,325,000	0.31	498,655	139,298	4,273	157,187	2,311,384
1904	7,700,000	2,233,000	0.29	515,416	128,350	478	166,876	2,494,952
1905	11,350,000	3,632,000	0.32	530,429	106,108	33	290,777	3,015,364
1906	14,350,000	5,166,000	0.36	770,713	154,292	1,866	364,251	4,957,907
1907	26,000,000	10,920,000	0.42	872,474	181,351	1,124	304,938	(b)

(a) Not reported. (b) Impossible to compute accurately in the absence of information as to unsold stocks.

Production.

In the early part of 1907, and during the summer months, the plants of the Aluminum Company of America were making use of every horse-power available. The demands for aluminium were very heavy. Toward the end

[1] In deference to the well known desire of Prof. Richards, we have retained in this article (except in the main caption) his preferred spelling "aluminium," although it is contrary to our style and the usual American form. "Aluminum" is uniformly employed elsewhere in this volume.—Editor.

of the year the demand fell off, owing to the general industrial depression, and particularly because of the relaxation in the demand from the automobile manufacturing and electrical industries. Approximately the same conditions existed in Europe. The general set-back was so serious that the Aluminum Company of America in November shut down temporarily its Shawinegan and Massena plants and half of its Niagara plant. Of course, it is morally certain that this set-back to aluminium manufacture is only temporary, and when industrial conditions improve the production of aluminium will proceed with former vigor. In connection with the general status of the industry, we may best reproduce portions of an

WORLD'S PRODUCTION OF ALUMINIUM.
(In metric tons.)

Year.	Great Britain.	France.	Switzerland. Germany. Austria.	North America.	Totals.
1897....	(a)310	470	800	1,815	3,195
1898....	310	565	810	2,359	4,034
1899....	559	763	1,300	2,949	5,571
1900....	569	1,026	2,500	3,244	7,339
1901....	560	1,200	2,500	3,244	7,504
1902....	600	1,355	2,500	3,312	7,767
1903....	(b)650	1,570	(b)2,500	3,403	8,123
1904....	(b)650	1,650	(b)3,000	3,494	8,794
1905....	2,250	4,425	3,675	6,560	16,810
1906....	2,500	4,500	4,000	7,325	18,325
1907....	3,700	4,500	8,000	16,329	32,529

(a) C. L. Neve Foster, British Mineral Statistics for 1897.
(b) Statistics of Metallgesellschaft, Frankfurt am Main.

editorial which appeared in the *Eng. and Min. Journ.* of Nov. 30, 1907, as follows: "The production of aluminium is forging rapidly ahead and the vision of the prophets that this metal in the comparatively near future will become of common industrial importance is growing clearer and clearer. Several companies are preparing for the aluminium business in the United States, and when the Bradley patents expire in February, 1909, there will be a battle royal between the Aluminum Company of America and its new competitors. The older company will occupy the superior position because of its prestige, experience and large capacity for production, but the price for the metal will inevitably come down. An expert in the aluminium industry, in whom we have great confidence, foresees that aluminium will be produced eventually by the hundreds of thousands of tons yearly and considers that a large figure may be expected in the not very distant future. Indeed, the one hundred thousand ton mark may be passed inside of five years.

"If we consider the statistics of production in 1906 this estimate does not appear unduly extravagant. In that year the production of the United States and Canada was considerably upward of 7000 metric tons; the production of the world was 18,325 metric tons, which was more than twice as great as in 1904. The production has been, indeed, increasing

by leaps and bounds. The Aluminum Company of America undertook the installation of new equipment and plant in 1905 which was only partially completed in 1906. It was pointed out in THE MINERAL INDUSTRY, Vol. XV, however, that on this account the increase in production in 1907 and 1908 will be very marked, and by the end of 1908 the production of aluminium in the United States will make a significant comparison with the production of copper, taking into consideration the relative bulks of the two metals."

It is well known that the producers of aluminium can stand an important reduction in price and still make a fine profit.

United States and Canada.—Arthur V. Davis, general manager of the Aluminum Company of America, reported in September, 1907, that the new reduction plants of the company at Niagara Falls and Shawinegan Falls were completed and in operation. The increase in the company's output enabled it to catch up on orders and prompt deliveries were then being made. While the company will not give out the capacity of its new plants, it is stated that the Shawinegan plant in Canada is now more than three times its size of 1906 and the Niagara Falls plant has been increased to about the same extent, giving a present capacity in excess of the demand for aluminium. The rolling mill capacity at New Kensington, Penn., has been increased, so that all the sheet demand can be taken care of there, though within a few months a new rolling mill at Niagara Falls will be completed, having about two-thirds the capacity of the New Kensington mill. In the plant at New Kensington, there has been installed a continuous mill for rolling aluminium sheets, which is said to be the only one of the kind ever built. The Aluminum Company of America also improved its bauxite refining plant at East St. Louis, Ill.

MARKET CONDITIONS.

The price of aluminium in the United States at the beginning of 1907 was 36c. per ℔. In April it rose to 50c. per ℔. During the autumn it declined along with the other metals and closed the year at 38c. per ℔. Early in January, 1908, it was reduced to 33c.

The price of aluminium in Europe, which in 1905 rose to 3s. 3d.@3s. 9d. per kg., according to delivery, and remained at that level during 1906, commenced to decline in the early part of 1907 and by midsummer stood at 3s. per kg., which continued nominally during the remainder of the year, with no difference between spot and future deliveries, spot having previously commanded a premium. In the summer there was considerable talk respecting a further reduction, which was at first denied, but later was admitted, and a new price—2s. per kg.—came into effect Jan. 1, 1908, but as early as Oct. 1 large sales were made at 2.75s. According to the price list of the British Aluminum Company issued late in 1907, aluminium

ingot, 98-99 per cent. pure, was offered at £1 25s. per lb. in ton lots, or over, less 2.5 per cent. discount for cash in the month following delivery. The metal bought under these terms can be resold only in the British Isles, India, or the South African colonies. Along with the reduction in price in Europe it is reported that the understanding among the principal producers as to the regulation of price was extended for five years, but recent developments seem to negative this rumor of a secret understanding, since European aluminium is now being freely offered on the American market at prices below the domestic quotations.

PRICE OF ALUMINIUM AT NEW YORK.
(In cents per pound.)

Grade.	Dec., 1905.	July, 1906.	Dec., 1906.	July, 1907.	Dec., 1907
99% pure	35	36	38	42	33
90% pure	33	34	37	41	32
No. 12 casting alloy	35	36	37	41	32
No. 21 casting alloy	33	34	35¼	39¼	30¼
No. 31 casting alloy	30	31	33¼	37¼	28¼

The above prices were for ton lots or over; the prices for small lots were 2 to 3c. per lb. higher.

The Aluminium Industry in Europe.

The European producers of aluminium in 1907 were still only four in number, but there will be many additions in 1908. Their production in 1906 and 1907 is estimated as follows:

Name	1906		1907	
	Metric tons	Pounds.	Metric tons	Pounds.
British Aluminium Company, Scotland	2,500	5,500,000	3,700	8,150,000
Société Electro-Metallurgique Francaise	2,500	5,500,000	2,500	5,500,000
Aluminium Industrie Aktien Gesellschaft	4,000	8,800,000	8,000	17,600,000
Société des Produits Chimiques d'Alais	2,000	4,400,000	2,000	4,400,000
Total	11,000	24,200,000	16,200	35,650,000

At the last meeting of the stockholders of the British Aluminium Company, John D. Bonner remarked that inasmuch as all the aluminium producers on the Continent and in America were enormously increasing the amount of power at their disposal, the then existing famine in aluminium should soon be succeeded by a plethora of the metal. Such a change in the situation would undoubtedly lead to a material modification of the present price. Meantime, the company was enjoying a period of great prosperity. Later on it would probably have to be content with a much smaller percentage of profit, though even then it should be able to obtain satisfactory results for the shareholders provided there should be a sufficient expansion in the demand for the metal to enable it to sell, even at a lower level of

price than had ever yet ruled, the comparatively speaking huge output it would be able to command.

The *Engineer* (London) of May 17, 1907, gave the following list of the companies and works producing aluminium at that date, with the power controlled by each works:

British Aluminium Company.—1. Foyers, N. B., 5000 h.p. 2. Sarpsfos, Norway, 10,000. Extensions and new works in progress:—Loch Leven, N. B., Conway Valley, N. Wales, and Orsieres in Switzerland; combined schemes equal to 60,000 h.p.

Société Electrometallurgique Française.—3. La Praz, France, 7500 h.p. 4. Gardannes, France, 7500.

Compagnie des Produits Chimique d'Alais.—5. Calypso, France, 10,000 h.p. 6. St. Felix, France, 2500.

Aluminiumindstrie Aktiengesellschaft. — 7. Neuhausen, Switzerland, 5000 h.p. 8. Rheinfelden, Germany, 5000. 9. Lend Gastein, Austria, 15,000. Extensions and new works in progress: — On the Rhone and Navigence in Switzerland; combined schemes equal 50,000. Total, 75,000.

Aluminum Company of America.—10-11. Niagara Falls (two works) 10,000 h.p. 12. Shawinegan Falls, Canada, 5000. 13. Massena, U. S. A., 12,000. (This company is greatly underestimated, its consumption of power in 1907 having averaged 50,000 h.p.)

One Italian Company.—14. Pescara, North Italy, 3000 h.p.

The aggregate controlled by these six companies and 14 works is 97,500 h.p. Assuming that 4 h.p. years are required to produce 2240 ℔. of aluminium, this total power would suffice to produce 24,000 long tons of the metal per annum. This output has never yet been attained, however, for two reasons. In the first place, the reduction plant in many of the works is not equal in capacity to the generating plant, for until the sudden expansion in the use of the metal in 1906, the demand for aluminium did not warrant such an increase in producing power.

In the second place, the maximum power development of the water power controlled by the aluminium companies is only attained during the winter and spring months of the year, and for a period extending from one-third to one-half of the year the output of electrical energy is far below that indicated by the above figures. These two causes have tended greatly to reduce the output of aluminium by the various producing companies in the past.

Since the above enumeration, which may be considered as referring to the beginning of 1907, was made there have been many other new plants installed or projected, which are mentioned in the following paragraphs.

Belgium.—The Société Belge-Néderlandaise d'Aluminium, which has

works at Selzaete, states, that it is the only one in Belgium which is specially equipped for the production on advantageous terms of pure oxide of aluminium, which is of primary necessity in the output of this metal, and that it is very favorably situated from the exportation point of view. As a result of the development of the trade, the company has decided to increase its production by at least 100 per cent., and the orders already placed for this purpose led to the assumption that the additional plant will be brought into use during 1907.

France.—The Société Froges d'Aluminium is preparing to develop 40,000 h.p. on the Durance at Largentière. The Société Péchiney d'Aluminium started half of the St. Jean-de-Maurienne works, utilizing 12,000 h.p., in September. When completed this will be a works capable of producing 4000 tons of aluminium per year. The Gardanne and Salandres works have been equipped to supply alumina for the Saussaz Calypso and St. Jean works. The Société d'Electrochimie enlarged its Prémont works to 4000 h.p. The alumina is supplied by the Usine de la Barasse, near Marseilles, The Société des Forces Motrices de l'Arve began to produce aluminium at Chedde. A branch of this company viz., the Société des Produits Electrochimiques et Métallurgiques des Pyrénées, is constructing a works of 12,000 h.p. at Auzat, near Vicdessos (Hautes Pyrénées). Early in 1908 about 4000 h.p. will be applied for aluminium; the Bédarieux bauxites will supply the alumina. The Société l'Aluminium du Sud-Est is installing works on the river Neste, at Arreau (Hautes Pyrénées). Its branch establishment, Société Electrométallurgique du Sud-Est, commenced production on April 1, 1907. The Société Anonyme Metallurgique d'Aluminium (Peniakoff processes) is still negotiating for manufacture of the metal.

Italy.—Early in 1907 an aluminium works was completed near Bussi in the Abruzzi, by the Roman company which for some years past has owned a 4000 h.p. factory in the same neighborhood for the manufacture of caustic soda, bleaching powder, etc. A new station has been built about four miles distant, and is developing power from the rivers Tirino and Pescara to the amount of 12,000 h.p. From this station current is supplied to light and serve the towns of Aquila, Chieti, and all the other places in the vicinity, and also to certain works where copper sulphate, artificial fertilizers, and nitrogen compounds are being made. The aluminium factory is to obtain some of its bauxite from the mines at Lecce dei Marsi, in the province of Aquila, but the greater part will be procured from Japan.

The Societá Italiana di Aluminium completed a works at Popoli. The Elba company is reported to be considering the production of aluminium at its Portoferraio works.

Norway.—The Strangfjord works (British Aluminum Company) was

completed toward the end of 1907. Another new plant (2000 h.p.) was commenced near Kristiansand by the Anglo-Norwegian Aluminum Company, which will utilize the Vigeland falls on the Otterdal.

Switzerland.—The Aluminium Industrie Aktiengesellschaft, which already controls 25,000 h.p. in the three works at Neuhausen, Rheinfelden and Lend Gastein, is carrying out two large schemes of water power development in Switzerland, by which from 50,000 to 75,000 h.p. will be obtained from the waters of the Navigence and the Rhône. The Navigence will give 18,000 h.p. from a head of 60 m., and the Rhône 18,000 h.p. from a head of 100 m. The former requires a tunnel 8000 m. long; the latter one of 5000 m. A metallurgical works is being built at Chippis, Valais, at the confluence of the Rhône and Navigence. Construction was well advanced at the end of 1907.

United Kingdom.—According to the official report of the British Aluminum Company, the profit in 1906 was £155,024. Progress was made with the development of the Loch Leven power scheme. Under the contract with Sir John Jackson, Ltd., the hydraulic portion of the work has to be finished by Aug. 15, 1908, by which date, under contracts made with several firms, all the generating plant should be installed in the power house. By that date also the aluminium factory is expected to be erected and equipped. With a view to obtaining an increased production of aluminium prior to the completion of the Leven works, the company acquired a partially developed power at Stangfjord, in Norway, and took steps to develop some 3000 h.p. at Leven, which latter can be utilized without interfering with the progress of the main scheme. The board also acquired the concession for a water power of considerable magnitude at Orsières, in Switzerland, and intends to proceed with the construction of the works without delay. Already the construction of a railway line to connect Orsières with the main line of the Swiss Federal Railways at Martigny has been commenced. To meet these outlays, the share capital of the company has been increased from £700,000 to £1,300,000.

In 1895, on the acquisition of this company from the Société Anonyme pour l'Industrie de l'Aluminium de Neuhausen of the English patents for the manufacture of aluminium, the latter company undertook to furnish to this company detailed information as to the process then used at its Neuhausen works, and this company, in return, undertook only to use such information in works in the United Kingdom or in the colonies or dependencies of the United Kingdom. The Neuhausen company recently contended that the British Aluminum Company could not manufacture aluminium abroad without infringing that agreement. As the British company was about to enter upon the manufacture of aluminium upon the Continent, an agreement was come to with the

Neuhausen company to end all outstanding provisions of the 1895 contract upon the British company paying to the Neuhausen company the sum of £100, and engaging for a maximum period of 10 years from Jan. 1, 1907, not to use or permit to be used any part of the Orsières water power for the production of aluminium.

Two new English companies of 1907 are the Anglo-Norwegian Aluminum Company (capital, £110,000) and the Aluminium Corporation, Ltd. (capital, £500,000). The former has erected works in Norway, and the latter in Great Britain. Both were expected to be in operation early in 1908, and the Aluminium Company expected to have 12,000 h.p. available in 1909.

Notes on the Metallurgy of Aluminium.

Chemical Compounds.

Double Sulphides.—When aluminium sulphide and manganous sulphide are heated together in the electric furnace, compound sulphides similar to aluminates are formed, which may be called sulph-aluminates (Houdard, *Compt. Rend.*, CXLIV, 1907, p. 801). Manganese sulphide forms $MnAl_2 S_4$, and iron sulphide $FeAl_2S_4$. These compounds form poorly shaped crystals having no action on polarized light. $CrAl_2S_4$ is obtained in a crystalline condition by heating a mixture of Al_2S_3 and CrS in an atmosphere of hydrogen.

Nitride.—By the action of dry ammonia on aluminium dust, A. H. White and L. Kirschbraun (*Journ. Am. Chem. Soc.*, XXVIII, 1906, p. 1343) find that the aluminium takes up as much as 1.8 per cent. of nitrogen, or 36 times its volume. The most favorable temperature for this absorption is 700 deg. C.

Aluminates.—Z. Weyberg, of the University of Warsaw, (*Centralblatt für Mineralogie und Geologie*, XX, 1906, p. 645) describes the formation of lithium aluminate, $Li_2Al_2O_4$, by fusing together a mixture of spodumene (Li_2O, Al_2O_3, $2SiO_2$) with 10 parts of Li_2SO_4 and leaching the melt with water. There remains a snow-white sandy powder of the aluminate, insoluble in water but slowly decomposed by dilute acids. The particles were fused and showed strong double refraction, which would put them out of the spinel class of ordinary aluminates.

Physical Properties.

Tensile Strength.— F. G. A. Wilm, in *Brass World*, November, 1907, states that aluminium hardened by copper can be considerably strengthened by heating at least to a critical temperature, which he calls the recalescence point, and immediately chilling in cold water. With 1 per cent. copper, the required temperature is 455 deg. C.; with 2 per cent., 471 deg.; 3 per cent., 482 deg.; 4 per cent., 494 deg.; 6 per cent., 500 deg.;

10 per cent., 505 deg.; and 15 per cent., 508 deg. The alloy is heated anywhere between the above temperatures and its melting point before being chilled. The 4 per cent. copper alloy, cast in a chill mold, and then subjected to this treatment, had its tensile strength increased 48 per cent., elongation 2 per cent. With sand castings, wire, sheet and rod, similar beneficial results are claimed to be produced.

Alloys.

A New Bronze.—W. Ruêbêl, of Hamburg, Germany, (*Metal Industry*, March, 1908) patented Aug. 20, 1907, a bronze consisting of 39 per cent. copper; 34 per cent. iron; 18 per cent. nickel; 9 per cent. aluminium. The proportions correspond to the molecular formula Cu_2Fe_2NiAl. In making this new alloy, the wrought iron or low carbon steel is put in the melting pot with the nickel. These are brought to whiteness, and then one-tenth of the aluminium is added. When this has incorporated, the copper is added, and then the rest of the aluminium. The whole is stirred with a carbon rod, left in the fire 10 minutes longer, and then cast. The alloy is stated to be as hard as nickel steel, stronger, and very resistant to sea water, moist air and acid waters. Six parts of this alloy added to 94 parts of yellow brass is stated to make the brass equal in strength to the best bronzes.

Zinc-Aluminium.—The Syracuse Aluminum and Bronze Company makes automobile castings of aluminium and zinc, the strongest being 83.42 per cent. aluminium, 16.24 per cent. zinc, and 0.34 per cent. copper. Endurance tests of these alloys, made by the Henry Souther Engineering Company, are reported in *Metal Industry*, January, 1908, showing up to 10,000,000 alternations of fiber stresses of 10,000 lb. per sq.in. necessary to fracture the metal.

With Copper.—H. C. H. Carpenter and C. A. Edwards have made an extensive report on these alloys to the Alloys Research Committee of the British Institution of Mechanical Engineers. Their conclusions are as follows: (1) There is no need of remelting alloys in order to obviate brittleness by securing thorough mixing. With due care to get the proper casting temperature and avoid gas absorption by the metal, the first cast may be made as good as any succeeding cast. [This conclusion is so different from ordinary practice, that we must assume that the investigators, not being practical foundrymen, were not able to regulate the conditions of succeeding melts so as to improve on their first melt.] (2) The alloys can be prepared directly from the metals quite as satisfactorily as by making the 50 per cent. alloy first and then diluting it down. (3) Up to 7.35 per cent. aluminium the bronzes have moderate strength and great ductility; from 8 to 10 per cent. great strength and also fair ductility. Specimens of the bronzes were tested

in sea water, along with Muntz metal and Naval brass. When the water was uncontaminated by sewage, the aluminium bronzes were superior to the other metals; in contaminated water the results were various. In fresh water, the aluminium bronzes were more easily attacked than the others.

With Copper, Tin and Antimony.—F. W. Fletcher (U. S. Pat., 867,194, Sept. 24, 1907) describes a hardened aluminium suitable for horseshoes, containing 30 parts of aluminium, 0.92 part copper, 0.31 part tin and 0.02 part antimony.

Solders.

C. Ellis has patented the following solder for aluminium: tin 30 parts, zinc 7, aluminium 0.75, manganese 0.10. (U. S. Pat., 863,058, Aug. 13, 1907). The ingredients should be melted together in a tightly closed crucible, as air seems to injure the alloy. (*Metal Industry,* October, 1907.)

Aluminothermics.

Preparation of Chromium.—Emile Vigoroux (*Bull. Soc. Chim.*, IV, 1m., 1907, 10) mixes 270 grams of powdered aluminium, 600 grams of Cr_2O_3 and 120 grams of CrO_3, and ignites in a magnesia-lined crucible. The reaction lasts about one minute, and the chromium obtained is 98.5 per cent. pure, the impurities being silicon, iron and aluminium.

Preparation of Manganese.—Dr. Hans Goldschmidt and O. Weil find that when Mn_3O_4 is reduced by aluminium the reaction is slow and unsatisfactory, but that it may be quickened and rendered more efficient by mixing a relatively small quantity of a higher oxide with the Mn_3O_4. A suitable mixture (given in their U. S. Patent, 860,799, of July 23, 1907) is 73 kg. of Mn_3O_4, 5 kg. of MnO_2, both powdered, and 22 kg. of finely powdered or granulated aluminium, mixed and ignited in the usual manner.

Preparation of Boron.—K. A. Kuehne, in U. S. Patent 861,129 of July 23, 1907, prepares metallic boron by using two parts of finely divided aluminium, five parts $KClO_3$ and three parts of a boron salt. The mixture is put into a crucible and ignited, melting together to a thin liquid white hot. The excess of unused aluminium remaining contains crystals of boron, which are obtained by dissolving away the aluminium by dilute acid.

F. E. Weston and H. R. Ellis also examined the reduction of powdered B_2O_3 by aluminium powder (*Trans.* of the Faraday Society, 1907). The product was Al_2O_3 and mixtures of boron with aluminium borides.

Reduction of Silica.—F. E. Weston and H. R. Ellis studied the action of powdered aluminium on infusorial earth, precipitated silica and fine, white sand. The reaction could be started cold, using a fuse of Mg ribbon and BaO_2; when sand was used, however, the mixture had to be heated

to low redness to get the reaction to start. The products of the reaction were in each case Al_2O_3, Si and some aluminium silicides.

Chemical Analysis.

Separation from Beryllium.—B. Glassman (*Berichte der deutsche Chem. Gesell.*, XXXIX, 1906, p. 3366) states that if sodium thiosulphate solution is added to a slightly acid solution of beryllium and aluminium salts, the aluminium is completely precipitated on boiling, as hydroxide, while the beryllium remains in solution. Tests showed an error of less than 0.05 per cent. in the results.

Determination in Sulphate Solution.—Having aluminium, iron or zinc present as sulphates, S. E. Moody (*Am. Journ. of Science*, XXII, 1906, p. 483) proposes to add potassium iodide and iodate and boil. The sulphates undergo hydrolysis, iodine is liberated in proportion to the amount of sulphuric acid thus formed, and is determined. In a parallel boiling process the weight of total oxides obtained is determined, the ferrous and ferric oxides by permanganate titration, zinc by the electrolytic method, and then the total alumina and basic alumina present may be calculated.

Manufactures.

Bimetallic tubing is being made now on a considerable scale for condenser tubing. Aluminium and copper, or aluminium and steel, are drawn together, so that either metal is inside or outside the other. There seems to be developing a field for condenser tubes where, for instance, the water is impregnated with sulphuric acid, and the aluminium lining protects the copper and is perhaps equally good as a heat conductor.

Bimetallic cable is an aluminium cable for electrical conductors which has six strands of aluminium around a center strand of steel. When properly applied and exactly of the right size, the six outside strands, on account of their softness, hug close together and form an impervious metallic shield around the steel core, so that neither corrosion from atmospheric agencies nor from galvanic action takes place. By using a steel core of 100,000 ℔. tensile strength, the resulting cable is superior in strength to anything which can be made of copper or aluminium alone.

Extruded metal is aluminium extruded through a die, like lead pipe, at a temperature somewhat below the melting point. Extruded shapes of various cross sections are now being placed on the market, such as moldings, etc., and main electrical conductors, or bus-bars, of any desired length can be extruded free from all joints. Bus-bars of any desired cross section and of any desired length can now be obtained, resulting in considerable advantage to all users of heavy currents.

ALUNDUM.

As in previous years, the Norton Emery Wheel Company, of Worcester, Mass., was the sole producer of alundum. There is no foreign production, the Norton company controlling the patents on the process and on the furnace used. Its works are at Niagara Falls, N. Y. The quantity and value of this abrasive produced since 1904, when its manufacture was begun, are as follows:

PRODUCTION OF ALUNDUM IN THE UNITED STATES.

Year.	Pounds.	Value.
1904	4,020,000	$281,400
1905	3,612,000	252,840
1906	4,331,233	303,186
1907	6,751,444	405,086

Alundum is a product of the electric furnace, prepared from the mineral bauxite. It is made as follows: The bauxite is first calcined to drive off the combined water and is then melted in a specially designed electric furnace, at a temperature ranging from 5000 to 6000 deg. F. It is important that the molten charge be kept at a certain consistency, and as the temperature is at all times under the control of the furnace tender this end is easily attained. Following treatment in the furnace the charge is poured into ingot molds, cooled and passed through a powerful crusher. After further crushing with rolls, followed by washing, drying and careful sizing, the product is ready for manufacture into grinding wheels, rubbing and sharpening tools, etc.

Alundum, while corresponding in chemical composition to corundum, is harder than that mineral. This fact is due to the state of fluidity to which the bath is brought in the furnace and the rate at which it is cooled. Alundum powder is used for cutting and drilling rubies and sapphires for watch jewels, etc.

AMMONIA AND AMMONIUM SULPHATE.

By C. G. Atwater.

The ammonia production of the United States is derived almost exclusively as a secondary or incidental product of three industries, viz., coke making in the by-product oven, gas making in the coal-gas retort, and the manufacture of bone-black. Certain other industries produce small quantities. There are in operation at least two installations of by-product gas producers, akin to the Mond producers used in England, in which ammonia is recovered, but the amount is not statistically important.

Of the industries named above the by-product coke oven is now by far the largest producer, and seems destined to continue in this position for some time to come. The need of ammonia in this country, indeed, has a great deal to do with the introduction of the by-product oven here, and the apparent ease with which the home market has absorbed the large increment of ammonia offered by the increase in the number of these plants clearly shows that this need was far from being over-estimated.

The coal-gas retort occupies the second place as a producer of ammonia, it having been passed by the by-product oven early in the development of that industry. The potentialities of the gas retort as a producer of ammonia are, however, not to be lightly passed by, as may be seen by referring to the statistics of production in England, given later on. That country, the largest producer of ammonium sulphate in the world, owes its preëminence to the recovery of ammonia from the coal-gas retort, which has yielded the major part of its output for many years. This is due to the widely extended use of coal-gas in England, and to the economies practiced in its manufacture. In the United States, on the other hand, carburetted water gas, which produces no ammonia, is much more largely used, and in the existing coal-gas works the ammonia recovery is often not as large as more careful methods would yield, it being sometimes wasted altogether. The tendency now seems to be toward a larger use of coal-gas, because of its low cost, with a corresponding increase in the amount of ammonia recovered. On the other hand, the production of ammonia from the gas works in England is increasing but slowly, partly due to the introduction of carburetted water gas there, and the production of ammonia from by-product coke ovens is receiving the larger yearly increment.

The third source of ammonia in the United States is the carbonization of bones to produce bone-black. As far as can be learned the quantity

of bone-black annually produced is about stationary, if indeed it has not fallen off; hence the amount of ammonia thus produced, never very large, is not increasing.

It should be stated here that the scope of this article does not extend to the various products of the abattoir and fertilizer industries known in the trade as "ammoniates," but is limited to the processes in which ammonia is recovered as a product of destructive distillation and subsequently worked up into different merchantable forms, as sulphate, aqua ammonia, etc. The amount of ammonia, or more specifically, nitrogen, contained in the various packing-house fertilizer products as dried blood, tankage, steamed bone meal, hoof meal, etc., is very large; it has been estimated as 70,000 tons of nitrogen for the year 1906.[1] This industry is so complicated and so extensive as to require treatment by itself.

Table I gives the ammonium sulphate equivalent of the ammonia produced in various forms in the United States for each year since 1898. These figures are based on the data published in "Mineral Resources of the United States" for 1898, 1902, 1903, 1904, and 1905, the intervening years being estimated from the best data obtainable. As no figures have appeared from the above source for the last two years, these also have been estimated. Allowance has been made throughout the table for the ammonia produced in the bone-black industry.

TABLE I.—UNITED STATES AMMONIA PRODUCTION, EXPRESSED IN SULPHATE EQUIVALENT.

(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.
1898......	17,000	1902......	36,124	1906......	(a) 75,000
1899......	(a)19,500	1903......	41,873	1907......	(a) 89,000
1900......	(a)27,600	1904......	54,664		
1901......	(a)29,279	1905......	65,296		

(a) Estimated

The annual increase in production as shown in Table I is principally due to the by-product coke oven, which, according to reports made to me by the plants in operation, produced 62,700 net tons of sulphate and sulphate equivalent in 1907. The by-product ovens are responsible, therefore, for 75 per cent. of the total output, considering the estimates for the remainder of the production to be approximately correct. On the same basis the increase in production for 1907 over 1906 is about 20 per cent.

Table II gives the ammonia consumption of the United States, expressed in sulphate equivalent, including the amount of sulphate imported as such, and the average market price for each year since 1899. The figures for production used in arriving at the total consumption are those

[1] *American Fertilizer*, December, 1907, p. 14.

in Table I, and those for the average market price are computed from the quotations regularly published by the *Engineering and Mining Journal.*

As shown by Table II, the imports of sulphate have not maintained a regular movement either up or down, but a falling off in demand in 1906 was followed by a sudden increase to 30,114 tons, 66 per cent. more than the imports for the largest preceding year. The consumption, however, has increased regularly, the highest single increase being that of 1907 over the preceding year. This indicates that the demand for ammonia in this country is a constantly increasing one, and that it is keeping well ahead of the home production. This tendency has been strengthened by the growing use of sulphate as a fertilizer, due partly to the awakening of the American farmer to the value of fertilizers in general and partly to the rise in the price of other nitrogenous fertilizers. The use of sulphate in this field will unquestionably be much greater in the near future.

TABLE II.—UNITED STATES AMMONIA CONSUMPTION, EXPRESSED IN SULPHATE EQUIVALENT. (*a*)

(In tons of 2000 lb.)

Year.	1899	1900	1901	1902	1903	1904	1905	1906	1907
Imports	6,976	8,411	14,486	18,146	16,777	16,667	15,288	9,182	30,114
Total consumption	26,476	36,011	43,765	54,270	58,650	71,331	80,584	84,182	119,814
Average price	$50.29	$57.40	$55.16	$59.90	$62.10	$61.71	$62.92	$62.33	$61.93

(*a*) The figures for consumption and price are for the calendar year, while those for imports are for the fiscal year ending June 30.

The importation of sulphate of ammonia alone does not, however, show that the increased demand is in this direction only. Increases in demand for other forms of ammonia are usually met by the home product, and sulphate is imported to supply the deficiency, since it is the most convenient and available form for shipping purposes.

It will be noted that the total estimated consumption of the United States is now nearly 120,000 net tons of sulphate and sulphate equivalent. This is more than is consumed by any other country except Germany. As shown by Table II, the average price of sulphate for 1907 remained about the same as for 1906.

TABLE III.—AMMONIUM SULPHATE AND SULPHATE EQUIVALENT PRODUCED IN THE UNITED KINGDOM. (*a*)

(Tons of 2240 lb.)

Year.	1901	1902	1903	1904	1905	1906	1907
Gas works	142,703	150,055	149,489	150,208	155,957	157,160	162,000
Iron works	16,353	18,801	19,119	19,568	20,376	21,284	22,000
Shale works	40,011	36,931	37,353	42,486	46,344	48,534	51,000
Coke ovens	12,255	15,352	17,438	20,848	30,732	43,677	59,200
Producer gas and carbonizing works	5,891	8,177	10,265	12,880	15,705	18,736	21,800
Total	217,213	229,316	233,664	245,990	269,114	289,391	316,000

(*a*) These figures are from the Alkali Inspector's reports, except those for 1907, which are from Messrs. Bradbury & Hirsch's review.

United Kingdom.—According to Table III the output of the United Kingdom amounted to 347,600 net tons in 1907, or four times that made in the United States for the same year. The product of the gas works in 1907 was 51 per cent., that of the iron works 7 per cent., that of the shale works 16 per cent., that of the coke ovens 19 per cent. and that of the carbonizing and producer gas works 7 per cent. of the total.

A large increase became noticeable in the coke oven production in 1904, and has kept up steadily ever since. It is from this source that most of Great Britain's annual gain in output comes. The coke oven production was 66,300 net tons in 1907, somewhat more than was made by the coke ovens in the United States for that year.

Great Britain's home consumption is stated to be 87,500 gross tons for 1907, the remainder of the output being exported. The statistics of the export trade are chiefly remarkable for the position that Japan has assumed as the largest single customer, taking 64,270 tons, Spain, the previous leader, being satisfied with 44,541 tons, while the United States comes third with 24,920 tons.

The average price per ton for 1907 is given as £11 15s. 8d., a drop of 5s. 1d. from the price for 1906, and the market conditions are said to have been good, on the whole.

TABLE IV.—PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT FROM BY-PRODUCT COKE OVENS AND GAS WORKS IN GERMANY. (*a*)

(In metric tons of 2204.6 lb.)

Year.	1900	1901	1902	1903	1904	1905	1906	1907
Gas works	16,500	17,000	18,000	20,000	21,000	22,000	....	(*b*)30,000
Coke ovens	88,500	113,000	117,000	120,000	152,000	168,000	....	257,000
Total	105,000	130,000	135,000	140,000	173,000	190,000	(*b*)235,000	(*c*)287,000

(*a*) Dr. N. Caro, *Zeit. f. angew. Chem.*, Sept. 14, 1906.
(*b*) Deutsche Ammoniak-Verkaufs-Vereinigung, 1906-7.
(*c*) *L'Engrais*, March 6, 1908.

Germany.—From Table IV it is evident that the production of ammonia in Germany is still increasing more rapidly than in any other country, an increment of 45,000 tons in 1906 having been followed by one of 52,000 tons in 1907. This development, it should be stated, is almost entirely due to the by-product coke oven, in part to the addition of new plants and in part to the more continous operation of the plants already existing, which has been made possible by industrial conditions. The production from gas works is of minor importance and increases but slowly.

The exports from Germany in 1907 were 57,493 metric tons, and were for the first time largely in excess of the imports, which amounted to 33,522 metric tons for the year. As a consumer Germany stands easily first among the nations, having taken about 263,000 metric tons last year.

France.—France produced 52,700 metric tons of sulphate or its equivalent in 1907, exported 9682 tons and imported 30,110 tons; therefore consumed 72,948 tons for the year.

Japan.—The progress made by Japan in the consumption of sulphate of ammonia is remarkable. Its production has been estimated, though not authoritatively, at 500 tons per annum, from gas works and by-product coke ovens, but the importations from England have been increasing at a surprising rate, as shown by Table V.

TABLE V.—EXPORTS OF AMMONIUM SULPHATE FROM THE UNITED KINGDOM TO JAPAN.

(Tons of 2240 lb.)

1901	1902	1903	1904	1905	1906	1907
1,290	2,429	3,612	14,981	33,861	33,237	64,270

Whether these figures represent actual consumption, or are partly to cover future needs is impossible to say.

TABLE VI.—ESTIMATED WORLD'S PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT.

(Metric tons of 2204.6 lb.)

Country.	1902	1903	1904	1905	1906	1907
England	233,100	237,520	250,050	273,550	294,170	331,220
Germany	135,000	140,000	173,000	190,000	235,000	287,000
United States	32,800	38,000	49,600	59,250	68,000	81,400
France (*b*)	40,000	52,000	43,000	47,300	49,100	52,700
Belgium and Holland (*b*)	38,000	35,000	(*a*)39,000	24,200	30,000	(*a*)55,000
Spain (*b*)				10,000	10,000	(*c*)12,000
Italy (*b*)	45,000	45,000	48,000	4,500	5,000	11,000
Other Countries (*b*)				40,500	40,000	65,000
Total	523,900	547,520	602,650	649,300	731,270	895,320

(*a*) Including Norway, Sweden and Denmark.
(*b*) Estimates from *L'Engrais*.
(*c*) Including Portugal

Plants for Making Sulphate of Ammonia.

By C. G. Atwater.

Sulphate of ammonia has been manufactured in the United States for a number of years, but until the advent of the by-product coke oven the necessary ammonia was nearly all produced in coal-gas works, in small quantities, at points more or less remote from each other, so that where sulphate plants were installed they were necessarily of small capacity and simple type. These conditions have, in a great measure, been changed. A number of large by-product coke oven plants have been built, many of which are successfully converting the ammonia they recover into sulphate. Moreover, the growing interest displayed by agriculturists in the use of

sulphate of ammonia as a fertilizer, as indicated by the increased importations, gives assurance of a ready market for this product.

As in other lines of manufacture, we may expect to find the most economical production in the plants that operate on a large scale. This is particularly true of products like sulphate of ammonia, the making of which involves a certain amount of manual labor which may be cheapened or eliminated in large installations. It is partly for this reason that the present output of sulphate in this country comes almost entirely from by-product ovens. These plants are by far the largest individual producers of ammonia, and therefore afford the opportunity for the making of sulphate in quantity. The usual recovery of sulphate from coal is about 1 per cent. of the weight of the coal carbonized, so that for the production of five tons of sulphate per day 500 tons of coal must be carbonized. There are but few coal-gas plants of this daily capacity in the United States, but on the other hand most of the by-product oven plants exceed it. In England coal-gas works of such capacity are more frequent, and those of smaller size are comparatively closer together, so that opportunity has been afforded for the establishment of ammonium sulphate plants that manufacture on a large scale, in some cases drawing their supply of liquor from several sources. In Germany also the large number of by-product oven plants built there has caused a great deal of sulphate to be made. These two countries produce two-thirds of the sulphate output of the world. It is the object of this article to describe the construction of some of the later types of sulphate plants, as built here and abroad.

Chemistry of the Process.—The chemistry of ammonium sulphate has been frequently and adequately described elsewhere[1], so that it will suffice to give only an outline here. The ammonia is recovered from the unpurified coal gas by washing it with water, and this, as crude ammoniacal liquor, forms the raw material for the sulphate plant. It usually contains from 0.5 to 2 per cent. of ammonia, existing in various forms. Those that part with their ammonia on boiling, as the carbonate and sulphide, are known as "free" ammonia, while those that do not decompose at this temperature, as the chloride, sulphate, sulphite, thiosulphate and ferrocyanide, are known as "fixed" ammonia.

The crude liquor is passed through a still, encountering a current of live steam, and is thus deprived of its free ammonia. The liquor containing the fixed ammonia is then mixed with a certain proportion of milk of lime (or other alkali), the effect being to free the fixed ammonia, which is then driven off by steam distillation, as before. The reactions that render the fixed ammonia free may be typified by that for ammonium chloride, which follows:

$$Ca(OH)_2 + 2(NH_4)Cl = CaCl_2 + 2H_2O + 2NH_3$$

[1] See Lunge, "Coal Tar and Ammonia"; Cammille Vincent, "Ammonia and Its Compounds"; R. Arnold, "Ammonia and Ammonia Compounds"; F. Schniewind, *The Mineral Industry*, Vol. X, p. 149.

The gaseous ammonia, mixed with steam and the volatile impurities, as carbonic acid, hydrogen sulphide and tarry vapors, pass from the still to the condenser or separator, where a certain amount of water is removed, and are then led to the saturator. Here they are brought into intimate contact with dilute sulphuric acid and chemical combination ensues which may be expressed by the following equation:

$$2NH_3+H_2SO_4=(NH_4)_2SO_4$$

This reaction is attended with considerable heat. The sulphate of ammonia forms as small crystals and settles to the bottom, and the incondensible gases are led away to waste. Owing to the noxious character of these waste gases they are usually conducted to a smoke stack in operation, if one is at hand, or disposed of in some other manner. In the most modern plants the waste gases are chemically treated and rendered inocuous before being allowed to escape into the air.

Nature of Plant Required.—The essential parts of the apparatus are therefore a still, in which the ammonia is driven off, and a saturater, in which it is combined with acid. The still consists of the section for free ammonia distillation, the intermediate liming vessel, and the section for the distillation of the fixed ammonia set free by the action of the lime. This part of the apparatus is almost invariably made of cast iron. The stills are usually built up of separate sections fastened one above another and provided with a system of partitions, connecting passages and sealing hoods, so that the ammonia liquor entering at the top of the column and flowing gradually downward is exposed to the action of the steam which enters below and passes upward, bubbling through the successive seals. This is the general plan, subject to many individual variations in design.

Accessory to the still are the pumps and tanks for crude liquor supply, the apparatus for slacking, measuring and injecting the lime, and the necessary steam piping, water piping, sewer connection for spent liquor from the still, etc. Frequently provision is made for a condenser which strengthens the ammonia leaving the still by removing some of the water vapor, and for a heater which permits the preheating of the crude feed liquor by the heat from the outgoing spent liquor or the waste gases. In some cases the concentration of the ammonia leaving the still is effected in whole or in part by the cold feed liquor.

The saturator consists of a tank or vessel of wood or iron, lined and protected with sheet led to prevent corrosion by the acid. The ammonia vapor is led beneath the surface of the acid in the saturator by a dip pipe or bell, also lead protected, usually so arranged as to compel all the vapors to bubble through the acid and allowing all those that are not absorbed to be carried away by a special connection, so that they shall not escape into the air. Accessory to the saturator are the tanks and

pumps for supplying and handling acid and mother liquor and the appliances for removing the sulphate from the saturator, draining it, drying it and handling it to storage. Arrangements are also made to remove any entrained acid or water that may escape with the waste gases from the saturator, and in some cases a plant for the chemical treatment of these gases is provided. Provision is also made for the supply of the steam necessary to run the still and pumps, and where conveying and other machinery is adopted, power is also needed.

In the following descriptions of various plants the simpler ones will be taken up first, and those of more complicated character next in order.

Bamag System.—Fig. 1 shows the "Bamag" system (Berlin-Anhaltische Maschinenbau- Actien-Gesellschaft) built in the United States by Bartlett,

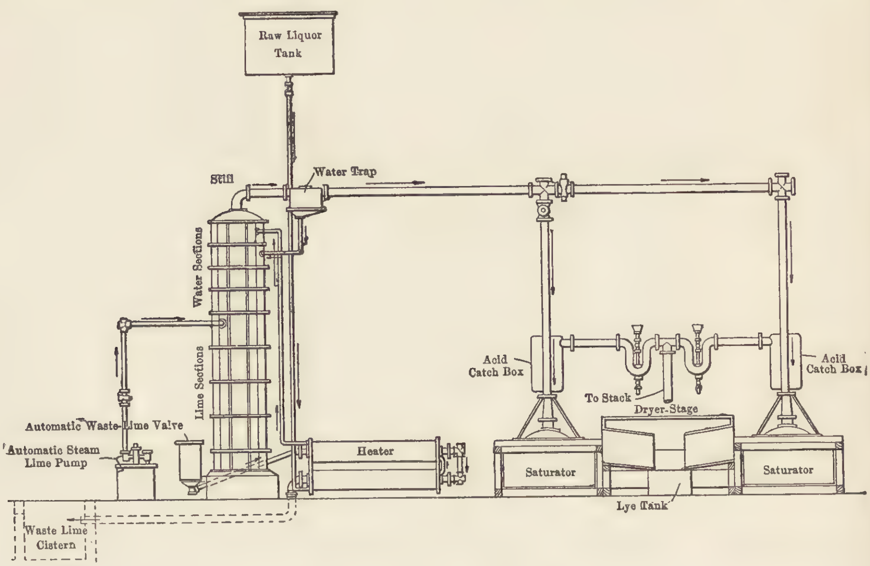


FIG. 1.—AMMONIA SULPHATE PLANT. BAMAG SYSTEM.

Hayward & Co. This plant consists of an ammonia still with an overhead wrought iron crude liquor feed tank with wooden float and scale, automatic steam lime pump with agitator tank, automatic waste liquor valve, water trap, crude liquor heater, two saturators with lead covered wooden acid boxes, lead bells and lead covered acid catch boxes, lead covered wooden drying stage and mother liquor tank, as shown. There are also an all-iron hand pump, a steel storage tank for acid, an air compressor for lime agitation and a centrifugal dryer for sulphate, not shown in the drawing.

The crude liquor passes by gravity to the heater, where it is heated by the waste liquor from the still, the hot feed liquor passing then to the

upper part of the still, where the volatile ammonia is driven off. The lime is pumped into the intermediate section of the still and is thoroughly mixed with the liquor as it passes downward through the lime sections. The waste liquor leaves the still by the automatic waste valve. This waste, it is stated, should not contain more than five parts of ammonia in 100,000.

A water trap or separator placed close to the still serves to remove the condensed water from the ammonia gases, the condensate draining back to the still. The ammonia gases then enter the saturator through the lead bell. The unabsorbed gases pass from the bell to the acid catch

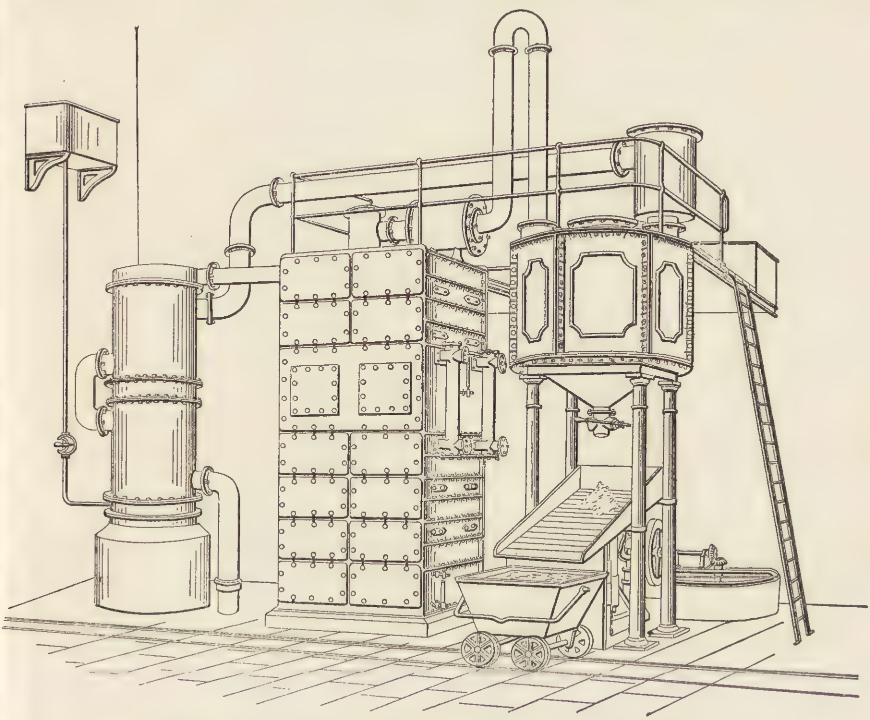


FIG. 2.—WALKER'S SULPHATE OF AMMONIA PLANT WITH SQUARE STILL AND SELF-DISCHARGING SATURATOR.

box where the liquid acid is retained; the waste gases are sent to the stack.

The formation of sulphate in the saturator is greatest under the lead bell, whence it must be removed by a wooden rake, and dipped out by hand with a copper ladle when the acid is nearly neutralized. The sulphate thus removed is then drained on the drying stage and the drainings run into the mother liquor tank. The sulphate may be then handled to the centrifugal dryer and thence to storage.

In this type of plant the still is operated continuously and the satu

tors alternately, one being saturated while the other is having the sulphate dipped out and the acid bath prepared for another run. Its operation also involves handling the sulphate by manual labor from the saturator to the drain board, thence by wheelbarrow to the centrifugal dryer, and again from the dryer to the storage bin.

Walker Plant.—An arrangement eliminating some of the manual labor is shown in the following plant, which is by C. & W. Walker, of London, England. (See Fig. 2.) The still is of the square type, the free still being the upper sections, the liming chamber the intermediate section, and the fixed still the lower sections of the column. The crude liquor flows from an overhead supply tank through the tubes of a vertical tubular heater and absorbs heat from the waste gases coming from the saturator. The tubes of the heater are provided with expansion joints at either end. The lime is mixed with water in a mixing tank provided with mechanical agitators driven from the steam pump which also injects the lime milk into the still.

In the lower section of the still and also in the liming chamber there are perforated steam coils, supplied with steam through a reducing valve which maintains the desired amount of pressure. The lower coil serves for the steam admission to the still, while that in the liming chamber is primarily to agitate the lime and cause it to mix properly with the liquor. Special emphasis is laid on the ease with which the still can be cleaned and upon the freedom from stoppages from light tar carried in with the liquor, by virtue of the size and arrangement of the overflow connections possible in a square still.

The ammoniacal gases escaping from the still pass through a baffle box where the condensed water is removed, thence through an inverted U-pipe to the saturator, which is of special type. It is cylindrical in form and has a flat top and a conical bottom, the whole being supported on columns above a drainage board. The shell of the saturator is of cast iron and it is lined throughout with lead. The sides are of segmental plates, between the flanges of which the lead sheets are inserted and firmly held, the lead being afterward burned together to insure a thoroughly tight joint. The same style of joint is used in the bottom and top. The ammonia inlet and the waste gas outlet connections are made to the top of the saturator and are of the usual form. In the center of the top is a clean-out opening of large enough diameter to permit a man to enter the saturator to make repairs, and from this opening depends a lead curtain pipe which dips into the acid in the saturator, thus allowing access at any time. The discharge of the saturator, which is facilitated by the conical form of the bottom, takes place through a Howell patent discharge valve made of specially prepared copper. This valve consists of a casing in which is mounted a plunger capable of backward and forward motion by means of

a lever. The movement of the plunger serves to clear the passages of any incrustation or deposited sulphate.

The discharged sulphate and mother liquor are received on the drainage board and the liquor is drained off to a cistern, while the sulphate when sufficiently dry may be handled to a centrifugal dryer. Or, it is stated, the saturator may be placed directly over the centrifugal dryer if desired. As the sulphate is much hotter than when it is dipped out with a ladle in the usual way, it is said to dry much more rapidly, and to be of better color. The yield is also claimed to be a little higher.

In addition to the saving of labor in handling the sulphate which this form of saturator makes possible, an additional advantage is claimed for the durability of the lead lining, which is not bruised or abraded by the action of the dipping ladles. The waste liquor from a still of this type

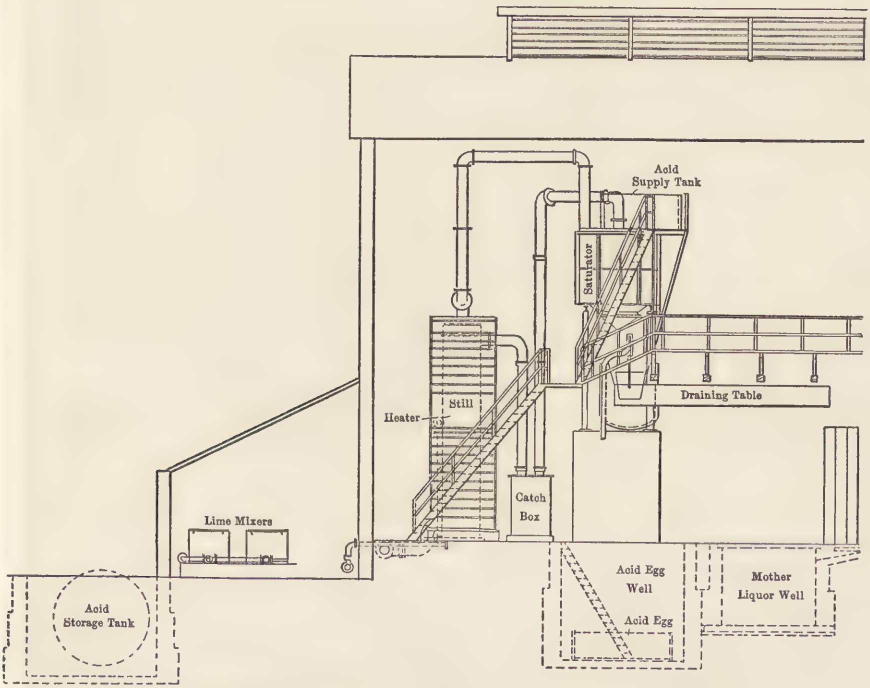


FIG. 3.—SULPHATE OF AMMONIA PLANT BY JOHN H. DARBY. ELEVATION.

should, it is stated, contain less than 0.006 per cent. NH_3, and this with the consumption of 8 cwt. of good coal per ton of sulphate produced from crude liquor of 5.25 deg. Twaddell.

Darby Plant.—Figs. 3 and 4 show in plan and elevation a sulphate of ammonia plant typical of those erected by John H. Darby, of Brymbo, Wales, in connection with by-product coke oven plants. This design

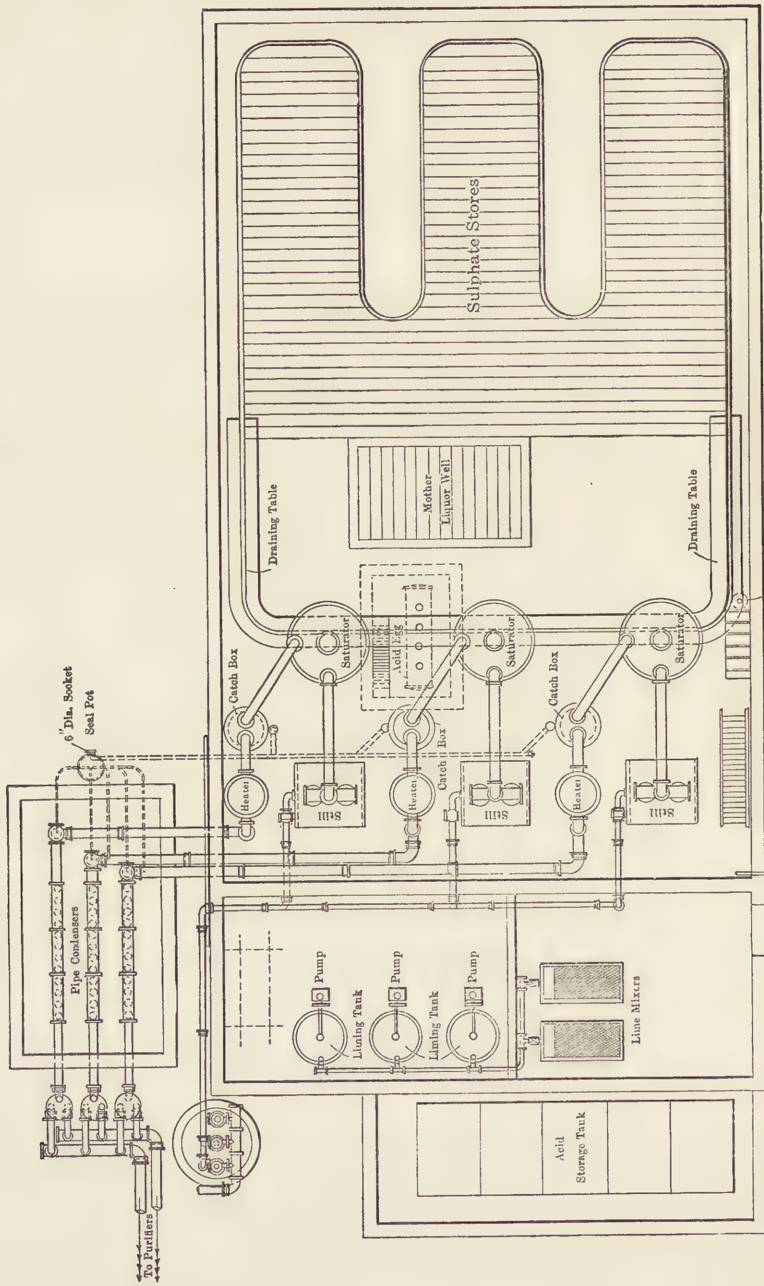


FIG. 4.—SULPHATE OF AMMONIA PLANT BY JOHN H. DARBY, GROUND PLAN.

shows a step further in the direction of saving labor, in that the sulphate from several stills is handled mechanically from the saturators to storage without any hand labor whatever. The plant shown is said to be capable of dealing with 250 to 270 tons of coke oven liquor per 24 hours[1] and is erected in three independent units, any of which may be laid off without interfering with the others. Each unit consists of a still, a heater and a saturator.

The cold in-going ammonia liquor contains from 1.25 to 1.50 per cent. of ammonia (NH_3) of which 0.30 to 0.40 per cent. is fixed ammonia.

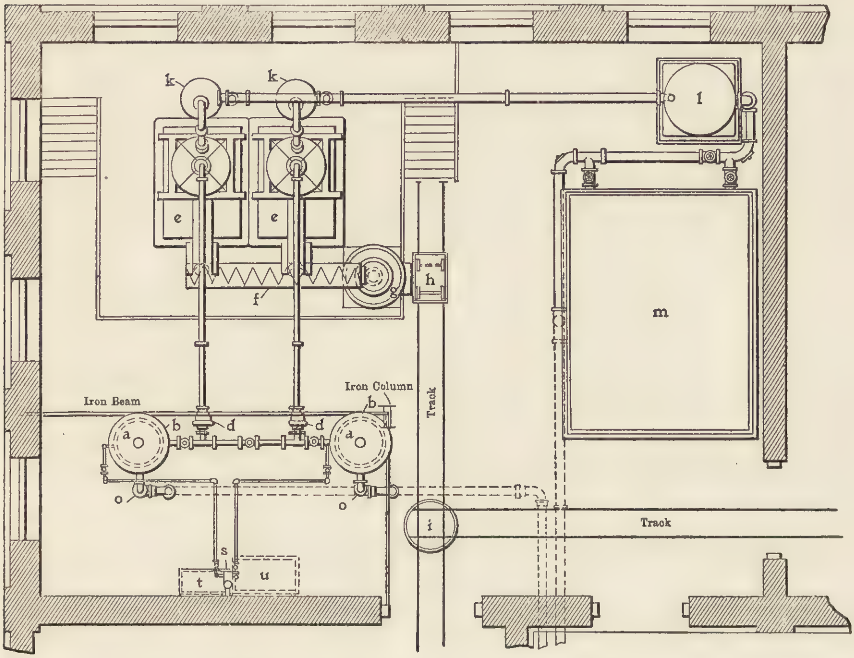


FIG. 5.—SULPHATE OF AMMONIA PLANT BY CARL FRANCKE.

A, stills; *b*, condensers; *c*, catch boxes; *e*, saturators; *f*, conveyer screw; *g*, centrifugal; *h*, dump car; *i*, turntable; *k*, acid catch box; *l*, tubular cooler; *m*, purifier; *o*, automatic waste valve; *s*, automatic lime pump; *t*, lime mixing box; *u*, slacked lime box.

The actual loss of ammonia in the effluent liquor is given as 0.004 per cent. The liquor is fed by gravity through the heater, where it is preheated to about 80 deg. C. by the waste gases from the saturator. The still is square in section and of the combined free and fixed type, with a liming chamber about the middle of the column. Each compartment has four hoods, which are accessible by removing the side plates. The overflows from one compartment of the still to another are particularly liable to block up, and to avoid having to expose the whole compartment of the

[1] Equivalent to 12 to 16 tons of sulphate with liquor of the strength mentioned below.

still by removing the side plate, the overflows are made exterior to the still and may be cleaned by removing a small cover plate.

The limy effluent liquor from the still is sealed in seal pots of sufficient depth to maintain a pressure of about 5 to 8 lb. per sq.in. in the stills, and then flows to settling tanks to clarify and cool.

The ammonia vapor is delivered to saturators of the self-discharging type, elevated sufficiently to discharge the salt into buckets running on trolleys around the salt store. A continuous table catches the drainage from the buckets and delivers it to the mother liquor tanks, whence it flows into the mother liquor well below the floor. The same well collects the drainage from the sulphate store. The mother liquor flows by gravity from the well into an egg and is blown by compressed air into an overhead tank for feeding the saturators. The fresh acid is blown by compressed air from the large acid storage tank outside the house to the elevated acid and mother liquor mixing tank, where the bath is prepared before flushing it into the saturators.

The foul gases from the saturators pass through the catch pots where some of the watery vapor is intercepted, and thence through coolers (where the ingoing liquor is heated) to condensers. The uncondensed gases, chiefly sulphureted hydrogen, are passed through a Claus kiln to recover the sulphur, and finally through oxide purifier beds of the heaped-up type as a safeguard.

Francke Plant.—Fig. 5 shows a sulphate of ammonia plant which is part of a complete plant for making concentrated liquor, anhydrous ammonia and sal ammoniac as well, being the design of Carl Francke of Bremen, Germany. Only that portion that has to do with the production of sulphate is shown. There are two stills of circular section, operating in connection with two saturating boxes of special type, the connection between stills and saturators permitting them to work as units or together. The stills are of the single column type, comprising fixed and free stills and liming chamber, and each is provided with a water-cooled condenser and a water separator for the ammonia vapors. The waste liquor escapes from the still through an automatic seal to the sewer. Tanks are provided for lime mixing and agitation, and automatic pumps for supplying it to the stills.

The hot waste gases from the saturators are passed through a water separator and then pass to a tubular cooler, in which they heat the crude ammonia feed liquor. The gases themselves are then passed through a purifying box as shown, before they are allowed to escape to the atmosphere.

The saturators are arranged so as to be capable of continuous operation, the ammonia gas passing through the acid twice, and the deposited sulphate being removed from the saturator by an inclined screw conveyer. This is known as the Theil system. The general arrangement of the two

boxes is shown in Fig. 5. This figure also shows the screw conveyer which delivers the drained sulphate to the centrifugal dryer and the dumping car below, into which the dried salt falls from the centrifugal and in which it is transported to the salt storage room. The details of the saturator are shown in Fig. 6. The box itself is of wood, lined with lead, and

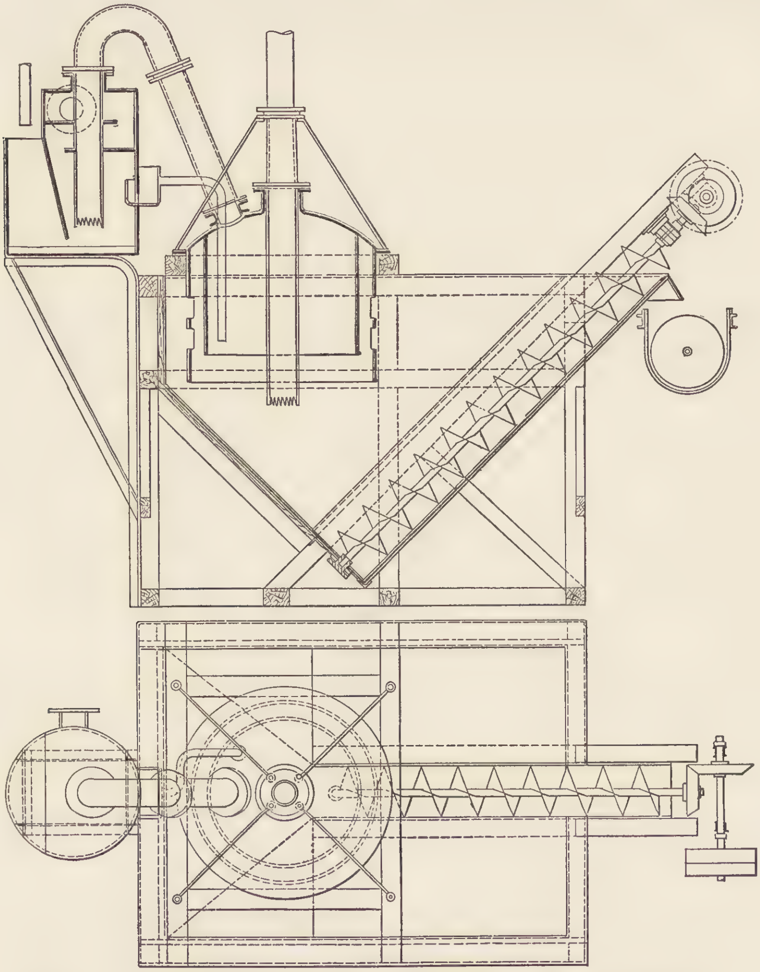
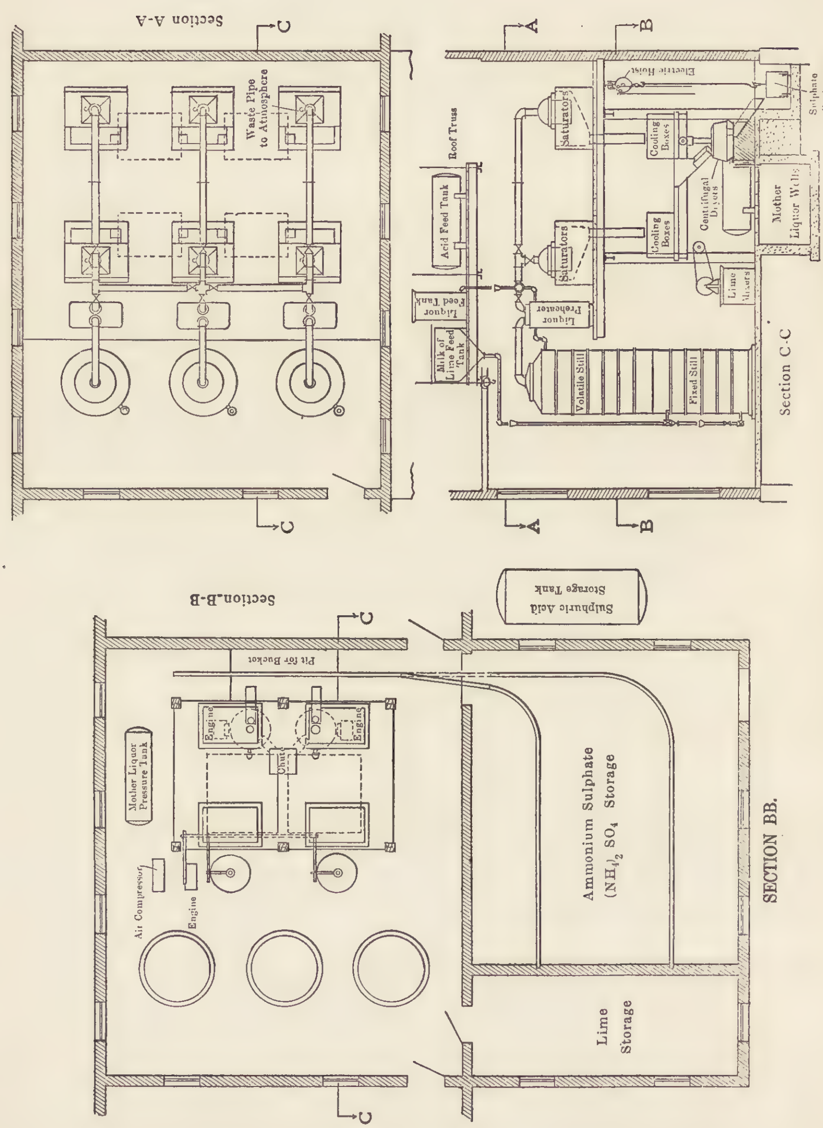


FIG. 6. THEIL SATURATOR WITH SECONDARY ACID CHAMBER AND MECHANICAL SULPHATE DISCHARGE.

is rectangular in plan, three sides being vertical down to about the level of the lower bell, while the fourth side has a continuous slope. The saturating bell itself has the usual center pipe of lead, through which the gases from the still are led beneath the surface of the acid and made to bubble through it, but surrounding this pipe are two lead bells instead



SECTIONS AA–CC.

FIG. 7.—SULPHATE OF AMMONIA PLANT BY UNITED COKE AND GAS COMPANY.

of the usual single one. The outer bell dips deeper than the inner one, and is provided with a double row of openings, the upper ones being just covered by the acid at normal level, while the lower ones are about 12 in. beneath the first. The object of these openings is to provide for circulation of the acid as it becomes heated and neutralized by chemical combination under the bell, so that it will rise between the inner and the outer bells and pass out at the upper openings, while the fresher and cooler acid will pass in by the lower openings to take its place under the bell. The circulation of the acid can occur to some extent beneath both bells until enough salt has been deposited to fill, partly or wholly, the space between the outer bell and the floor of the saturator, when circulation in this direction will be cut off. This is the time, however, when the acid bath is approaching neutralization, and consequently when the loss of ammonia by over-saturation is most likely to occur. It is claimed that such loss, amounting at times to 10 per cent. of the ammonia treated in the older types of saturator, will be avoided by the use of this form of bell.

Up to the point at which the acid is nearly neutralized, the ammonia is all absorbed in the main saturator by virtue of the active circulation. What ammonia escapes after this point has been reached will be taken care of in the secondary saturator. This apparatus has also the advantage of making continuous operation possible with but one saturator. When the salt begins to form the acid in the secondary saturator is run into the main box, and fresh acid supplied instead. The saturation is then completed and the salt removed from the box, the ammonia that escapes being taken care of by the acid in the second chamber, the contents of which suffices for 1½ to 2 hours' run. When the salt is out fresh acid is run in through the second chamber, as before, and the saturation is proceeded with.

The method of removing the salt from the saturator, as shown in Fig. 6, consists of a screw conveyer of phosphor-bronze running in a lead-lined conveyer trough formed in the inclined floor of the saturator itself. This conveyer is operated by power in the illustration, though it is stated that it may be operated by hand if desired. The salt is guided into the conveyer trough by the three sides of the box which slope inward from the lower level of the bell so as to form an inverted pyramid.

The salt so removed from the saturator is sufficiently dry to make a drainage board unnecessary, but the second conveyer in which the salt is handled to the centrifugal is given a slight pitch toward the saturator so that further drying may ensue. The conveyers are run until the centrifugal is fully charged, the salt is then dried by rapid revolution, and the charge is dumped into the car below, and a fresh charge is run over.

It is claimed that the continuous operation of this plant offers no

difficulties whatever, and that the yield of finished salt is increased.

United Coke and Gas Company.—Fig. 7 shows the sulphate of ammonia plant designed by the United Coke and Gas Company, of New York. This plant consists of three units, each of a still, heater and two saturating boxes, arranged so that they may be operated separately or in combination, as desired. The saturating boxes are located on an elevated floor sufficiently high to admit of the gravity discharge of the salt to cooling boxes on a lower level and to centrifugal dryers on the ground floor, the dryers in turn discharging to trolley buckets below, by which the salt is conveyed to the storage room. By this arrangement the labor of handling the salt is reduced to a minimum.

The stills are of the single column type, comprising volatile and fixed sections with intermediate lime section. The lime is mixed in mechanical lime mixers on the ground floor and pumped up to an overhead feed tank in which it is agitated by compressed air, entering the stills through a sight feed. The crude liquor is fed by gravity to a heater, passing thence to the upper section of the still. Through this heater pass the exit gases from the still on their way to the saturator, this arrangement serving to increase their strength in ammonia, at the same time preheating the feed liquor. The saturators are lead-covered, with an inclined bottom and a bottom discharge valve, the middle pair having two discharge valves so that they may empty into either of the two cooling boxes beneath them. Mother liquor for making up the acid bath after a saturator is discharged is supplied at this elevation from the pressure tank by means of compressed air as desired, the acid being delivered from the overhead acid storage tank as shown.

The cooling boxes, of which there are four, are of wood, lined with lead. After a charge has been dropped from the saturator to the cooling box and has been allowed to stand a short time the mother liquor is syphoned off the top and discharged to the mother liquor well. The sulphate is then shoveled into the chute leading to either of the centrifugal dryers. Of these there are two, operated by belts, power being furnished from a small engine placed on the main floor. Power from the same source serves to drive the lime mixers. From either centrifugal a chute leads the dried salt to a dumping bucket standing in a pit provided for the purpose, and carried by an electrically operated hoist and trolley. The trolley delivers the salt to the storage room where it is bagged for shipment.

The capacity of this plant is approximately 15 tons (of 2000 lb.) of sulphate of ammonia (25 per cent. NH_3 test) per 24 hours, under ordinary conditions of operation.

I wish to acknowledge the courtesy of the gentlemen mentioned in this article, who have aided me in its preparation by affording data for the description of the various plants.

ANTIMONY.

The only antimony smelter that ran continuously in this country in 1907 was that of Mathison & Co., at Chelsea, Staten Island. The smelter at San Francisco ran for only a short time when prices were high, but for the last 10 months of the year it remained closed. The production of antimonial lead by the lead refiners was 9614 tons, against 10,120 tons in 1906. An additional quantity of antimonial lead was made by the Hoyt Metal Company, at Granite City, Ill., by smelting antimony ore along with non-argentiferous galena, thus making the alloy as a direct product, and not as a by-product as do the lead refiners. This production, however, was merely of an experimental nature and too small to be of statistical consequence. It is contemplated, however, that a considerable tonnage may be made in this way during 1908.

ANTIMONY STATISTICS OF THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Imports.		Exports.		Production.			Consumption.
	Metal or Regulus.	Ore.	Metal or Regulus.	Ore.	In Hard Lead.	From Domestic Ore.	From Imported Ore. (a)	
	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons
1897......	573	2,751	....	....	2,217	245	1,100	4,135
1898......	1,013	1,863	13	17	2,118	250	738	4,106
1899......	1,580	1,991	*Nil.*	*Nil.*	1,586	234	796	4,196
1900......	1,816	3,018	21	*Nil.*	2,476	151	1,207	5,638
1901......	1,837	866	*Nil.*	25	2,235	50	336	4,458
1902......	2,871	840	37	104	2,904	*Nil.*	294	6,032
1903......	2.563	1,337	40	*Nil.*	2,552	*Nil.*	535	5,610
1904......	2,028	1,245	16	214	2,515	*Nil.*	412	4,939
1905......	2,869	988	*Nil.*	*Nil.*	2,561	*Nil.*	395	5,825
1906......	3,950	1,124	12	*Nil.*	2,358	150	450	6,866
1907......	4,331	1,380	24	6	2,240	105	552	7,204

(*a*) Estimated at 40 per cent. extraction from net imports of ore.

Market and Prices.—The antimony market opened in 1907 at about 24c. per ℔. with a good demand and a moderate quantity booked ahead. At that time the statistical position was apparently good. The high prices, while they brought out a fair supply of antimony, apparently did not bring out too much for the world's needs. There had been a severe falling off in the production of antimonial lead and this greatly increased the consumption of pure antimony.

The market continued steady for the first three or four months and then gradually commenced to sag. As it has since developed this decline was due to a variety of causes. The market for needle antimony had to a large

extent been artificially sustained by a considerable proportion of China's output being held in strong hands on the Continent. With the appearance of larger quantities of ore from northern Africa, Turkey and Australia, the holders became afraid and needle antimony was allowed to drop.

From that time onward there was a steady and rapid decline, accelerated by the financial depression in this country and a consequent diminution of demand. At the end of 1907 it was reported that there were large stocks of antimony ore held in Europe which were mined to be sold on the basis of about 20c. per lb. for the metal.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK.
(Cents per pound.)

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901.													
Cookson's	10.00	10.00	10.00	10.31	10.25	10.25	10.25	10.25	10.12	10.09	10.00	10.00	10.12
Hallett's	9.12	9.22	8.90	8.94	8.75	8.75	8.75	8.43	8.50	8.47	8.37	8.31	8.74
Others	9.25	8.85	8.77	8.73	8.63	8.63	8.63	8.50	8.37	8.34	8.25	8.00	8.55
1902.													
Cookson's	10.00	10.00	9.87	9.87	9.87	9.87	9.75	9.75	9.69	9.44	9.25	9.20	9.71
Hallett's	8.17	8.04	8.06	8.06	8.17	8.25	8.25	8.15	7.92	7.72	7.44	7.25	7.96
Others	7.86	7.75	7.75	7.75	7.90	8.00	8.00	7.90	7.65	7.37	7.22	6.92	7.67
1903.	8.25	8.25	8.25	8.25	8.00	7.50	7.44	7.15	7.00	7.00	6.56	6.75	7.53
Hallett's	7.00	7.00	6.87	6.87	6.75	6.69	6.50	6.40	6.34	6.25	6.25	6.35	6.69
Others	6.75	6.62	6.50	6.50	6.50	6.44	6.25	6.19	6.00	6.00	6.00	5.95	6.31
1904.													
Cookson's	6.938	7.594	7.875	7.875	7.531	7.200	7.188	7.188	6.913	6.984	7.592	8.388	7.439
Hallett's	6.250	6.781	6.825	6.750	6.578	6.438	6.485	6.688	6.537	6.578	7.328	8.160	6.783
Others	5.688	6.203	6.475	6.406	6.203	5.961	5.969	6.062	6.015	6.172	7.204	8.088	6.371
1905.													
Cookson's	8.375	8.375	8.375	8.219	8.406	11.025	12.625	14.500	13.700	13.000	12.500	14.000	11.100
Others	8.063	8.063	7.638	8.125	8.406	10.175	11.875	13.500	12.900	12.000	11.250	12.750	10.400
1906.													
Cookson's	15.0	16.0	17.5	21.31	25.25	26.0	25.25	25.0	24.5	25.2	26.14	26.25	22.78
Hallett's	14.0	15.0	16.5	20.81	24.38	25.0	24.25	24.0	24.0	24.81	25.25	25.24	21.94
Others	13.5	14.25	16.15	20.25	23.31	24.0	23.19	22.75	22.25	23.63	24.50	24.70	21.73
1907.													
Cookson's	25.906	25.062	24.90	24.125	21.937	15.75	11.875	10.906	10.75	11.75	11.00	9.662	16.969
Hallett's	25.219	24.062	23.75	21.344	18.562	13.812	10.50	9.687	10.00	10.406	9.937	9.05	15.527
Others	24.156	23.437	23.025	20.875	17.75	12.65	10.125	9.375	9.65	10.047	8.906	8.088	14.840

Antimony Mining in the United States.

In the early part of 1907, when the price of the metal was high and financial conditions were favorable, antimony mines were eagerly sought and great activity in development work was apparent in several of the western states. Later, owing to declining prices and the financial depression, there was a corresponding falling off in mining. There is no reason, however, why the industry should not be profitable, even at present prices, if the ore occurs in sufficient quantity, and can be mined under ordinary conditions and concentrated economically. The production of antimony ore in the United States in 1907 was 210 short tons, averaging a little under 60 per cent. Sb, valued at $28,432; against 295 tons, valued at $44,250, in 1906.

The most important deposits are in Utah, Idaho, Washington and

Nevada, but discoveries and development work were reported during 1907 from different points in California and Alaska and in the vicinity of Silver City, South Dakota. In the last mentioned district the Grand View Mining Company employed 18 men and operated a five-stamp mill. In Nevada county, California, six miles south of Grass Valley, a promising vein of antimony was uncovered, chunks of ore weighing as much as 26 lb. being found. Owing to the low price of the metal development work is not being prosecuted.

Idaho (By Robert N. Bell).—The Stanley mine, situated near Burke, Shoshone county, gave the district its initial shipment of clean antimony ore in 1907. The development of the mine was greatly extended and there is now a large tonnage of ore piled up at the mouth of the lower adit awaiting better prices. The property is developed by two adits 150 ft. apart, the lower one being about 300 ft. vertically below the croppings of the vein over the present development. The uncertainty as to the size of the orebody has been to a large extent dispelled, the main ore shoot having been encountered in the lower adit and connection made with the upper adit by a raise which was all in ore. A drift was extended 100 ft. in ore on the lower level.

The ore seems to occur in flats and pitches, the raise from the lower adit being on a pitch of about 50 deg. for the first 50 ft., flattening out to about 35 deg. for the remainder of the distance. The orebody here varies from 5 to 20 ft. in width and is said to carry, in addition to the antimony, an average of $20 per ton in gold. There seems to be hardly any other mineral in the vein matter, in which respect a sharp contrast is presented with the neighboring lead-silver deposits. The vein strikes in a northerly and southerly direction and the footwall is clearly defined. About 1000 ft. south of the present development, on the same vein, a streak of the same class of ore has been revealed by a short crosscut adit. For the treatment of this ore the management proposes the erection of a 10-stamp mill, with special machinery for saving the antimony.

During the summer an option was taken by Spokane parties on an antimony property at Kingston, at the mouth of Pine creek, Shoshone county. Preparations for mining, however, were abandoned on account of the decline in the price of the metal. This property, which was at one time a producer, has lain idle for some years.

Nevada.—The only producing mines are those of the Antimony King Mining Company, situated near Austin and Battle Mountain. The ore occurs in well defined contact fissures between slate, which usually forms the footwall, and calcarous sandstone and lime porphyry. The shipping ore is found in lenses from 30 to 150 ft. in length. At Austin the vein is said to be 70 ft. wide, while those at Battle Mountain are from 8 to 10 ft. in width. Production from these mines in 1907 continued to be insignificant.

Utah.—The deposits of antimony ore on Coyote creek have been known for a long time, and were first described by Prof. W. P. Blake, in 1883. Attention was again directed toward them by the high price of the metal in the latter part of 1906 and early part of 1907.

The mines are situated in Garfield county, on Coyote creek, which is a tributary of the Sevier river. The valley of the creek is about five miles long and leads into Grass valley at its southern end. From the east fork of the Sevier, where it cuts across the valley, good roads run to the junction of the south fork and turn north from that point to Marysvale, a station on the Rio Grande Western Railroad.

The rocks of the district are soft, gray, granular sandstone underlaid by a thin bed of limestone and conglomerate boulders. The beds are of Eocene age, and are almost horizontal, forming perpendicular cliffs on both sides of the narrow valley and its branches. The conglomerate is about 100 ft. thick. Above it, in the sandstone, which contains some gypsum, the antimony deposits are found. The ore is disseminated through a band about 5 ft. thick, and rests directly upon the conglomerate. The ore is found both in regular sheets, ½ to 2 in. thick, and in lenticular masses approximately 2 ft. thick in the center and 20 ft. in diameter.

In recent years these properties have been worked intermittently by leasers who paid the owner a royalty of $75 per ton of stibnite extracted. The stibnite shipped contained on an average 71 per cent. antimony, and during mining operations several thousand tons of 20 per cent. ore were thrown on the dumps.

Early in 1907, the Utah Antimony Company began operations in this district, and although it has some ore developed in the mine, work for the present will be confined to the low-grade ore on the dump, of which there is claimed to be a large quantity. The ore is absolutely free from lead. The concentrating plant, which is capable of treating from 60 to 70 tons of ore per day, was put in operation during the latter part of 1907. A 55 per cent. concentrate is produced from ore carrying 11 per cent. antimony.

The Antimony Mining Company of Utah, also operating in Garfield county, spent the greater part of 1907 developing its mines near Coyote. Some ore was broken down but no shipments were made.

Washington.—Important deposits of antimony occur in Okanogan and Stevens counties. In the former, near Methow, the Gold Creek Antimony Mines and Smelting Company, which succeeded the Antimony Creek Mining Company, blocked out a considerable tonnage and produced a small quantity of ore in the course of development work. It is the intention of the company to erect a 25-ton concentrator.

During the summer the Sunset mine, near Northport, Stevens county, produced ore carrying antimony, gold and silver. An important strike

of antimony ore was reported 28 miles south of Kettle Falls, in the same county. Development work is being done but the owners do not expect to ship any ore until connection is made by the Columbia River Railway and Navigation Company with the Spokane Falls & Northern Railroad, which will provide for the shipment of the ore to Spokane for reduction.

The Lucky Knock Mining Company produced a small quantity of high-grade ore from its mines near Loomis.

Antimony Mining in Foreign Countries.

Australia.—This country is in a position to furnish a steady supply of ore, even at the present low price of the metal, and seems to be the most likely source of the supply for some years to come. The increased demand for the metal early in 1907 led to a resumption of operations on several claims in the Hillgrove and Metz districts, New South Wales. In spite of declining prices the industry prospered during the year, and two additional furnaces were started. An important factor in the industry is the ability of the operators to market ore carrying from 20 to 30 per cent. antimony, which is made possible by the operation of the smelter of the Metals Smelting Company at Balmain in Sydney.

There was also renewed interest in mining in Victoria. Antimony was produced at Costerfield and Reedy creek and is known to occur at Alexander, near Woods' Point, at Maryborough, and in Bendigo and Warrandyte. Antimony was at one time profitably mined at Leiderberg; work has also been prosecuted at the Panama mine, Lauriston, and at Whroo, south of Rushworth.

Bolivia.—According to G. Preumont (London *Min. Journ.*, March 7, 1908), an antimony mine near La Paz, at Palea, did good business during 1906, but is now closed down on account of the drop in prices. The mine is about 21 miles from the town, and the road, especially in the wet season, is bad, being practicable only for pack animals; moreover, transport is dear, and sometimes difficult to obtain. The mine is worked by adits from the hillsides, and the veins appear bunchy and irregular, but fairly persistent, and often of good width. The ore is very pure, and hand sorting is sufficient to obtain an export product averaging over 50 per cent. Sb. The mining is carried on in an unsystematic manner, being left entirely to the discretion of the miners, who are paid on the ore extracted.

A few antimony prospects are also known on the western side of Quimsa Cruz, at the head of the Luribay waters, but are not worked at present. The situation is not very favorable, since ore would probably have to be shipped via Oruro-Antofagasta—an expensive route. On account of the dearness of freight from mines to port, it is safe to say that antimony mining will not assume much importance in Bolivia for some time to come.

Canada.—The Dominion Antimony Company in 1907 mined 3042

tons of ore, of which 1403 tons were sorted out for shipment. About 450 tons was first-grade ore, averaging 45 per cent. antimony, and 1.5 oz. gold, the remainder being second-grade ore, averaging 20 per cent. antimony and 0.77 oz. gold.

Antimony was discovered on the property of the Golden Crown Gold and Silver Mining Company, operating on the north fork of Carpenter creek, in the Slocan district of British Columbia. The ore occurs in a lode about 30 ft. in width, which is traceable for three or four miles. As a rule the ore occurs cleanly, running high in antimony with small amounts of silver and bismuth, but in places it is mixed with soft, yellow quartz and is then difficult to sort.

China.—According to a recent British Consular report, the province of Kwangsi is believed to be richer in antimony than in any other mineral, but the export of this ore was made a government monopoly some years ago and the production, which was once considerable, has since completely ceased. The mines are situated mostly in remote districts. The mining industry in the province of Hunan is under the control of the Mining Bureau. Part of the ore mined is exported directly either by the Mining Bureau, or by private firms to whom it disposes of certain quantities under special agreement. The remainder is sold to two companies in Changsha, by which it is converted into regulus containing about 68 per cent. metal. This product is exported. In 1905 a total of 3696 tons of regulus and refined metal was exported, and in 1906 this was increased to 4221 tons. The exportation of antimony ore and metal from China in 1902 was 12,011 tons; in 1903, 9404 tons; in 1904, 7762 tons; in 1905, 6288 tons. The importation of antimony ore from China into the United Kingdom in 1906 was 1610 tons, against 316 tons in 1905; of antimony regulus and metal, 1628 tons in 1906, and 791 tons in 1905.

France.—The industry was not affected by the declining prices until the latter part of the year, and then not seriously. Production amounts to between 12,000 and 13,000 tons yearly, one half of which is converted into metal and the other half used in the manufacture of different salts and the oxide. The principal producers of antimony in France are: Société des Mines de Lucette, Société Italienne des Mines et Fonderies de Brioude, Société de Brioude-Auvergne, Société des mines d'antimoine de la Ramée, Société des mines et usines du Collet-de-Dèze, Société des mines d'antimoine de Mérinchal.

Italy.—Antimony is extensively mined at Pinerolo, near Turin, and throughout the Sardinian Islands; some of the product is exported.

New Zealand.—An Auckland syndicate, which has been developing an antimony prospect at Russells, made one shipment to London. A property is being worked on a small scale about six miles from Opua, at Bay of Islands. About 50 tons of ore have been shipped and development work is progressing.

Peru.—According to G. Preumont (London *Min. Journ.*, March 7, 1908), the conditions for antimony mining are more favorable in southern Peru than in Bolivia; especially in the Puno department and all along the strip of territory bordering the inner part of the railway system of Mollendo, the conditions are much better, being in much nearer proximity to a port of shipping, and having the facilities of railway communications. Here, also, transport from mines to the rail is easier and cheaper to obtain, the country possessing a large number of llamas—i.e., the sheep camel of South America; these animals carry loads of 100 ℔. easily, traveling at a rate of 15 miles a day. Antimony mines are found in the vicinity of Puno on Lake Titicaca and at different places along the line of the railway extending from Juliaca to Checacupe, and now being constructed onward to Cuzco, but generally at some distance east of the line up to as much as 100 km. The mines may be grouped as follows, beginning from the southern end:

The Pucara field has a few prospects hardly as yet touched, and one mine, that of Señor Chabanein, which exported some small parcels of ore in 1906. This mine is the nearest to the rail, being only about 9 miles from the station of Tirupate. It is in soft decomposed schists, and is worked as a quarry to obtain the bunches of mineral. The ore is remarkably pure. Araranca, further up the line, has one or two small embryo mines; started work in 1907, and since closed down. Aguas Calientes, a few miles more to the north, has also some antimony lodes, one of which is said occasionally to carry visible gold in the quartz matrix.

All these mines are fairly close to the line of railway, none being much over 10 miles from a station. Some distance east, and about 50 miles distant, are the fields of Macusani, where the lodes are well defined, and generally of better width. The country is metamorphic schist, very hard ground, and the climatic conditions are severe, owing to the great altitude.

North of Sicuani is the group of Chimboyo antimony mines, apparently of promising value, but too far distant from the rail to be worked at present. The region is wild and mountainous, and the only means of communication are paths of abominable description.

Most of the mines could easily be made to produce a fair monthly tonnage for export—say, from 20 to 50 tons of first-class ore that is well above 50 per cent. Sb. It is not possible to obtain any data as to the cost of production from the owners, who in all cases, when working, adopt the system of contract pay—i.e., paying a fixed price per quintal of 100 ℔. of ore produced and delivered to them. From information collected on the spot it would appear as if this price varied between 4s. to 8s. per quintal at the mines. With proper organization, and working on an adequate scale, it is probable that the lower figure would be near the

actual mining cost in many instances; and accepting this figure as a fair average, the ton of ore would come to somewhere about 90s. That is for ore picked, sorted, and ready for export.

The rail freight is moderate, and as the mines are so situated as to be all comprised in a definite zone, where the mileage varies from 550 km. to 675 km. from the port, the freight can be reckoned closely. This at $0.75 per km. per metric ton will be between the minimum of 19s. 6d. per ton and the maximum 23s. 8d. Averaging it at 21s. 6d., and taking into consideration the difference between the metric ton and the English long ton, also the dead weight of sacks and loss for moisture, the sum of 25s. appears to fairly represent the railage item to Mollendo. Agency, 7s. 6d.; shipping, 10s.; freight (maximum), 37s. 6d.; total 55s. Total for rail and sea transport to Europe, £4 per ton. Transport from mine to the head of rail is a more variable quantity, according to the position of the mine. As a rule it is reckoned at from 5 to 10 centavos per quintal of 100 lb. per *legua*, or 5.57 km., if llamas are employed, as is usually done. Mines situated within easy distance, i.e., 10 or 12 miles, would all be able to secure transport at less than £1 per ton; for those less advantageously placed it would vary from £2 to £2 10s.

From this it will be seen that antimony ores can be landed in the European market from southern Peru at prices ranging from £9 10s. to £11.

Turkey.—The industry has been seriously affected by the low price of the metal. The most important producing district is Murat Dagh, between the towns of Ushak and Kutaya. At present the principal producing mine is the Djinli Kaya, at Odemish. This mine has a rather crude concentrating plant of 40 tons daily capacity which loses about 30 per cent. of the pure ore in slimes and waste.

PRODUCTION OF ANTIMONY ORE IN FOREIGN COUNTRIES. (*a*)
(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907
Austria	905	864	679	410	201	126	18	41	103	1,673	1,071	......
Bolivia	......	......	......	1,213	1,174	190	126	59	7	17	571	......
Canada (*f*)	*Nil.*	*Nil.*	1,118	(*e*)	6	219	13	128	87	340	18,617	1,425
France & Algeria	6,333	5,466	4,571	7,592	7,963	9,867	9,715	12,380	9,065	12,543	1,307	......
Hungary	1,361	1,800	2,201	1,965	2,373	323	748	205	1,080	94)	......	......
Italy	5,086	2,150	1,931	3,791	7,609	8,818	6,116	6,927	5,712	5 083	5,704	......
Japan (*b*)	828	348	1,006	712	81	119	88	153	104	96	97	......
Mexico (*c*)	3,231	5,873	5,932	10,382	2,313	5,103	1,279	1,856	1,775	2,035	......	......
N. S. Wales (*d*)	135	172	84	332	252	90	57	13	111	394	2,490	......
New Zealand	21	10	......	......	5	30	......	......	......	......	......	......
Portugal	595	417	245	59	38	126	68	83	31	84	481	......
Queensland	......	......	......	41	......	......	......	......	*Nil.*	24	*Nil.*	*Nil.*
Spain	54	354	130	50	30	10	67	42	245	77	180	......
Turkey	100	400	(*e*)	1,173	267	224	(*f*)481	(*f*)1,903	(*f*)298	(*f*)188	(*f*)1,036	......
United States	136	454	(*e*)	544	300	100	*Nil.*	*Nil.*	*Nil.*	*Nil.*	267	190

(*a*) From official reports of the respective countries. It will be observed that this table omits the production of China, for which no statistics are available. (*b*) Mostly crude antimony. (*c*) Export figures, except for 1903, which represents production. (*d*) Metal and ore. (*e*) Not reported. (*f*) Previous to 1906 the figures represent exports for the fiscal year ending June 30; the figures for 1906 and 1907 were collected by *The Mineral Industry* and represent production for the calendar year. (*f*) Exported.

PRODUCTION OF ANTIMONY METAL IN FOREIGN COUNTRIES.
(In metric tons.)

	1895	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906
Austria..........	296	422	425	343	271	153	114	24	14	36	90	*Nil.*
France...........	779	969	1,033	1,226	1,499	1,573	1,786	1,725	2,748	2,116	2,396	3,433
Hungary (*a*)....	465	500	523	855	940	846	706	683	732	1,007	756	954
Italy............	423	538	404	380	581	1,174	1,721	1,574	905	836	327	537
Japan..........	641	517	823	235	229	349	429	528	434	321	190	627

(*a*) Regulus.

Progress in the Metallurgy of Antimony.

In recent years there has been remarkable progress in the metallurgy of antimony and its present status is far in advance of the accounts in the latest text books. The economical treatment of auriferous and argentiferous ores was formerly a perplexing metallurgical problem, but it has been solved by the simple volatilization process, introduced and practiced in France. In France also has been introduced the process of smelting antimony ore in the blast furnace, which is a natural procedure for ores that do not contain a significant quantity of the precious metals. In this process the presence of lead is no draw-back since antimonial lead or plumbiferous antimony now sell for practically the full alloy value. Indeed, antimonial lead is now produced designedly by smelting in the blast furnace antimony ore along with non-argentiferous galena. This is done at St. Louis, Mo., and at Anhalt, and perhaps elsewhere, in Germany. At Anhalt the first product is an alloy high in antimony, which is subsequently reduced to the percentage desired in the market by admixture of molten lead.

The general status of the antimony industry in Europe, with some brief notes on the metallurgical processes, was summarized in The Mineral Industry, Vol. XV, in an excellent article by F. T. Havard. Additional metallurgical data are given in the following notes and in the appended articles.

Volatilization in a Converter.—A. Germot in *Revue des Produits Chimiques*, Dec. 15, 1907, describes the following process: Antimony sulphide is melted in a converter, and air is blown in. The air burns part of the sulphur and produces sulphurous acid and antimony, which remains in the crucible. The operation is made continous by the addition of more ore. The sulphurous acid escapes through a pipe at the upper part of the converter, and carries off antimony sulphide fumes which are sublimated in special chambers. If a current of air is made to act on these fumes oxysulphides or antimony oxide will be produced, according to the amount of air.

Smelting in the Blast Furnace.—H. Herrenschmidt, of Le Genest, France, in *L'Echo des Mines et de la Metallurgie*, Oct. 21, 1907, gives data respecting the latest process employed at Mayenne. This consists of the fol-

lowing operations: (1) Smelting in a water-jacket furnace to produce a matte containing a little iron; (2) Treatment of this matte in a small converter to produce antimony oxide; (3) Reduction of this oxide with coal either in a small water-jacket furnace, a reverberatory furnace, or a crucible. In this method the converter discharges the sulphurous gas through a chimney, the oxide being caught in condensing chambers of adequate capacity.

The smelting of antimony ore in a water jacket is not new, having been employed at Lucette as early as July, 1898. This apparatus gives very satisfactory results with certain ores. The product of the water-jacket furnace, with the ore of Mayenne, contains from 92 to 96 per cent. antimony and 4 to 6 per cent. iron. To transfer this product into the converter to make oxide of antimony and to reduce the latter to metal is impracticable, because in the converter the metal becomes more and more ferruginous and forms lumps, thus retarding the oxidation. Anyway, it would be bad practice to convert metal containing 92 to 96 per cent. antimony into oxide, since in the starring furnaces the refining, indispensable in all cases, is effected by the addition of a small quantity of antimony sulphide. In speaking of the treatment of antimony ores by the converter process, the direct production of a final product is generally contemplated.

Smelting in Australia.—A brief description of the first smelting establishment for the reduction of antimony ores in Australia was given in the *Queensland Government Mining Journal* (August, 1907). The works are erected in Balmain, Sydney, and are owned by the Metals Smelting Company. A reverberatory furnace and the necessary supplementary plant were erected, and smelting was started during February, 1907. The company is now turning out about 24 tons of star antimony regulus per month, and shows an extraction of about 91 per cent. of the ore content. The refined regulus is produced in one operation and the starring of the metal is done subsequently in a separate furnace reserved for that purpose.

The works are very little affected by variations in price of antimony, as the smelter's margin remains the same whatever market prices may be. Moreover, the establishment of a smelter of this character is of great importance to the producers of antimony ore, as now they can sell large stocks of ore ranging from 20 to 30 per cent. antimony, which previously had been thrown aside in the process of picking ore which was done in order to bring its antimony content up to 50 per cent., the minimum grade that could be exported to Germany and England with any profit. The company has stationed a purchasing agent at Hillgrove, New South Wales, which is a district producing a considerable amount of antimony ore. Moreover, producers of ore in New Zealand and Queensland now find it advantageous to dispose of their product at Sydney.

Notes on Antimony Smelting.[1]

By G. Pautrat.

The object of these notes is to describe the plant and the processes now employed at a smelter in Mayenne, France, for the production of antimony metal and oxides from stibnite ore. The ore comes in two classes: Rich, carrying 50 to 60 per cent. antimony; and poor, consisting of quartz impregnated with stibnite, and carrying 10 to 20 per cent. metal. Only the rich ore is used for the recovery of metal, the poor being burned for the manufacture of oxides.

The prevalent method of antimony smelting, as practiced elsewhere, involves five steps: Liquation of the stibnite by simple fusion; oxidation of the crushed sulphide by roasting; reduction of the oxide with coal, to yield crude metal; refining by fusion with soda; treatment of volatile oxide caught escaping from the oxidation roast. This process is tedious and requires an excessive amount of labor, especially during the oxidizing

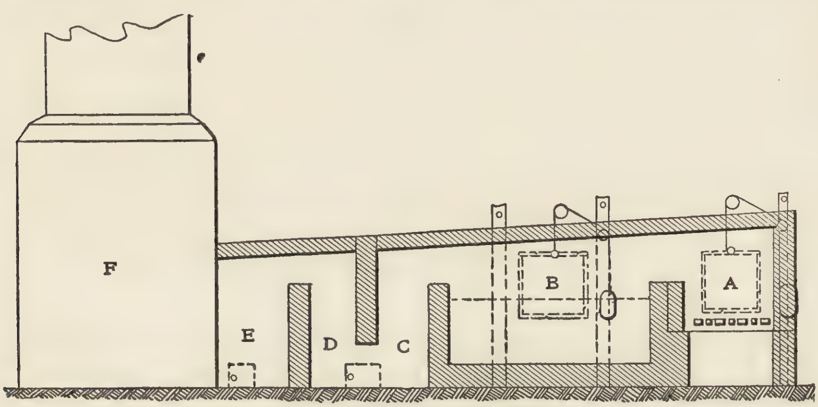


FIG. 1. ANTIMONY SMELTING FURNACE

roast, when the mass must be thoroughly rabbled so as to break the oxide crusts and permit complete desulphurization.

A much shorter process is employed at the smelter under discussion and, although it has been criticized as being incapable of yielding pure metal, it does actually operate to entire satisfaction. It involves only two steps: Fusion with scrap iron to produce crude metal; followed by a refining fusion, yielding metal with only 0.5 to 0.8 per cent. of impurities.

First Fusion.—The reverberatory furnace used in the first operation, Fig. 1, has a total length of 8 or 9 m., width of 1.25 m., and hight of 1.6 to 1.7 m. from hearth to roof. It comprises fire-box, *A*, hearth, *B*, and dust chambers, *C*, *D* and *E;* the utility of these chambers is questionable

[1]*Eng. and Min. Journ.*, Sept. 14, 1907.

since but little oxide is caught in them, at their prevailing high temperature.

Assuming the furnace empty, the process is begun by heating the hearth and charging a mixture of fusible slags and Solvay-process soda. A charge of the rich ore, mixed with enough soda to give a quick fusion, is then added, and the whole is stirred until melted. Silicious and other impurities form a slag which is skimmed off and thrown away.

When the surface of the bath is clear, scrap iron, usually in the form of detinned cans from which the lead solder also has been carefully removed, is added and stirred in, after which the door is closed and the temperature raised. The metallic antimony, robbed of its sulphur, settles underneath; the concurrently formed iron sulphide floats on top. This process is repeated until the level of the bath is raised conveniently near to the working door. The furnace is now ready for continous operation.

As soon as each addition of iron has undergone reaction the layer of iron sulphide is ladled off in an iron pot, care being taken not to dip too deeply, and poured into molds. The furnace is then allowed to regain its heat, when the crude metallic antimony is ladled out in the same way and cast into molds; between 300 and 350 kg. of metal are removed each time. This cycle of operations—introduction of ore, with stirring; skimming of slag; addition of iron, with stirring; pouring of iron sulphide; pouring of crude antimony—occupies about three hours. The usual life of a well-built hearth, working at this rate, is five to six months. The weights of the materials put into this furnace at each charge are about as follows: Crude ore in lumps, 450 kg.; washed fine ore, 150 kg.; slag from second fusion, 20 kg.; soda, 20 kg.; iron scraps, 240 kg.

The crude antimony, the principle impurity of which is iron, has a crystalline fracture. If the working of the charge in the hearth is not thorough, the crude metal may carry as much as 4 or 5 per cent. of iron, and will show a finely crystalline fracture. If the furnace work is done with care, the metal will carry not more than 1 or 2 per cent. of iron, and its fracture will be exceedingly coarse. A representative sample of crude antimony shows the composition: Fe, 2.75; S, 1.95; As, 0.54; Pb, 0.22; SiO_2, 0.12; Sb (by difference), 94.42 per cent.

Both of the waste products of this first fusion carry antimony, and at present no attempt is made to save it. The slags should not carry over 1 or 2 per cent. antimony. The amount of antimony carried off in the iron sulphide occasionally reaches 10 or 12 per cent., averaging 6 or 7 per cent. It occurs here in the sulphide state, or as globules of metal, the latter proving that the furnace was not sufficiently hot to permit clean separation. It would be a simple matter to crush the iron sulphide and then recover the metallic antimony, inclosed in it, by a jigging process.

Second Fusion.—The refining operation is done in a furnace like the

first, but smaller. The crude metal is remelted, taking care to avoid oxidation by covering the bath with a thin layer of fusible slag, composed mainly of soda. The melted bath is kept for a while at a brisk heat, to allow the impurities to rise, after which the door is opened and the impure oxides are skimmed. The workman judges the progress of the refining by pouring little ingots from time to time, carefully covered with slag; the end is denoted by a fern-leaf crystallization of the surface of the ingot. Before drawing off the refined metal, a protective covering is thrown over the bath, composed of antimony oxide (the unmarketable portion of the oxide furnace product), oxy-sulphides, soda, and a little coal. The antimony is then ladled out in iron pots, and cast in rectangular molds holding 20 to 21 kg. of metal. A small portion of the protective material is taken out of the furnace and put into each mold before the metal, so as to give a bright, crystalline surface to the ingot. This is supposed to advance its market price, but is not an accurate index to its purity. Analyses of two samples of such antimony are given: I. Sulphur, 0.069 per cent.; iron, 0.19; lead, trace; arsenic, 0.265; silica, trace; antimony, 99.45; total 99.974. II. Sulphur, 0.075; iron, 0.07; lead, 0.081; arsenic, trace; silica, 0.008; antimony, 99.69; total, 99.924.

As a rule, three charges are put through the refining furnace in 24 hours. The slags from this process frequently carry as much as 20 per cent. of antimony, and are put back into the first treatment as noted above.

Efficiency of the Process.—The amount of antimony put into the first furnace at one charge is about as follows, in kilograms: Lump ore, 450 kg. at 55 per cent. Sb, 247.5; fine ore, 150 kg. at 45 per cent. Sb, 67.5; slag, 20 kg. at 20 per cent. Sb, 4.0; total antimony in charge, 319.0.

From the 2552 kg. of antimony thus put into the furnace in eight charges about 2350 kg. of crude metal is obtained, a loss of 7.93 per cent. At 95 per cent. pure, this contains 2232 kg. antimony, from which, after refining about 2127 kg. of pure metal is obtained, a loss of 4.71 per cent. The actual efficiency of the process is thus 83.3 per cent., but some antimony is introduced in the oxides used in the refining furnace. Disregarding this factor, the efficiency would be about 79 or 80 per cent.

Double-Hearth Furnace.—It has been found possible to combine two hearths in a single furnace, the crude metal being reduced on one hearth, then, while still melted, transferred to and refined on the other. The individual processes are carried out exactly as already described for separate furnaces. A gas-burning jet was once tried in a double furnace of this sort, but its excessive heat ruined the structure.

Oxide Manufacture.—The apparatus for converting the low-grade ore into antimony oxide consists of a furnace, condensing chambers, and a chimney equipped with an upward projecting steam jet to provide draft.

The furnace consists of an inclined grate composed of square bars, *A*, Fig. 2. The ore is charged in at *B*; the oxides pass out at *C*, and the burnt ore is raked out through the bars at the bottom. The condensing chambers, *D*, *E*, and *F*, are cast-iron pipes, with V-elbows at the top and bottom. A sliding plate, *G*, in each bottom elbow allows the condensed oxides to be removed. The series of condensers passes at *N* into the bottom of the chimney, *H*.

A coke fire is started in the furnace; on this is spread a layer of ore in lumps of first size, then more coke, another layer of ore, and so on. The air reaching the furnace is ample to oxidize the ore, the residue averaging not more than 1 or 3 per cent. of antimony.

The product of the first condensers is rendered a gray color by admixture of coke dust, and contains oxides of iron and arsenic; it is nevertheless merchantable. The middle condensers yield a white oxide, analyzing 90

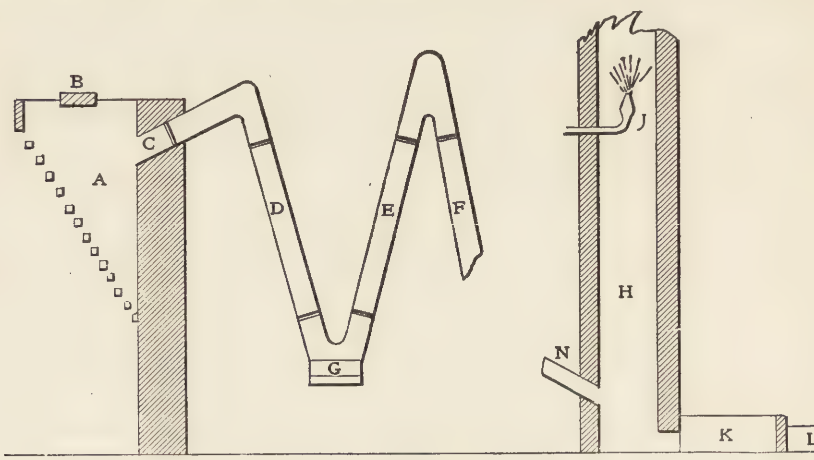


FIG. 2. ANTIMONY OXIDE PLANT

to 92 per cent. Sb_2O_3. In the chimney, the steam from the jet, *J*, condenses and moistens the remaining oxides, which run down the shaft and are collected periodically by decanting from the vessels, *K* and *L*. This oxide, after drying, is either sold or used in the antimony refining process, as described above.

The best white product will average about as follows: H_2O, 0.75; Sb_2O_3, 91.30; Fe_2O_3, 1.15; As_2O_3, 5.25; SiO_2, 1.60 per cent. The arsenious acid which always accompanies the antimony oxide, is the more volatile of the two, hence is found in greater abundance toward the chimney, as shown by the following determinations of arsenious acid present: Gray oxide, 4.52; white oxide, 5.25; wet oxide, 6.46 per cent.

The output of these oxide furnaces is not wholly satisfactory, owing to losses during charging, while the condensation of oxide also is far from

perfect. The plant under discussion has four such furnaces, which are served by two men.

The French Process for the Treatment of Antimony Ores.

By F. T. Havard.

At the small town of Brioude in the Haute Loire, France, are two antimony smelteries. One of them, retaining the old crucible and reverberatory process, has changed hands several times without apparently improving or being improved by the fortunes of the various masters. The history of the development of its sister plant is, therefore, all the more striking, for within a few years an entirely new process has been brought from an experimental stage to a commercially and technically successful basis by the orginality and energy of two men, M. Chatillon and L. Brunet. It was the former who encouraged the peasants of the Auvergnes to collect and cart to his house the loose antimony bearing stones which had lain on their farms for ages without awakening curiosity or suspicion of their wealth. In the first stages of treatment of this antimony sulphide ore, M. Chatillon decided that a process of volatilization offered the largest margin of profit with the least possible loss, and he proceeded to build the plant, which under his successor's care and development has become a highly successful antimony metal and paint producing works. The orginal idea was that the antimony should be volatilized in a shaft furnace, the draft of which was induced by fans placed at the end of a long and complicated flue and filter system wherein the fume of antimony oxide was collected. But the percentage of deposited oxide was unsatisfactory; accordingly several changes were made of the following character:

The whole flue system and fans are now sprayed by water which is recirculated. This enables the manager to reduce considerably the area of his flue system while obtaining better deposition. The shaft furnace is designed on the lines of a gas producer with a bottom of movable grate bars permitting the tapping of the partly fused slag, which, carrying all the precious metals in the orginal ore, is yet free of antimony. The excess heat of the shaft furnace is used to reduce the antimony oxide in a contiguous open hearth furnace, whence all gases and fume pass about the tubes of a boiler (which develops a great part of the power necessary to drive the fans and work the pumps for the whole plant) and thence to the flues, fans and stack.

Furthermore, the work of the reduction of the antimony oxide in the open hearth is so carefully performed that the quality of the resulting metal, known as French Star, can not be challenged and is sold at correspondingly high prices. Perhaps, however, the greatest strength of the French process lies in its applicability to all kinds of ore. Even rich

gold bearing ore carrying only a low percentage of antimony may be treated without risk, and, considering the large returning charges generally allowed by the miners of this class of ore, with much profit. As the plant is so compact it is evident that the cost of erection and equipment is very low in comparison to that required to build a smeltery for the treatment of other metals; in actual figures this should not amount to more than $40,000 for a plant capable of producing 1000 tons of regulus yearly. The cost of maintenance is moderate. The basic linings of shaft and open hearth furnace last fairly well, but the blades of the fans must be well protected to resist the sharp attack of the acid water.

It has been found expedient to work with two shaft furnaces, each of which can be shut off by a damper from the open hearth. Some advantage would be gained by building independent flues from the shaft furnace to the boiler and flue system, so that the open hearth might be cut off from the rest of the system, to permit on occasions the diversion of part of the shaft furnace gases direct to the boiler and flue system. These shaft furnaces are about 12 ft. high and 3½ ft. square, or 4 ft. in diameter if of round section. The hight of the inclined grate bars is from 2 ft. at the lower and 4 ft. at the higher end above the pit of water into which the slag is raked. The hight of the ore column is 4 to 5 ft. and the distance from the top of the column to the feed floor is about 4 ft. The side flue from the shaft furnace to the open hearth is 3 ft. in diameter. The lining of the shaft furnace is of basic aluminous brick, one and one-half brick thick. The inside area of the open hearth is 9x5 ft. The hight from floor to roof is 5 ft. The linings of the bath and the roof are made of chrome brick. The antimony slag eats even this—the most refractory brick obtainable. The open hearth is inclosed in a case of strong iron plate and held in a firm grip by vertically tees and horizontal tie bars. The flues which are square in section are partly of brick but principally of strong sheet iron. The fans are inclosed in wooden cases. Under the flues and fans are shallow concrete baths into which the water from the spraying of the fans and flues collects and cools in the course of circulation.

The shaft furnace works with a hot top; indeed, the whole ore column is kept at a lively temperature from top to bottom. The ore, ground fine and mixed with small coke, is fed through vertical pipes distributed evenly over the mouth of the furnace. In this way an even division of the charge over the ore column is insured. The percentage of coke used depends on the character of the ore, but is necessarily high, since no flux is used and excess heat above that needed for the smelting of the ore is required for the open hearth. As soon as the charged ore meets the column, volatilization commences and by the time the slag is raked from the bottom bars, the antimony has been entirely driven out. One man is required constantly to rake out the slag through the movable iron bars. This slag,

which is extremely viscous and at times only partly fused, contains all the gold of the original ore and is sold to smelteries at a fair refining charge, for partly by reason of the quantity of partially burned coke which it carries it is readily fusible when mixed with the ordinary charges on lead and copper furnaces.

The volatilized antimony is collected in the form of oxide in the above mentioned concrete baths, and dug out at regular periods. It is then dried on the iron flues and charged through hoppers into the open hearth along with ground coke or wood carbon and an agent, such as soda, which helps the slagging of impurities. The slag is drawn off intermittently. When the metal is tapped into molds a covering composed of the slag of the hearth furnace mixed with an alkali salt, which prevents any oxidation, is poured over the surface, so that the top of the ingot, when cool, shows the star formation so much desired by dealers. The slag drawn from the open hearth is necessarily rich and runs as high as 30 per cent. antimony. This slag is readily bought by alloy makers or lead smelteries at a fair price.

The fume from the shaft furnace along with any antimonious gases from the hearth passes with the rest of the gases by way of the boiler tubes, where deposition already commences, to the water sprayed flues and fans. The wash water with its contents of antimony oxide, solid carbon and some gangue matter (which has been drawn from the shaft furnace by the draft) is led into large shallow baths, where the solid matter settles and the clean surface water overflows to a reservoir whence it is pumped into circulation. The end gases which pass the fans are led to a long underground flue, where further deposition is effected, to the main stack. The percentage of antimony which finally escapes into the air is from 7 to 10 per cent. of the metal brought to the shaft furnace. After some time the circulating water becomes extremely acid owing to the sulphur dioxide dissolved in it. The sulphur dioxide is formed in the shaft furnace by the decomposition of the antimony sulphide of the original ore. As soon as the acidity of the wash water has reached a certain strength, the liquor acts on the antimony oxide in the fume and takes a great deal into solution. This action is cleverly used in producing a paint consisting of a mixture of antimony oxide and barium and calcium sulphite. On the addition of barium carbonate to hot liquor carrying the antimony oxide in solution, barium sulphite and antimony oxide are deposited. This precipitate, on being filtered, dried, milled, redried and sieved, furnishes a paint of excellent color and durability; and as it is sold at about the same price as the oxide its manufacture is a profitable business.

This ready formation of sulphite of barium or calcium by means of the works' sulphurous acid has led the French company to employ the reaction to a great extent in its preparation of the sulphide (golden antimony);

the oxysulphide, a brown pigment; and other antimony pigments. For information on this subject I refer readers to British patents No. 16,490 and 16,490A of 1905.

Valuation of Antimony Ore.

By Walter Renton Ingalls.

A list of the antimony minerals, together with their antimony content, is as follows, the sulphide and oxidized ores being separately designated:

	Per Cent.			Per Cent.
(S) Stibnite..........	71.80		(O) Senarmontite...	83.56
(S) Chalcostibite.....	48.90		(O) Valentinite.....	83.56
(S) Berthierite.......	57.00		(O) Cervantite.....	79.20
(S) Zinkenite.........	42.60		(S) Kermesite......	75.30
(O) Stiliconite........	74.90		(O) Volgerite.......	58.91
(S) Durfeldtite.......	30.52		(S) Livingstonite...	53.12
(O) Burunite.........	50.11		(S) Guejarite......	58.50

The most common ore of antimony is stibnite. Senarmontite, valentinite and cervantite are also commercial ores. The common standard for stibnite ore is 50 per cent. antimony.

Valuable information as to the classes of antimony ore that are marketable, the methods of smelting and the market values are to be found in an article by F. T. Havard in The Mineral Industry, Vol. XV, pp. 43-48, to which reference should be made. The information in the following paragraphs is additional.

The market for antimony ore is confined to so few smelters that the price is chiefly a matter of private contract, and sliding scales, such as are employed in the purchase of other ores, are but little, or not at all, used as a basis for antimony ore. However, Mr. Havard states that in May, 1906, dealers were still offering ores to smelters on the old formula of $V = 0.009\ T\ (P—330)$, in which V is the value of the ore in francs per 1000 kg., T the units of antimony in the ore, and P the price of antimony in francs. On this basis, an ore assaying 50 per cent. antimony, with the metal at £60 per 2240 lb., or 1485 francs per 1000 kg., would be worth 519.75 francs per metric ton, or roughly £21 per 2240 lb., or a trifle more than 8s. per unit. This old formula is converted into terms of U. S. currency and short tons by substituting $58 (as a close approximation) for the deduction of 330 francs. Then if antimony be worth £60 per 2240 lb. = $260 per 2000 lb., the value of 2000 lb. of ore assaying 50 per cent. antimony would be $90.90, or $1.82 per unit.

I have explained this old sliding scale to show how the prices now paid by the buyers of antimony ore compare with it. The meaning of the formula is, of course, that the smelter pays for 90 per cent. of the antimony

content of the ore at the market price for the metal less 330 francs per ton of metal to allow for the cost of freight and smelting and the necessary profit. Mr. Havard states that the smelters using the English process estimate their working costs, including loss, at 200 @250s. per ton of metal produced. He states that the working costs by the French process amount to about 200 francs per ton of metal produced from ore containing 50 per cent. antimony. Of course as in all such statements it is doubtful as to precisely what is meant, i. e., whether the stated cost represents (1) direct operating without general expense, or (2) including general expense, or (3) direct operating, including general expense, plus amortization charges The general meaning is either the first or second definition.

American buyers commonly make a basis of $1 per unit (22.4 ℔.) when the metal is worth 8c. per ℔., for ore containing about 50 per cent. antimony, with a variation of one-third of the difference in the market price for the metal per 2240 ℔. up or down. Thus, with the metal at 8c. per ℔., or $179.20 per 2240 ℔., the value of an ore containing 52 per cent. antimony, is $52 per ton of 2240 ℔. If the price for the metal is 10c. per ℔., or $224 per 2240 ℔., the addition of ⅓ ($224—$179.20) = $14.93 is made, giving the ore a value of $66.93. Similarly at 6c. for the metal the value of the ore is $52—$14.93 = $37.07. At 13c. for the metal this ore would be worth $89.33 per ton. The formula previously discussed would give $47.74 per 2000 ℔. at 8c. and $94.54 at 13c.

The present American basis price of $1 per unit may become $1.05 or $1.10 if the ore be of particularly high grade, especially free from objectionable impurities and otherwise desirable. The determination of price for the metal is made in the usual manner, i. e., stipulations by agreement that it shall be the average of the month of arrival of the ore, or the average of the week preceding and the week following arrival, etc.

The subject of payment for the gold and silver content of an antimony ore is one upon which smelters are apt to be reticent. European smelters pay for the precious metals in some cases; in some cases they do not. Some smelters are able to recover the precious metals; others are not. The fact that no specific credit is made for the precious metals does not imply, however, that they are not being paid for, because they may be receiving consideration in the form of a higher price per unit of antimony than could be allowed if some portion of the gold and silver were not to be extracted profitably. In any case the allowance for the precious metals is a matter of private negotiation and is so variable as to be incapable of general quotation.

In fact the entire discussion of the subject of the valuation of antimony ore can be only of a general character. There are few buyers of the ore, and when they are apathetic there is practically no market and the value of the ore is relatively much less than when there is a demand for it. The

antimony business is one in which great mystery has prevailed with respect to the technical part. Of course this has stood greatly in the way of progress. The old methods of smelting, which still are largely practised, are crude and costly. The modern French process of volatilization is considerably more economical and affords a high percentage of extraction of gold and silver, which in the older English process of precipitation-smelting are recovered only with difficulty and incompleteness. The increased value of antimonial lead during recent years, which now sells for practically the full value of each of the constituent metals has made it profitable to produce that alloy by the combined smelting of lead and antimony ore (which of course are preferably both free from gold and silver), whereas formerly it was an undesirable by-product in the process of refining work-lead.

ARSENIC.

By Reginald Meeks.

The production of white arsenic in the United States in 1907 increased materially over that of 1906. The chief producers continued to be the Washoe and Everett smelting works. The Mineral Creek Mining and Smelting Company, which made a small production in 1906, made no production in 1907, the attention of the company being occupied with construction work and development of the orebody. The United States Arsenic Mines Company, of Pittsburg, Penn., discontinued operations at its mine at Rewald, Va., more than a year ago. The Summit Mining Company, of Wellsville, N. Y., has not as yet produced any arsenic for market. Its mine, in the Darrington district of Washington, is not being operated at present. Details of the American production and consumption of white arsenic in 1907 are given in the accompanying table. Besides the production of white arsenic there was an output of arsenic ore from a mine in New York, which was exported.

STATISTICS OF WHITE ARSENIC OF THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.	Pounds.	Value.
1897	....	....	....	7,242,004	$352,284	$0.05	7,242,004	$352,284
1898	....	....	....	8,686,681	370,347	0.04¼	8,686,681	370,347
1899	....	....	....	9,040,871	386,791	0.04¼	9,040,871	386,791
1900	....	....	....	5,765,559	265,500	0.04¾	5,765,559	265,500
1901	600,000	$18,000	$0.03	6,989,668	316,525	0.04½	7,589,668	334,525
1902	2,706,000	81,180	0.03	6,110,898	280,055	0.04½	8,816,898	361,235
1903	1,222,000	36,691	0.03	7,146,362	256,097	0.03½	8,368,362	292,788
1904	996,456	29,504	0.03	6,391,566	226,481	0.03½	7,388,022	255,985
1905	1,545,400	50,225	0.03¼	6,444,083	219,198	0.03⅓	7,989,483	269,423
1906	1,663,000	83,150	0.05	7,639,507	336,609	0.04½	9,302,507	419,759
1907	2,020,000	101,000	0.05	9,922,870	553,440	0.05½	11,942,870	654,440

Market and Prices.—In the New York market 1907 opened with high prices and a great scarcity of white arsenic. In February and March the prevailing quotation was 7¾c. for large quantities and small lots sold as high as 8¼@9c. In April large amounts of arsenic were brought into the United States from Spain, earlier shipments having been curtailed because of difficulties in shipping from the mines. The augmented supply caused a reaction in price to 7c., but later the market recovered and remained steady at 7¼c. per lb. In spite of the high price certain dealers believed that there would be a further advance and a heavy demand for

future delivery did indeed raise the price to 7⅜@7½c. in mid-summer. This buying was purely speculative, however, and the error in judgment was clearly demonstrated at the close of the year.

From August to December the market was heavy and prices fell off gradually to 6¾c. per lb. About the middle of December a violent break occurred, the price dropping to 5¾c., and the market closed weak. The average price for the year was approximately 6.975c., which represents the selling price of New York dealers in reasonably large lots. The corresponding average in 1906 was 6.448c. The producers disposing of their large outputs under contract realized about 5c. per lb. each year.

The depressing influences in the second half of 1907 were: (1) large speculative stocks bought in August and unloaded at the close of the year upon an already weak market; (2) heavy imports; (3) increased production abroad with consequent increase in competition; (4) recession in business and the general financial stringency. The situation abroad at the close of 1907 showed a tendency to weaken, although prices were slightly higher than here. Many countries recently entered the list of producers and offers were coming in freely. High-grade arsenic is now coming from France, Spain, Belgium, England and Australia, while a little of inferior quality and low grade comes from Italy.

Arsenic Ore.

Besides the production of white arsenic above reported, there was also produced in the United States in 1907 about 800 tons of mispickel ore, valued at $12,000, which was exported. This was produced in New York and represents the inauguration of a new industry, which is described in the following paragraphs:

New York (By E. K. Judd).—The only mine in the east that is now producing arsenic ore and, in fact, probably the only mine in the United States that is now working for its arsenic value alone, is situated near Carmel, Putnam county, N. Y. This town is 50 miles north of New York City and is on the Putnam division of the New York Central Railroad. The mine belongs to the Putnam County Mining Corporation, and the entire output is exported. The ore has to be hauled three miles, over a good highway, to the railroad.

The ore occurs as a band, composed of stringers of arsenopyrite, having an aggregate width of 12 to 20 ft., including the intermediate barren parts; the country rock is gneiss. The ore, as it is hoisted from the mine, averages 10 or 12 per cent. arsenic. There are two such ore zones; these intersect at an angle of 60 deg. One of these has been developed to a considerable extent, while the other, not owned by the company, has been only prospected. The intersection of the veins lies at the foot of a hill, affording

an excellent opportunity for adit development, of which, however, advantage has not been taken.

The present development on the company's property consists of a shaft, 100 ft. deep, from the bottom of which a drift runs for about 75 ft. along the zone of mineralization, which, at that depth, is said to have a width of 12 ft. No work has been done in the mine for some time, and it is now flooded, but material on the dump has kept the dressing works in operation until very recently.[1]

To concentrate the ore, it is first passed through a jaw crusher and then through rolls. The crushed ore is then treated in eight Joplin hand jigs. The screens of these jigs are made of cast-iron sectional plates, perforated with ¼-in. slots, very much like certain kinds of furnace grate bars. The only recovery of arsenopyrite is made from the hutch, and all the material remaining in the screen is thrown away. The capacity of one of these jigs is 4½ tons of crude or 1½ tons of concentrated ore per day. Labor for each jig costs $2 per day. The concentrate averages 25 per cent. arsenic.

This inefficient concentrating method has probably been justified by the fact that the market for the ore is not altogether regular, so that the whole plant not infrequently has to be closed down for weeks. It would seem, however, that a steam driven plant should not only make a better saving of ore at a much less cost, but it would also involve little or no more loss in fixed charges, during the period of idleness, than the present plant. It is intimated that, after certain changes in management, the remaining ore, as to the existence of which there seems little doubt, may be mined and treated in a more economical way.

Arsenic Production in Foreign Countries.

Among the most important sources of arsenic are the silver mines of Saxony, the tin mines of England, the mispickel mines of Spain, and the mispickel mines in the Province of Ontario, Canada. The United States consumes a large proportion of the world's production of metallic arsenic, white arsenic, orpiment, and red sulphide of arsenic. Spain in 1906 produced 1114 tons of white arsenic. Germany produces the largest quantity of metallic arsenic and arsenious acid. England has fallen far behind, although it held the first place in 1902. In France there are three mispickel mines: two in the department of Aude and one in Puy-de-Dome. The production of these three mines in 1906 amounted to 6534 tons. Portugal contributes from 1300 to 1500 tons of white arsenic per annum.

Canada.—White arsenic was produced in Canada in 1907 by the Canadian Copper Company, at Copper Cliff, and by the Deloro Mining and Reduction Company, at Deloro, Ont. The production of the latter company was

[1] *Eng. and Min. Journ.*, Feb. 1, 1908.

small, its plant not having been put in operation until December. During 1907 the whole plant was remodeled and from now on the company will make a regular output of white arsenic, the same as formerly. The production of the Canadian Copper Company was largely from the treatment of Cobalt ores. A deposit of arsenical pyrites was reported in the vicinity of Temagami station, but the arsenic content proved too low for economical extraction.

Portugal.—The arsenopyrite mines of Pintor, in the parish of Nogueira do Cravo, in Aveiro, are, at present, the only mines in Portugal being worked for arsenic. The operations are confined exclusively to the production of arsenious acid, although the ore contains small quantities of copper, silver and gold. The veins, which are numerous and in some cases attain a thickness of 5 m., are composed of quartz and arsenopyrite. They occur in Archean schists, which are intersected by occasional masses of granite. The Anglo-Peninsula Mining Company has developed the mines extensively in order to increase production. Arsenious acid is produced in different furnaces according to the nature of the ore. The coarse material (3 to 4 cm. in size) is treated in 26 small kilns, about 4 m. high, 2.4 m. long and 2.4 m. wide. Coal is used for fuel, the volume required being about one-tenth that of the ore treated. The fumes are conducted into a large chamber 300 m. long. The product is refined in reverberatory furnaces, shipped to England and reshipped to Australia.

Spain.—The Carballino Gold and Arsenic Mines, Ltd., is an English company, recently organized, with headquarters in London. The capital stock consists of 140,000 shares of £1 each. Of the capital stock £80,000 was for the purchase of the mine and £60,000 for working capital. The mine is situated in the Province of Orense in northwest Spain. The ore is said to contain considerable gold, and it is believed that the veins become richer with depth. It is proposed to concentrate the ore at the mine and ship the product to Gonfreville. A mill was completed in October, 1907.

WORLD'S PRODUCTION OF ARSENIC.

(In metric tons.)

Year.	Canada. (*a*)	Germany. (*b*)	Italy.	Japan.	Portugal.	Spain. (*b*)	United Kingdom. (*a*)	United States. (*a*)	France. (*d*)
1896	*Nil*	2,632	320	6	....	271	3,674	....	....
1897	*Nil*	2,987	200	13	524	244	4,232	....	....
1898	*Nil*	2,677	215	7	751	111	4,241	....	....
1899	52	2,423	304	5	1,083	101	3,890	....	2,600
1900	275	2,414	120	5	1,031	150	4,146	....	4,705
1901	630	2,549	....	10	527	120	3,416	272	7,491
1902	726	2,828	....	12	736	71	2,165	1,226	5,372
1903	233	2,768	50	6	698	1,088	916	554	6,658
1904	66	2,829	80	4	1,370	400	992	452	3,117
1905	*Nil.*	2,535	....	8	1,562	1,140	1,552	701	3,627
1906	*Nil.*	3,052	....	(*c*)	1,322	1,114	1.625	754	6,534
1907	317	(*c*)	....	(*c*)	(*c*)	(*c*)	(*c*)	916	(*c*)

(*a*) Arsenious acid. (*b*) Oxide, sulphide, etc. (*c*) Not yet available. (*d*) Ore.

Progress in Technology.

More and more attention is being devoted to the recovery of arsenic as a by-product and it is becoming realized that the possibilities of increasing the production from formerly wasted material are very large. Many lead and copper ores contain appreciable percentages of arsenic, e.g. the copper ores of Butte, Mont., which are the source of the white arsenic production of the Washoe smelter. In the case of lead ore the old mines at Eureka, Nev., were noteworthy for their arsenic content. These mines produced about 200,000 tons of pig lead, and in the course of its production there was made speiss which now lies on the Richmond and Eureka dumps at Eureka, Nev., to the amount of 130,000 to 200,000 tons, as estimated by a former official of one of the companies. This speiss is estimated to average 30 per cent. arsenic, 3 per cent. lead, 2 per cent. copper, 2 to 3 oz. silver per ton, and \$3 to \$4 gold. It is a great possible source of arsenic in the future.

Arsenic from Lead Smelting.—In the smelting of lead ore arsenic goes partly into speiss and partly escapes with the flue dust. If the latter be collected by bag-filtration the arsenic fume is also recovered and the burned dust and fume from the bag-house finally becomes so rich in arsenic that a special disposition has to be made of it. In 1907 bag-houses were installed at the United States and Murray smelters at Salt Lake City, especially as means to ameliorate the smoke nuisance (see article on "Lead"). It is reported that the concentrated arsenical dust from the Murray bag-house will be shipped to Everett, Wash., for extraction of the arsenic. The United States bag-house makes 10 tons of burned fume per day, which is said to contain 30 to 35 per cent. of arsenic. This high percentage is due doubtless to the smelting of a large amount of ore from Eureka, Nev.

Production of Arsenic as a By-product in Sulphuric-acid Manufacture.—Until recently the arsenic removed from sulphuric acid has been wasted, owing to its production in the form of sulphide, which is not a commercial commodity. During the last year or two the United Alkali Company has conducted experiments with a new process by means of which the arsenic is recovered as arsenious acid. The process promises to be a commercial success, and in all probability will bring an important new supply of arsenic on the market, as well as make it possible to use highly arsenical ores profitably in sulphuric-acid manufacture.[1] The process is described in a series of British patent specifications, the most important of which is No. 5151 of 1906. That the process is a success is shown by the fact that five plants are already in operation, and that arrangements are being made for building several more.

[1] *Eng. and Min. Journ.*, June 20, 1907.

According to this process, the arsenical sulphuric acid, as it flows from the Glover tower, is first brought into contact with a reducing agent such as charcoal, in order to bring the arsenic to the arsenious state. It is then brought into contact with dry hydrochloric-acid gas, the result being that the arsenic is converted into liquid arsenious chloride. This chloride is an oily liquid and a good deal of it can be separated from the sulphuric acid by settlement. The sulphuric acid drawn off from the settling tanks still contains arsenious chloride. To remove the latter, air is blown through the acid. The chloride comes off as vapor, and is taken to a scrubbing tower. Here it comes in contact with water, with the result that arsenious acid and hydrochloric acid are formed. The hydrochloric acid is used over again, and the arsenious acid is collected as a commercial product. Very often, however, the arsenic contains selenium. If so, some of the arsenious chloride, which had previously separated as an oily liquid, is added to redissolve the whole of the precipitated oxide, and then, on addition of water, the selenium is found to be precipitated.

The process is naturally one which requires very careful attention, owing to the existence of arsenic as a volatile compound. The manufacturers and the alkali inspectors, however, speak well of the process, and it may develop into a standard method of recovering arsenic.

Uses.—Arsenic finds its greatest use in the manufacture of paris green, but it is also used for the following purposes: as an insecticide and preservative in taxidermy; for sheep and cattle dips; arsenic soap; wood preservatives; weed destroyers; in the manufacture of glass and enamel; in dyeing and for calico printing and as a constituent of dyes themselves. A small amount is used in the drug trade.

ASBESTOS.

Although there was increased activity in the development of asbestos properties in 1907, the production of the United States remains insignificant. The statistics of production and imports are given in the following table:

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.		
	Short Tons.	Value.	Value per Ton.	Manufactured.	Unmanufactured.	Total.
1897	840	$ 12,950	$15.42	$10,570	$264,220	$ 274,290
1898	885	13,425	15.17	12,899	287,636	300,535
1899	912	13,860	15.20	8,946	303,119	312,068
1900	1,100	16,500	15.00	24,155	331,796	355,951
1901	747	13,498	18.08	24,741	667,087	691,828
1902	1,010	12,400	12.27	33,313	729,421	762,734
1903	(a) 887	(a) 16,760	(a) 18.90	32,058	657,269	689,327
1904	(a) 1,480	(a) 25,740	(a) 17.40	51,290	700,572	751,862
1905	3,100	126,300	40.74	70,117	776,362	846,479
1906	(a) 1,695	(a) 28,565	(a) 16.85	96,162	1,010,453	1,106,615
1907	....	.....	....	200,371	1,104,109	1,304,480

(a) Statistics of the United States Geological Survey.

Of the two classes of asbestos of commerce, the tremolitic or amphibole variety is by far the more abundant; however, on account of its shortness of fiber and lack of strength, the demand for it is not great. Chrysotile asbestos has a long, tenacious fiber, is well adapted to spinning into rope and weaving into cloth, and is much more valuable.

Prices and Market Conditions.—The prices of the different grades of asbestos vary greatly, depending on the length and quality of the fiber. In 1907, No. 1 crude brought from $300 to $350 per ton; No. 2, from $175 to $250; and No. 3, from $100 to $150. Poorer grades were sold as low as $37.50 per ton.

Occurrence in the United States.

California.—Prospecting for asbestos was actively prosecuted during 1907, the large masses of serpentine on Klamath mountain and in portions of the Sierra Nevada, affording the most promising fields. Several new discoveries were reported from the western border of the great belt of serpentine, which extends from Plumas county south through Sierra, Nevada, Placer, Amador, and Calaveras counties. During the summer the Twin Peaks Asbestos Company was organized to work deposits near the city of San Luis Obispo. W. S. Haworth and T. E. Morgan, filed on several deposits of asbestos near Green Valley, Placer county.

On one of these considerable work was done and a good quality of the mineral was found. Asbestos of the chrysotile variety was reported by F. T. Smith and J. T. Dillon near Washington, Nevada county.

Idaho.—Early in 1907 the Victor Mining Company was incorporated to work an asbestos deposit on Canyon creek, near Wallace. No production was reported from this source during 1907.

Michigan.—Asbestos was found near Republic, and in the serpentine rocks north of Ishpeming, Marquette iron range, but the deposits have not been opened, although the mineral is said to be of commercial grade.

North Carolina.—In November, 1907, the Carolina Asbestos Manufacturing Company, of Greensboro, was organized, and it is now constructing a plant at Greensboro, which is intended to begin operations at an early date. The raw material will come from North Carolina and elsewhere.

Vermont.—The Lowell Lumber and Asbestos Company, of Lowell, mined and shipped crude asbestos during 1907. The construction of a fiberizing plant was begun in February, 1907, the work being completed and operations started on Jan. 15, 1908.

Wyoming (By H. C. Beeler).—During 1907 deposits of chrysotile asbestos, situated on Casper mountain, in central Natrona county, seven miles south of the town of Casper, attracted attention, and several companies were organized to operate in this locality. The United States Asbestos Mining and Fiberizing Company was the first in the field, and considerable surface work was done to open up and outline the extent of the deposits in its holdings The asbestos occurs in serpentine; the average fiber is of good length, as far as shown by present work, spinning fiber up to 2 and 3 in. being not uncommon. So far this fiber has only been roughly treated, but enough has been shown to indicate its commercial importance. The work outlined for 1908 by the various companies is sufficient to determine accurately the commercial importance of the deposits. Casper mountain is a low uplift, lying on the south side of the Platte valley, and a tramway of seven miles from the top of the mountain to the Chicago & Northwestern Railway at Casper is all that is necessary to put the product on the cars at a minimum expense. Reports of remoteness from transportation of this field have been grossly exaggerated; the general economic features of the country are favorable for production at a low cost.

Asbestos in Foreign Countries.

Africa.—The best grade of asbestos comes from the Carolina district, South Africa, where several new companies started operations during 1907. The Angolo-Swiss Asbestos Company, which entered this field early in the autum of 1906, began to mine asbestos early in 1907. Up

to June 30, shipments amounted to 60 tons, and the monthly production since that time has been about 20 tons. Recent developments have proved the existence of large quantities of asbestos of a fairly good grade in the Carolina district, the richest deposits being situated about 50 miles to the southeast of Machadadorp station, on the railway running between Delagoa bay and Pretoria.

Numerous deposits of asbestos are also found in West Griqualand, Cape Colony, on the borders of the Orange river. Here the mineral is fibrous and of exceptional strength, but contains relatively so little lime and magnesia and so much iron that it crumbles and weakens when exposed to heat, and therefore is not particularly valuable.

Australia.—It is reported that English capitalists are preparing to to operate recently discovered asbestos deposits in Western Australia, in the vicinity of Perth. The quality of the mineral is said to be equal to that found in the Canadian mines.

Canada.—This country still leads the world in the production of long-fiber (chrysotile) asbestos, the principal deposits of which are in the Thetford-Black Lake district, and also in the vicinity of Templeton, Danville and East Broughton, in the province of Quebec. The special features of interest during 1907 were an increased output, higher prices realized for the product, further consolidation of mining interests, the introduction of electric power by the Shawenegan Power Company, and the continued successful working of the East Broughton district, which is chiefly a fiber producer.

In the Black Lake district the American Asbestos Company operated two quarries. The mill produces from 20 to 25 tons of asbestos per day. The entire plant is operated by electric current. The company has acquired the Montreal & Glasgow and the Manhattan properties, which have not been operated for more than 10 years, and has about 60 men at work developing. At present the deposits yield considerable crude asbestos No. 1. The foundations for the new mill of the Dominion Asbestos Company are completed. The mill will treat about 300 tons of rock per day, yielding about 20 tons of mill fiber. The company recently bought about 275 acres of asbestos ground from the Standard Asbestos Company. Some of the parts of the dismantled mill on the Montreal & Glasgow property will be used in the construction of a new milling plant at East Broughton. This was the first mill erected for the purpose of mechanically separating asbestos from the gangue. The Standard Asbestos Company installed another Cyclone pulverizer. The daily output of mill fiber is from 16 to 20 tons. The quarry which is near the mill was extended westward. About 80 men are at work. Two new mills were erected about six miles from Coleraine station, one by Boston people and the other by manufacturers of Providence, R. I.

STATISTICS OF ASBESTOS IN CANADA. (a)
(In tons of 2000 lb.)

Year (b)	Production. (b)				Exports (c)		Imports (d)
	Asbestos.		Asbestic.				
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Value.
1897	13,202	$ 399,528	17,240	$45,840	10,969	$ 510,916	$ 19,032
1898	16,124	475,131	7,661	16,066	18,424	510,368	26,389
1899	17,790	468,635	7,746	17,214	14,520	453,176	32,607
1900	21,621	729,886	7,520	18,545	18,164	490,900	43,455
1901	32,892	1,248,645	7,325	11,114	26,715	864,573	50,829
1902	30,219	1,126,688	10,197	21,631	33,072	1,131,202	52,464
1903	31,129	915,888	10,548	13,869	30,661	955,405	75,465
1904	35,635	1,167,238	13,011	13,006	34,636	984,836	83,827
1905	50,670	1,486,359	17,594	16,900	41,127	1,311,524	116,836
1906	59,283	1,970,878	20,127	17,230	59,864	1,689,257	138,000
1907	62,018	2,482,984	28,519	22,059	56,753	1,669,299	200,371

(a) From *Annual Reports* of the Geological Survey of Canada, and the *Statistical Year Book* of Canada. (b) Production is given for calendar year; exports and imports are for fiscal years ending June 30. (c) Mainly crude asbestos. (d) Manufactured articles entirely.

Cyprus.—The Cyprian Mining Company, Ltd., has been formed to exploit the asbestos deposits of Trodos, Cyprus. These deposits are said to be extensive and the company has secured important concessions and mining rights from the Cyprian Government. Work has been started and it is expected that production will begin early in 1908.

India.—A promising asbestos field was discovered in the northwestern part of India and it is reported that several veins were uncovered, some of which are as much as 4 ft. in width. This field, which is an extensive one, is now being exploited.

Italy.—Asbestos is abundant in the Italian Alps and Pyrenees, but so many obstacles are encountered in mining that the deposits have not been exploited to any great extent. An analysis of an Italian asbestos gave the following results: silica, 40 per cent; magnesia, 36 per cent; alumina, 5 per cent; oxide of iron, 4 per cent; soda, 2 per cent; moisture, 4 per cent; organic matter, 9 per cent.

Philippine Islands.—Extensive deposits of both tremolitic and chrysotile asbestos occur in the Dalumat and Baruyen schist areas, in the northern part of the province of Illocos Norte, but no work has as yet been done on them.

Russia.—Several important asbestos discoveries were made in this country. In the southern part of the Altai mountains, where much time and money have been spent of late years in exploration, deposits of long-fiber asbestos were uncovered, and it is expected that in a few years the output of this district will exceed that of the Urals. In the Minusinsk district, in the southern part of the Yenisei river country, and in the province of Perm, asbestos deposits, which have not been exploited to any great extent, were reported. In the Ural region occurrences of asbestos

are centered in the Kamensky and the Berezovsky forest districts. Here the demand is greater than the supply. The fiber is coarser and not as long as that of the Canadian asbestos, but more of the mineral is to be found in a given area of ground. No machinery or explosives are used in mining, and milling methods are antiquated.

Asbestos Mining and Milling in Canada.

In no other country has the mining of asbestos, and especially its preparation for the market, attained such proportions as in the Province of Quebec. There is little new to be said respecting the methods of mining for, with few exceptions, the mineral is quarried from open cuts or pits. As in all quarrying operations it is only necessary to conduct the work on a safe and economical basis. In many cases it is necessary to drill dry holes in order to prevent plugging with wet fiber.

Milling.—Although the practice varies slightly in different plants, the following is the general method for preparing the mineral for market: The broken rock is cobbed and the crude asbestos, with fiber at least $\frac{3}{8}$ in. in length, is sorted from the material of shorter fiber. The crude asbestos is hammered and screened, to liberate any adhering rock particles, and further classified according to the length of the fiber. All rock and the tailings from the cobbing go to the fiberizing plant, where it is first thoroughly dried. For this purpose either rotary dryers or steam heaters are employed. In the latter case exhaust steam is used to supply the heat. After preliminary crushing by means of breakers and rolls, the most important part of the process, the complete separation of the asbestos from rock particles, is begun. This part of the work is done by beaters and cyclones. A beater consists chiefly of a large cylindrical trommel of boiler plates, in which strong cutting knives are attached to a speedily rotating center axle. This apparatus is inclined so that the material travels from the feed to the discharge end. The cyclone, which has only recently been brought to a state of perfection, consists essentially of two beaters, shaped like screw propellers, running in opposite directions and making as high as 2500 r p.m. They are encased in a cast-iron chamber and all material reduced to about the size of a hazel nut or smaller is removed by air suction and falls upon a shaking screen, whence the liberated fiber is removed by an exhaust fan. Stony matter which has been shaken through the screen is either passed through the cyclone again, or milled to an impalpable powder and used for making plaster, cement and similar materials. The fiberized asbestos taken from the screen is blown into settling chambers to be classified by rotating screens. In most plants picking belts are employed where, after the preliminary crushing, barren rock or long-fiber material, which has not been exposed in the cobbing, is removed.

Use of Asbestos.

Asbestos is used for many and varied purposes. On account of its fire-resisting qualities it is especially adaptable for use in theater curtains, firemen's and electrician's gloves and garments, partition walls, asbestos paint, plaster, flooring, ceiling, wall decorations, bricks, tiles, slabs, and board. Excellent shingles are made with a mixture of asbestos and portland cement. Asbestos is also used in machinery packing, insulation of steam pipes, making grates for gas fires, filtering corrosive acids, and as a basis for insulations which must withstand somewhat elevated temperatures. Long-fiber asbestos is especially adaptable for manufacture into yarn, cloth, asbestos paper, and cardboard; the short-fiber material is used as a filler for paper, rubber, etc. The very short fiber is ground to a pulp and made into millboard, composition and mattresses. Recently asbestos wood has been made from "sand" waste.

ASPHALTUM.

BY EDWARD K. JUDD.

The most noteworthy events in the asphalt industry of the United States during 1907 were: An increase of over 50 per cent. in the already prodigious output of oil residue in California; the inauguration of a third great enterprise in the Texas field devoted to the manufacture and refining of oil asphalt; the installation of the first plant in Kansas for the extraction of asphalt from nearby crude oils; and the growth of interest in the asphalt and bituminous rock deposits of Oklahoma and Indian Territory.

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(Tons of 2000 lb.)

States.	1905			1906 (a)		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous Sandstone						
California	22,500	$63,000	$2.80	20,418	$47,427	$2.32
Kentucky	7,530	34,885	4.63	1,629	7,330	4.50
Indian Territory	(e)5,000	11,500	2.30	738	2,029	2.75
Arkansas	(e)1,500	7,500	5.00	900	5,400	6.00
Georgia (f)	....	....	....	400	8,500	2.13
Total	36,530	$116,885	$3.20	24,085	$70,686	$2.93
Asphaltic Limestone	6,029	42,000	6.96	*Nil.*	....	....
Asphaltum (b)						
California	57,687	545,503	9.46	71,539	711,150	9.94
Indian Territory (c)	*Nil.*	....	....	*Nil.*	....	....
Missouri	(e)1,000	17,500	17.50	*Nil.*	....	....
Texas	113,500	851,250	7.50	24,993	307,952	12.32
Total Asphaltum	172,187	$1,414,253	$8.20	96,532	$1,019,102	$10.55
Gilsonite (d)						
Utah	10,516	110,144	10.47	12,947	159,600	12.33
Indian Territory (c)	(e)1,000	25,000	25.00	1,952	16,432	8.42
Mastic						
California	*Nil.*	....	....	*Nil.*	....	....
Kentucky	*Nil.*	....	....	2,543	24,158	9.50
Pennsylvania	2,000	18,000	9.00	*Nil.*	....	....

(a) From the Mineral Resources of the United States. (b) Includes hard and refined, or gum, liquid or maltha, and oil residues. (c) Includes production of Oklahoma Territory. (d) Includes gilsonite, elaterite and grahamite. (e) Estimated. (f) First reported separately in 1906.

It would appear that the merits of oil asphalt, as a paving material, are at last acknowledged. Oil asphalts have long been, and are still, used for tempering the more brittle imported asphalts, but with improvements in the methods of producing, refining and mixing, and aided by vigorous campaigns of education on the subject, the residual asphalts derived from California and Texas petroleums are now employed exten-

sively, without the admixture of other asphalts, in the preparation of durable pavements. The United States, so far as now known, possesses no large deposits of natural asphalt comparable to those of Venezuela and Trinidad, and it is due to this fact, no doubt, that the oil asphalt industry has reached its present large proportions. Bituminous rock is quarried at a number of places in California, Arkansas, Indian Territory and Kentucky, and is used for road building, but in nearly every case the usefulness of the rock is limited to within a short radius of the producing point.

According to the fifth annual report of the General Asphalt Company, covering the year ended April 30, 1908, the most marked advances in that company's business have occurred in the manufacture and sale of asphaltic products, such as roofings, mastic, paints, etc. The manufacture of roofings by that company began in a small way in 1903, under the trade name, "Genasco"; sales of such materials in 1906 totaled $953,498, and in 1907 they were worth $1,684,884, an increase of 66 per cent. This large expansion has necessitated corresponding increases in the size of the company's plants which, during 1907, were worked to their fullest capacity. A large part of the increased earnings of the company in 1907 came from this branch of the business. The company owns and operates at Maurer, N. J., the largest factory of this kind in the world. The plant covers 25 acres of ground, is located upon the water-front of New York bay, and is equipped with its own docks and railway terminals. Connected therewith is an extensive laboratory and staff of experts whose time is largely devoted to analyzing and testing asphalts and asphaltic products and in the study and development of new manufactures. Smaller refineries and manufactories are located at Madison, Ill.; Los Angeles, Cal.; Trinidad, B. W. I., and in Venezuela.

Sales of refined asphalt to independent paving contractors have also shown a decided growth. On the basis of crude asphalt, the amount sold and consumed in 1907 was 165,373 tons, against 147,725 tons in 1906. The company's paving business shows a decline, caused largely by the increased output of manufactured products. During 1907 the company laid 2,402,041 sq.yd. of sheet asphalt for municipalities, against 3,298,104 sq.yd. in 1906; for private parties, 960,175 sq. yd. in 1907 against 1,060,428 sq.yd. in 1906; and at the close of 1907 had 535,160 sq.yd. under contract but uncompleted, against 921,554 sq.yd. at the end of 1906.

The General Asphalt Company owns deposits of asphalt in Trinidad, Venezuela, Indian Territory, Colorado and Utah, the original cost of which, including that of the Uintah Railway in Utah, was $6,342,311. The value of its real estate, based on an appraisal in 1903, at the time of the company's organization, is $959,830; and that of its plants, allowing for depreciation since 1903, is $3,204,303, a total property value of

$10,506,444, or over $80 per share on the outstanding preferred capital stock.

Asphalt Mining in the United States.

California.—The production of oil asphaltum during 1907 is estimated at 110,000 short tons, an increase of 55 per cent. over that of the preceding year. Aside from this marked increase, to which nearly all plants contributed, there were no noteworthy new developments. Prices, f. o. b. works, ranged during 1907 about as follows: "D" grade (for paving), $10@11.50 per short ton; "L" grade (liquid asphalt), $10.10@14 per short ton; "B" grade (with high melting point, for paints and roofing), $12.30@15 per short ton. Calforinia asphalts are at a great disadvantage in the eastern markets on account of the cost of transporting across the continent. Freight rates on asphalt from California to New York are as follows, per ton: All-rail, $12; rail and boat, via Galveston, $10; via Panama, $8; via Cape Horn, $3. Average prices of California asphalt in New York, during 1907, were: "D," $21@23.60; "L," $24@27; "B," $25@28 per short ton.

Indian Territory.—Several new companies were organized in 1907 to mine asphalt of various kinds in Indian Territory. Among others, were the Buckhorn Asphalt Company, at Sulphur; the Southern Asphalt Company, at Ardah; and the Grahamite Company, at Duncan.

Kansas.—The Standard Asphalt and Rubber Company, of Chicago, in 1907 built a plant at Independence for the manufacture of prepared asphalt products by a patented process. The crude materials employed are an asphaltic-base oil, from Oklahoma, and gilsonite from the company's mines in northeastern Utah. The products, going under the trademark "Sarco," include water-proofing paint, roofing and paving materials, etc.; the roofing asphalt is particularly noteworthy for the wide range of temperature between its brittle and its melting point.

Oklahoma.—According to a treaty between the Choctaw and Chickasaw Indians and the United States, all the known valuable coal and asphalt lands in both nations were segregated or set apart from allotment. The work was done by Joseph A. Taff in 1903 and 1904. Mr. Taff included in his segregation all the land containing asphalt that had been discovered up to that time, 7240 acres in all, of which 6880 acres had been leased by various companies and 360 acres were not leased. This land is included in 13 separate tracts, 12 of which are located in the Chickasaw Nation and one in the Choctaw Nation. The size of the tracts varies from 40 to 960 acres each. According to the terms of the treaty the leased land, 6880 acres, could not be sold without special act of Congress, so that only

360 acres of unleased asphalt land were on the market. Bids for this land were opened August 7, 1906, but all were rejected by the Secretary of the Interior. Since the time the land was segregated a number of additional deposits of asphalt have been discovered. The greater part of these are on alloted land, some of which can now be sold, and some of which will be on the market as soon as restrictions on the sale of the land have been removed. Some of this land lies near railroads and some of it is 20 miles away from transportation.

(By Charles N. Gould.) The asphalt deposits of the State of Oklahoma are among the most extensive in the United States. Exposures are found in the southern part of the State all the way from the Arkansas line to the Wichita mountains, but the greater part of the deposits occur in the region south of the Arbuckle mountains. Practically all the asphalt in Oklahoma is rock asphalt, and it occurs along fault lines, the rock on either side of the fault being impregnated with semi-liquid asphalt. The thickness of the so-called veins on either side of the fault varies from 2 or 3 ft. to more than 50 ft. Usually the sandstones are filled to a greater distance from the fissure than are the dense shales. Limestone is often impregnated with the material for a distance of 25 to 50 ft. The faults often outcrop on the surface for a mile or more, and the depth to which the fissures extend is unknown, but in some cases it is at least as much as 1000 ft. It is not known how extensive the Oklahoma deposits are but there need be no surprise if new beds continue to be discovered for the next hundred years.

Analyses of the asphalt made by Professor DeBarr, of the State University of Oklahoma, show that the composition of the material varies considerably. The so-called lime asphalt contains all the way from 2 to 10 per cent. of hydrocarbon, the remainder being calcium carbonate. The sand asphalt runs usually from 10 to 25 per cent. of hydrocarbon. The shale asphalt usually contains less than 5 per cent. of hydrocarbon and is not utilized. It is usually considered that a mixture of the lime and sand asphalt makes the best paving material.

At various times several of the different deposits have been worked. Mills have been erected at Gilsonite, three miles south of Sulphur, at Brunswick near Dougherty, and at Ardmore. In some cases the rock was simply crushed fine; in others an attempt was made to distil out the asphalt from the rock matrix. The rock has been used for paving in a number of cities, including Kansas City, Fort Worth and Ardmore.

The chief difficulties in the way of the profitable utilization of the asphalt of Oklahoma have been as follows: (1) The unfavorable attitude of the asphalt trust. Following well known methods, the trust has underbid independent operators and used either asphalt imported from Trinidad

or a manufactured product derived from oil refineries. It is even claimed that practically all the leases in the Chickasaw Nation are now held by the asphalt trust. (2) The lack of economical methods of utilizing the rock asphalt. The crushed rock is bulky, transportation charges often prohibitive, and the methods of refining the product on the ground have not always been satisfactory. (3) The high cost of transporting the raw product from the mines to the railroad. In spite of these objections, however, the fact remains that there are in Oklahoma vast deposits of asphalt, enough to last for untold generations, which are practically undeveloped.

Texas.—There are now three large concerns engaged in the manufacture (among other things) of asphalt from Texas and Louisiana petroleums. These are the Gulf Refining Company of Pittsburg, the Sun Company of Philadelphia, and The Texas Company. The last mentioned installed its operations in 1907. The Texas Company has works at Port Arthur, Port Neches and Dallas, Tex., and a refining plant at Marcus Hook, Penn. Its crude oil comes from the Beaumont, Sour Lake, Saratoga, Humble, and other Texas fields, from Lake Charles, Caddo, Amesville and other Louisiana points, and from Tulsa, Okla. Its products include various grades of illuminating, fuel and lubricating oils, and several varieties of asphaltic materials, such as paving, roofing, waterproofing and insulating cements, pipe dip, asphalt paint and felt roofing, all going under the name "Texaco."

The total arrivals of Texas asphalt at the ports of New York and Philadelphia during 1907 were 27,300 tons.

Utah.—Uinta county is the principal source of gilsonite in the United States. Gilsonite exists in vertical fissures in the sandstones and shales of the Eocene-Tertiary period. These fissures vary in width from 2 to 17 ft. from wall to wall. Until about three years ago, this product was mined in the vicinity of Fort Duchesne, Utah, and conveyed by Indian and Mormon wagon-freighters to Price, on the Rio Grande Western, at a cost of $11 per ton; that mined at Dragon, to the southeast of Fort Duchesne, was teamed to Rifle, on the same railway. The price for gilsonite in St. Louis was then about $50 per ton, and the cost, delivered there, was about $20 per ton. Several companies were doing a profitable business. Now, however, according to recent advices, all this is changed. The General Asphalt Company, which has obtained possession of most of the best veins, built a branch railroad, called the Uintah Railway, from Mack, a station on the Rio Grande Western, to Dragon, where is one of the chief producing veins. High rates on this, the sole outlet for the products, and a reduction of the price of gilsonite to $30 per ton have practically destroyed competition.

The old wagon road to Price, the former outlet, was supposed to be

kept in order by the county, but if this was not done, the United States Government, whose supplies for the troops at Fort Duchesne were hauled over the road, stepped in and made the repairs. Now, however, the Government contracts for these supplies over the Uintah Railway, and in consequence the Price road has fallen into such a condition as to be almost impassable for freighters. This has made the trust railroad practically the only means of transportation.

Relief, however, is in sight. Utah financiers have decided to build a railway from Provo over the Wahsatch range, and thence via the Strawberry and Duchesne river valleys to Jensen, a point on the surveyed line of the "Moffat road" on Green river, at the State line between Utah and Colorado. Engineers are now making the preliminary survey, and this line, which will pass through the center of the hydrocarbon fields, is practically assured.

The freight charge from Dragon to New York is, at present, $23.35 per ton, which includes the $10 haul on the Uintah Railway from Dragon to Mack. Independent shippers pay a wagon haul to Dragon in case they ship that way.

Although the General Asphalt Company at present has the monopoly of the market, it is likely that if the Moffat road, and the road projected by the Utah capitalists, ever pass through the Strawberry valley, and the owners and miners of gilsonite veins are placed in a position to save the present $10 haul, they will then be able to secure a fair amount of business in competition with the General company. The rate made by the Moffat road in such a case will be probably the same as that from Mack on the Rio Grande Western, and will thus eliminate the $10 charge.

The available tonnage, in Uinta county, of the pure form of crude bitumen known as gilsonite, was estimated, by the late Geo. H. Eldridge, of the United States Geological Survey, at 30,000,000 tons. The approximate annual consumption in the United States is between 8000 and 10,000 tons. This consumption seems to have increased materially during the last year or two, owing to the reduced price at which the product is now being sold.

The "Strip," as it is called locally, which is a piece chopped out of the Indian reservation many years ago, for the benefit of Utah men and others, for the reason that it contained a 4-ft. vein of exceptionally high-grade gilsonite, is now owned by the General Asphalt Company. The workings are about one-half mile long, and 90 ft. deep. Working underground with open lights was tried at first, but one day, at a point near the center of the length of the vein, the powdered gilsonite in suspension blew a hole in the earth, killing two men. After that, the deposit was worked by an open-cut or trench, so as to have daylight. This vein is not now being worked, for the reason that at the lower terminus of the Uintah

Railway the Dragon vein, 8 ft. wide, is supplying all present requirements.

Asphalt Mining in Foreign Countries.

Barbados.—The manjak deposits of Barbados occur in fissure veins which were formed during a period of upheaval which affected the Tertiary oil-bearing sands. The origin of the mineral is clearly seen in one of the veins in which, at a depth of 150 ft., the manjak passed into a thick asphaltic oil which could be bailed. The inference is that the fissure was filled with petroleum, issuing from the oil-bearing strata, and that the volatile parts were removed by oxidation. The Barbados manjak is purer and softer than that mined in Trinidad, and brings a higher price, or as much as $75 or $90 per ton in the English market. During 1907, New York imported 825 short tons of manjak from Barbados. It is used as the basis for black varnish and japan, being superior to domestic gilsonite in that it does not require the admixture of a black coloring agent.

Greece.—The first discoveries of asphalt deposits at Marathonopolis, on the western side of Peloponnesus, were made about five years ago. The material is a pure lime rock richly impregnated with bitumen. On the surface of the deposit the sun has bleached the rock to the color of ordinary lime, and this action has been attended by an enrichment of the lower part of the deposit. The material contains from 15 to 25 per cent. bitumen, which is about one-quarter fixed asphalt and three-quarters liquid petroleum. Dry distillation of the asphalt yields an oil which can be used in the manufacture of oil-gas, but which is more advantageously applied to the extraction of the bitumen. This last process is carried out either by the use of the oil, by heating the rock in a current of gas or acetylene to melt out the bitumen, or by extracting with benzol. The chief use of the bitumen is in the manufacture of an asphalt paving mastic which contains approximately 15 to 17 per cent. bitumen.[1]

Mexico.—The production of asphalt in Mexico is chiefly confined to the Tuxpam and Tampico districts on the Gulf of Mexico. A considerable amount of development work is now being carried on, especially in the Tuxpam district, as the Tuxpam asphalt is said to be the best in the world. The demand for Mexican asphalt seems to be in excess of the supply at the present time. Small shipments are made from time to time to Europe, and especially to Germany. The average price of Mexican asphalt is about 55 pesos per ton. The asphalt exported from Tampico is chiefly the residue from the oil produced by the Mexican Petroleum Company at Ebano, 40 miles west of Tampico. The distillate of this crude liquid asphalt is used for fuel oil. The distilling plant at Ebano could produce

[1] *Zeit. f. angew. Chem.*, July 5, 1907.

about 50 tons of asphalt per day, although this quantity has not been reached up to the present owing to the limited supply of barrels procurable for shipping purposes. Within the next few years it is expected that the production of asphalt will become one of the important industries of the Republic. During 1907 New York received 2230 tons of Mexican asphalt.

Russia.—The owner of the ozokerite works in the Island of Tcheleken, in the Caspian Sea, has applied to the Ministry of Trade and Industry for an additional allotment of land in various parts of the island. Investigations in Tcheleken have shown that there are on the island large deposits of ozokerite in no respect inferior to the best kinds found in Galicia, and further that all the allotments in the locality of Miyut, where the ozokerite industry has developed, are at the present time near the point of exhaustion. Russians and foreigners, as consumers, have applied to the works for ozokerite in quantities many times surpassing the output. For the last five years the output has averaged about 161 tons annually.

Switzerland.—Asphalt occurs prominently in the Val de Travers, in the Kanton of Neuenburg. This is in a typical Jurassic formation, certain calcareous strata of which, outcropping most extensively at Bois-de-Croid and at Presta, contain asphaltic matter. The thickness of the impregnated horizon ranges between two and eight meters; the workable part of the stratum contains 9 to 12 per cent. bitumen, and has the following average composition: Water, 0.90; bitumen, 10.34; calcium carbonate, 88.20; silica, 0.30 per cent.

Mining is conducted through underground workings. The asphalt is used for paving and for the production of illuminating gas. The asphalt mine at Travers is today the most profitable mining operation in Switzerland and also the only one in that country whose output is of any consequence in the European market. An English concern, the Neuchatel Asphalt Company, operates the mines, its annual output having a value of about 150,000 francs. The mine employs about 100 men, and the output during the last 20 years has exceeded 500,000 tons.[1]

Syria.—The asphalt mines at Hasbeya, near the headwaters of the Jordan river, have not been worked for several years, but old stocks accumulated during previous operations are still sold in Europe and the United States for making black varnish. The deposits of asphalt around the Dead Sea are said to be worthy of exploitation. Work has recently been begun, under an Anglo-Egyptian company, on bedded asphalt deposits in the Latakia mountains, a concession for which has been secured from the Turkish government. The deposits lie near Kferie, 30 miles northeast of the port of Latakia, on the road to Aleppo, and their great extent has been attested by competent surveyors. The surface

[1] Dr. C. Schmidt, in the "Handwörterbuch der Schweizerischen Volkswirtschaft, Sozialpolitik und Verwaltung."

material seems to be of fair quality, which will doubtless improve with depth. Permission to build a light railway from the mines to Latakia is being negotiated. An American is also buying asphalt lands in southern Palestine, around Beer-Sheba.

Trinidad.—The geology of Trinidad and the occurrence of its asphalt deposits were recently described by R. W. Ells.[1] The Tertiary beds of oil-bearing sandstones and shales are folded into four prominent anticlines, extending east and west across the island. Oil can be found by sinking a well almost anywhere on the island, and a number of deep borings, 1000 ft. or more, are now yielding profitable returns. Pitch Lake appears to be a basin into which petroleum collects, and by evaporation passes into asphalt.

The lake is a vast body of asphalt, brownish black in color, with an area of nearly 140 acres. It is situated near the west coast, at Point la Brea, at an elevation of about 100 ft. above sea level, and nearly one mile from the shore. In outline it is nearly circular, is deepest near the center, where a boring of 175 ft. failed to reach bottom, and shoals gradually toward the shores.

It has been ascertained that in the 14 years during which mining has been vigorously carried on, the level of the surface has been lowered 7 ft., or at the rate of 6 in. per year. In this period it is estimated that about 1,500,000 tons of the asphalt has been extracted and shipped.

The surface is hard, and the asphalt is mined with an ordinary pickaxe, the mineral breaking out readily with a sharp line of fracture. It is loaded into tram cars and either sent to the shipping point by a line of cable tram to the pier, or hauled along a second tram line by mules to the shipping point or to the boiling works, where it is purified by the removal of the contained water and of a certain amount of both organic and inorganic impurity. The digging is made to a depth of one to two feet, when the tram line is moved along the surface, but in a few weeks the depression thus made is filled and the surface is again level. There appears to be a certain slow movement going on which affects the greater part of the mass, and lines of flowage are seen in the apparently solid mineral as if the whole mass were in motion from the surface downward. This movement is apparently due to convection currents, which may be caused by the displacement of the whole mass through mining or possibly to the still further and continued inflow of semi-liquid pitch from the sides or bottom of the lake basin.

From the original lake basin immense quantities of the asphalt have been discharged seaward to the shore, where along the beach it now extends for more than a mile. This beach asphalt contains a somewhat larger percentage of impurity than that of the lake, since it has evidently picked

[1] Royal Soc. of Canada, May, 1907; summarized in the *Ottawa Naturalist*, Aug., 1907.

up certain inorganic, as well as organic, substances in its passage from the lake to the sea, the movement having apparently been made when the mineral was in a somewhat plastic condition. In composition the asphalt contains about 40 to 50 per cent. of bitumen, about 30 per cent. of water, the remainder consisting of the impurities mentioned.

The principal output from Trinidad is afforded by one of the General Asphalt Company's subsidiaries. During 1907 New York imported 35,400 long tons of asphalt from Trinidad. Exports from the island during the last eight years are stated in the accompanying table:

EXPORTS OF ASPHALT FROM TRINIDAD.
(In tons of 2240 lb.)

Year	Pitch Lake Asphalt.		Land Asphalt.		Total.	
	To United States.	Elsewhere.	To United States.	Elsewhere.	To United States.	Elsewhere.
1900 (a)	70,938	51,805	34,796	448	105,734	52,253
1901	80,449	55,605	31,767	3,150	112,216	58,755
1902	104,956	34,220	25,153	290	130,109	34,510
1903 (b)	123,582	41,950	18,478	4,886	142,060	46,836
1904 (c)	63,033	48,655	22,582	660	85,615	49,315
1905 (c)	47,947	54,054	12,126	770	60,073	54,824
1906 (c)	64,198	60,968	4,850	280	69,048	61,248
1907 (c)	86,824	53,560	4,145	200	90,969	53,760

Note.—A small proportion of the output undergoes a slight refining process before shipment. The above figures give the total exports, after recalculating the "épuré and dried" to their equivalent original amounts of crude material. Shipments elsewhere than to the United States are mainly to Europe. (*a*) Epuré and dried are not reduced to original crude in 1900. (*b*) For thirteen months ending January 31, 1904. (*c*) Years ending January 31, 1905, 1906, 1907 and 1908.

Turkey.—The asphalt mines of Selenitza were in regular operation during the whole of 1906. The production for the year was 6000 metric tons, of which 4800 tons were exported. The capacity of the mine will be much increased by the extension of the tramways and the erection of new refining furnaces. About 320 men are employed. Wages are low, but the cost of production is nevertheless high, on account of the lack of easy means of transportation. In order to transport material to the harbor, the company is obliged to employ numerous pack-trains, which are extremely expensive on account of the high price of fodder and the short life of the animals.[1]

Venezuela.—The bitumen deposits of Venezuela are extensive, running all along the coast from the Gulf of Paria to Colombia. They are particularly abundant at both ends of this line, viz: in the province of Bermudez and the basin of Lake Maracaibo. They are in form of superficial lakes similar in extent to that on the island of Trinidad, but much less thick. In the vicinity there are inflammable gases, petroleum and bituminous schist. Thus petroleum is found in the province of Bermudez at Punta de Aroya (Gulf of Cariaco), in Lake Maracaibo, at the confluence of the river Tara and Catalumbo, and between Escuque and Betijoque.

[1] *Montan-Zeit.*, Aug. 1, 1907.

A surface deposit of bitumen can be seen in Lake Maracaibo, 20 hectares in area, the formation of which is yet in progress, being fed by a thermal spring.

The principal operations are at Bermudez, where mining is conducted under the management of a receiver appointed by the Venezuelan court. The General Asphalt Company, one of whose subsidiaries used to own the deposit, has protested against the final decision of the Venezuelan government, by which their title was annulled. Shipments of Bermudez asphalt to New York during 1907 were about 32,000 long tons.

WORLD'S PRODUCTION OF ASPHALT AND BITUMINOUS ROCK.
(In metric tons.)

Asphalt.

Year.	Cuba.	Germany.	Hungary	Italy.(*c*)	Russia.	Spain.	Trinidad. (*b*)	United States.	Venezuela.	Total.
1900	....	89,685	2,900	33,127	25,090	2,331	161,299	8,326	17,981	340,739
1901	4,554	90,193	2,878	31,814	26,622	4,182	173,707	19,882	22,115	375,947
1902	4,966	88,374	2,773	33,684	12,360	6,064	167,253	36,923	10,770	363,167
1903	7,368	87,454	2,422	35,757	25,577	4,372	196,883	54,521	14,567	399,587
1904	8,926	91,736	2,221	34,227	(*d*)	3,761	137,089	77,250	23,535	375,845
1905	10,142	115,267	247	26,838	(*d*)	5,752	116,735	68,935	(*e*)25,000	368,916
1906	5,186	138,059	4,111	34,386	(*d*)	6,229	132,381	94,316	20,080	434,748

Bituminous Rock.

Year.	Austria.	France. (*a*)	Italy.	Spain.	United States.	Total.
1900	887	34,093	101,738	4,193	41,029	181,940
1901	541	29,815	104,111	3,956	37,393	175,816
1902	901	34,000	64,245	6,301	35,072	140,519
1903	1,273	....	89,690	6,277	37,334	(*f*)134,574
1904	1,435	....	111,390	100	19,454	(*f*)132,379
1905	4,363	191,509	106,586	750	32,337	335,545
1906	2,840	196,375	130,825	7,794	21,848	359,662

(*a*) France produces a large amount of bituminous shales, used for distilling oil, which is not included in these statistics. (*b*) Exports (crude equivalent) reported by the New Trinidad Lake Asphalt Co. (*c*) Including mastic and bitumen. (*d*) Not yet reported. (*e*) Does not include the production of Cuba. (*f*) Does not include the production of France.

Note.—There is a considerable production of asphalt stone in Switzerland of which no account is taken in the above table, the Swiss Government not publishing any mineral statistics. The production of manjak in Barbados is not included in the statistics given.

BARYTES.

By Edward K. Judd.

The prevailing high prices for finished barytes throughout 1907 stimulated the output from all the established districts, led to the development of some not hitherto productive deposits, and caused plans to be drawn up for several new grinding plants. As in previous years Missouri produced more than any other State, Tennessee, Virginia and North Carolina following in the order named. Kentucky for the first time reported a substantial output. The principal new developments occurred in Missouri, Kentucky and Nevada.

STATISTICS OF BARYTES IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.			Imports.				Consumption.	
	Short Tons	Value.		Crude.		Manufactured.			
		Per Ton.	Total.	Sh. Tons.	Value.	Sh. Tons.	Value.	Sh. Tons.	Value.
1897........	26,430	$4.00	$105,720	502	$ 579	1,300	$13,822	28,232	$120,121
1898........	28,247	4.00	112,988	1,022	2,678	687	8,678	29,956	124,344
1899........	32,636	4.20	137,071	1,739	5,488	2,111	22,919	36,486	165,478
1900........	41,466	3.90	161,717	2,568	8,301	2,454	24,160	46,488	194,178
1901........	49,070	3.22	157,844	3,150	12,380	2,454	27,062	54,674	197,286
1902........	58,149	3.21	186,713	3,929	14,322	3,908	37,389	65,986	238,424
1903........	(a)50,397	3.02	152,150	6,344	22,777	5,716	48,726	62,457	223,653
1904........	(a)65,727	2.66	174,958	6,689	27,463	5,920	48,658	78,336	251,070
1905........	53,252	3.68	196,041	7,879	36,796	4,827	39,803	65,418	272,649
1906........	63,486	3.98	252,719	9,189	27,584	4,808	37,296	77,483	317,599
1907........	65,579	3.83	251,308	18,344	77,683	10,006	96,542	93,929	425,533

(a) Statistics of the U. S. Geological Survey.

Market and Prices.—The year 1907 was an exceedingly active one in the paint trade, and this, coupled with the shortage of crude material during the previous year, caused prices on finished products to advance by at least 10 per cent. early in 1907. Prices ranged high throughout nearly the entire year, until a slight recession occurred in December. Quotations at the end of the year were about as follows: Foreign, first-class, water-floated, $21.50; Domestic, first-class, water-floated, $20.50; Domestic, first-class, dry-ground, $18@20; Domestic, off-color, $13.50@17.50 per short ton. Imports of finished barytes were greatly stimulated by the advancing prices. The crude mineral varied greatly in price, sales being recorded as low as $2 and as high as $5 per long ton, depending on the purity of the product.

Barytes Mining in the United States.

Kentucky.—Considerable attention was devoted in 1907 to the well-known but previously unexploited barytes deposits of central Kentucky. The Dix River Barytes Company began mining at a number of points around Danville, shipping its crude product to New York for grinding. The deposits in this locality are veins with well defined rock walls and in some cases yield appreciable amounts of strontium. At Nicholasville, south of Danville, the Jessamine Barytes Company opened deposits and erected a grinding mill. At several other places in this vicinity certain individuals began operations in a small way. A grinding mill is contemplated for Danville. The Marion Zinc Company, of Marion, reported a small production of barytes during 1907.

Missouri.—A new grinding plant was planned by local capitalists at De Soto, in the center of the producing district, but its erection was not begun at once. The same interests now have in full operation at East Alton, Ill., a plant for the manufacture of barium salts and compounds, the crude material for which comes from the Missouri district. The De Soto company has acquired mining rights over a large territory and has arranged to consume the output of other smaller producers. An explosion in the drying department of the plant of the Point Mining and Milling Company, at Mineral Point, caused a suspension of grinding operations at this plant for five months in 1907.

The total production of crude barytes in Missouri in 1907 was 40,442 short tons, valued at $162,365, as compared with 36,148 tons, valued at $155,975, in 1906.

(By E. R. Buckley.) There was very little change in the production or methods of mining barytes in Missouri during 1907. The output comes chiefly from Washington and Franklin counties, although there was an increased production from Cole and other counties of the central Ozark region. During the latter part of the year there was an almost complete suspension of mining for barytes owing to the financial depression. The land owned by the American Lead and Barytes Company was sold under foreclosure and it is expected that under the new management some improvements will be made in the methods of mining, which up to the present time have been the crudest possible. The Point Mining and Milling Company has nearly a year's supply of barytes in stock. Its mill has been remodeled and the capacity increased.

Montana.—The discovery of workable barytes is reported from a point two miles southwest of Missoula. The vein averages 3 ft. thick and dips 70 deg.; the quality is said to be good. Lump barytes is also reported from Custer and Dawson counties.

North Carolina.—A new concern is the Georgia Barytes Company, of

Asheville, which will develop deposits and erect a small mill in the South. Development of the barytes mines on Crowders mountain, Gaston county, was resumed in 1907 by the Clinch Valley Baryta Company, of Virginia, and substantial shipments were made from the Lawton mine. The operation near Stackhouse, under the direction of Mr. W. H. Terry, was closed in October, after having produced a good quantity of barytes. The Rector property, in the same locality, was again put in operation.

Nevada.—A promising new deposit was developed in 1907 by New York capitalists near Blair. The barytes occurs in a vein of which galena is the only other constituent. The company intends to build a mill for separating the two minerals; the lead will then be sold to smelters and the barytes, already crushed and thoroughly cleaned by the jigging process, will be ground for market.

Virginia.—In addition to the output of the mines operating in the vicinity of Honaker, Gardner and Richlands, in Russell and Tazewell counties, which continue to be the principal producers in the State, and the Bennett mine near Toshes, in Pittsylvania county, small shipments of barytes were made from mines reopened in Smyth, Bedford and Louisa counties. Mining on a small scale was recently resumed to the west of Marion in Smyth county. A property in Bedford county, about three miles west of Thaxton station, was opened during 1907. In Louisa county mining was again begun about three miles east of Lindsay, a station on the Chesapeake & Ohio Railway.

Barytes Mining in Canada.

Nova Scotia.—The Barium Production Company, of New York, in 1907 secured control of the Lake Ainslie barytes deposits on Cape Breton Island, and will establish a grinding plant in the vicinity of New York, with the intention later of going into the manufacture of barium salts and compounds.

(By W. Spencer Hutchinson.) The Five Islands district is in Colchester county, Nova Scotia, on the shore of Minas bay, an arm of the Bay of Fundy. It is 16 miles by wagon road to Parrsboro, the nearest railway station which connects, by the Cumberland Railway, with the main line of the Intercolonial Railway at Spring Hill Junction. The name of the place comes from a chain of islands which lie off shore, remnants of a point of land eroded by the sea. The geological structure is still revealed on the mainland in the range of hills extending along the shore. These are chiefly of soft, reddish volcanic ash loosely cemented, but with a great dike of black diabase forming a central rib, and with the ash partly capped by a flow sheet of diabase.

The erosion of the soft ash and the breaking of the diabase dike have formed the islands. The first island is not wholly robbed of the ash flanking the dike, but on the second, Diamond Isle, nothing remains but the precipitous cliffs of diabase. Parallel to the range of volcanic rocks and about four miles north of it, is another range composed of carboniferous slates and limestones, in which the barytes deposits have been found. The flat between is occupied by the marshes and flats at the mouth of the East river and by the gravel plain on the banks of Bass river. The principal barytes deposits have been found in the sedimentary rocks, and the most extensive workings are on Bass river about three miles from the Government wharf at Five Islands.

The mines were first opened for barytes in 1866 and from that date until 1871 the production is said to have been at the rate of 2000 or 3000 tons annually. Disagreement between owners of undivided interests in the property resulted in the closing down of the mines and the titles eventually came into very bad shape. Although the mines were idle for 35 years, their valuable character was known and attempts to clear the titles were made by various parties, without success, until taken up by A. R. Bayne, of Boston, who is now reopening the mines and will shortly begin shipment. Much of the old work was by lessees and was done as cheaply as possible without proper development and the mineral was gouged out wherever found.

The Bass river barytes is crystalline and only slightly yellow on the outer surface and cleavage faces, while many specimens are pure white. The barytes occurs in veins in crushed and metamorphosed slates. The position of the old workings indicates that the principal bodies of barytes were found in a zone, perhaps 70 ft. in width, extending in a general northwest course from the river. The country rock in this zone is very much crushed, being recemented by carbonate of lime, which also occasionally lines cavities in the vein and forms beautiful crystals of dog-tooth spar. The veins of barytes have well defined walls, but hold no regular course or position. They show widths from a few inches to 4 ft., with pocket-like masses sometimes 10 or 15 ft. in width. At one point an open cut exposes a large but very irregular mass in black slate. This seems to fill the voids in a faulted and broken zone in the slate, with large and small irregular fragments of slate inclosed in the mineral.

The analysis of a general sample taken from the bin at the mine gave the following results: Loss on ignition, 0.18 per cent.; SiO_2, 0.95; FeS_2, 0.07; Al_2O_3, 0.02; CaO, 0.02; MgO, 0.22; $BaSO_4$, 98.54; total, 100. The loss on ignition probably represents, besides moisture and combined water, organic matter and sulphur. The total amount of metallic iron present is 0.032 per cent. The magnesia probably occurs in the mineral in the form of talc, a hydrated silicate of magnesia. The analysis shows the

remarkable purity of the barytes and specially selected specimens are almost pure barium sulphate.

There is another deposit of barytes on East river about five miles northeast of the village. This vein is in chlorite rock, and an open cut and tunnel exposed it for a length of 50 ft. on the bluff above the river. One wall is regular and well-defined, but the other wall is generally broken and shows fragments of country rock inclosed in the mineral. This barytes is grayish white and wholly free from yellow stains. Both mineral and country rock break freely on cleavage faces and joints which will greatly aid in the profitable mining of the barytes.

Notes on the Milling of Barytes and the Manufacture of Barium Products.

By Edwin Higgins.

The following remarks are especially applicable to plants in which the barytes is crushed, bleached and afterward milled to an impalpable powder.

Preliminary Crushing.—Belts for operating crushers, rolls and other machinery should under no circumstances be allowed to come in contact with the moisture or steam of the tank room. Belting deteriorates rapidly under the action of the small amount of sulphuric acid carried off in the steam from the tanks.

In most cases wet crushing is more economical than dry crushing, especially in countries where freezing weather is encountered. Crude mineral, as usually delivered to the crusher, carries with it sand or clay which is often wet. Much time is lost in dry crushing such material in cold weather, the crusher jaws becoming coated with partly frozen clay and refusing to take hold of the rock properly. The system of wet crushing also aids greatly in removing the clay and sand from the mineral before it is conveyed to the bleaching tanks. When it is necessary to resort to preliminary washing to remove clay and other light impurities, this operation should be performed after the material leaves the crusher, especially when the mineral is friable, for if washing is carried on after the material leaves the rolls a much greater percentage is lost in fines. The mineral should not be crushed to a smaller mesh than is necessary to expose the impurities to the action of the acid. Jigging may be successfully resorted to for the removal of chert, limestone and other impurities of light weight.

Bleaching.—A circular tank, 8 ft. in diameter and 4½ ft. high, holding from 13 to 14 tons of crushed mineral, is convenient for bleaching, and may be satisfactorily steamed from one inlet. The connections should be made so that either steam or water may be supplied to the tank through

the perforations in the lead pipe. The perforations are best located at an angle of 45 deg. off the vertical diameter of the pipe, rather than directly on top. In supplying a fresh bath of acid to a tank of crushed mineral a convenient method is to turn on the steam and at the same time supply water to the tank from a hose. As soon as the water rises to the desired hight the acid may be added. It is best not to use too much solution at the start, for condensing steam soon begins to dilute the bath. It is good practice to use barely enough solution at the start to cover the charge of crushed material. The proper strength of the bleaching acid is from 20 to 30 deg. B. The bed of crushed mineral in the tank should be from 3 to 4 ft. deep, depending on the size to which the material has been reduced. Finely ground material in any undue quantity should not be introduced into the tanks, as it impairs the free circulation of the acid and serves to clog the perforations in the lead pipes.

A great item of expense in the bleaching of barytes is the steam required for heating and agitating the bath. This cost may be greatly reduced by the use of an ordinary injector, a mixture of air and steam being used instead of steam alone. The injector may be attached just below the valve admitting the steam to the lead tank pipes. The amount of air admitted may be easily controlled. By this simple arrangement the steam consumption may be greatly lowered, and the temperature of the baths kept at 200 deg. F.; better agitation will result and the acid will suffer less dilution from condensing steam.

A siphon made from an ordinary piece of lead pipe is a convenient device to have around the bleaching room. By its use any over supply of acid may be removed from a tank. If after bleaching the ore is washed in the tank, the siphon may be used to advantage in removing the wash water.

Handling the Acid.—Sulphuric acid is most economically and safely handled with the aid of compressed air. By keeping the acid in a large storage tank and forcing it in the desired quantity to a graduated tank situated above the bleaching tanks, much time is saved and danger avoided. From this graduated tank a given quantity of acid may be delivered to any of the bleaching tanks. On withdrawing the bath of acid from a tank of bleached mineral it often happens that there is sufficient strength left in the solution to warrant its further use. This acid, variously referred to as sludge, or pickle, may be run into a separate storage tank, to be later forced to the desired bleaching tank with the aid of compressed air.

Drying.—The bleached material should be dried at an even heat, and never should be sent to the mill only partially dried. Imperfectly dried material is liable to "sweat" before reaching the buhrs, in which case the finished product will be off color. If the bleached material is delivered to the dryer through mechanical conveyers care should be taken in order

that no oil may come in contact with it before the drying process is begun. A few drops of oil from a dripping journal box, or elsewhere, will greatly discolor a large quantity of the bleached material.

By-Products.

Venetian Red.—A fair quality of pigment, suitable for the manufacture of cheap red paint, may be obtained from the bleaching solution after its strength has been entirely dissipated in the bleaching tanks. If there is still some strength left in the solution, but not enough to warrant its further use in the tanks, it may be run into a separate tank containing scrap iron where, in a few hours, the remaining free acid will be consumed. By adding milk of lime to this solution, a precipitate consisting of the hydrous oxide of iron and calcium sulphate will be formed, which on being roasted, produces the red pigment.

Copperas.—It is also possible to make copperas from the pickle, or sludge: by evaporation or by concentrating the solution in lead lined tubs or tanks in the presence of scrap iron, heat being supplied by means of a coil of lead pipe through which steam is passed. When the solution has been brought to about 40 deg. B. it is transferred to wooden tanks for cooling. Here the crystals of copperas are formed. Additional surface for supporting the crystals may be provided by suspending strips of wood in the solution.

Fines.—If the crude barytes is of a friable nature and the amount handled is sufficiently large, it is a good plan to allow the wash water, used before and after bleaching, to pass through settling boxes. Here the fine barytes will be saved. The fines from the bleached material may be sent to the mill to be ground; that from the preliminary washing, if in great enough quantity, may be economically treated on almost any style of concentrating table.

Lithophone and Blanc Fixe.—There are about nine concerns that manufacture lithophone in the United States. Little information is obtainable on the subject. Lithophone is a precipitate resulting from mixing solutions of barium sulphide and zinc sulphate. The barium sulphide is formed by heating a mixture of fine coal and barium sulphate to bright red in a reducing atmosphere. The usual procedure is to mix the crude barytes with coal containing little ash, in the proportion of four parts (by weight) of barytes to one of coal. This mixture is charged into a cylindrical revolving furnace and kept at a bright red heat for from 2½ to three hours. The furnace is so arranged as to permit the admission of only a small quantity of air. After this treatment the charge contains from 60 to 70 per cent. of soluble barium sulphide, the remainder being chiefly barium carbonate which may be dissolved in either nitric or hydrochloric acid and the corresponding salt recovered.

Barium sulphide and zinc sulphate are both soluble in water. In Germany lithophone is made largely from zinc sulphate obtained directly from ore at the works in the lower Harz, but in the United States the zinc sulphate is chiefly obtained by dissolving scrap zinc, or zinkiferous by-products, in sulphuric acid. At one works scrap brass is the source of the zinc employed. The zinc is burned off from the brass, leaving molten copper behind, while the zinc oxide is collected in settling chambers and subsequently dissolved in sulphuric acid, the solution being of course suitably purified.

Blanc fixe, or chemically pure barium sulphate, is precipitated from a solution of barium sulphide either by sulphuric acid or by salt cake. When salt cake is used to precipitate the barium sulphate, sodium sulphide is formed by the reaction. This is used to a small extent as a depilatory in the tanning industry, the chief consumers being in Massachusetts and the neighborhood of Philadelphia, Penn. Sodium sulphide is worth about $30 per ton, but of course the market is extremely small and it would be impossible to dispose of any large quantity of by-product in this way. However, one concern in Kentucky is manufacturing both blanc fixe and sodium sulphide. E. E. Dwight & Co., of Webb City, Mo., manufacture barium sulphate and zinc sulphide, using presumably acid mine water which occurs in that district.

Blanc fixe is absolutely white and free from grit and is used in the manufacture of the finer grades of white paint, oilcloth and paper. A large quantity of the blanc fixe used in this country comes from Germany.

Determination of Barium Sulphide.—According to L. Wessely (*Chem. Zeit.*, 1907, XXXI, 71-72), the sample is finely powdered, and 10 grams are very gradually introduced into 500-700 c.c. of boiling water, with continual shaking. Any lumps which form must be taken out and powdered again. When solution is complete, the whole is poured and rinsed into a liter flask, cooled, made up, and filtered. Of the filtrate, 25 c.c. are titrated with N/10 hydrochloric acid and methyl orange; then 300 c.c. of water, 2 c.c. of dilute hydrochloric acid, and a measured volume of N/10 iodine solution nearly equal to the volume of acid required in the first titration are placed in a roomy flask, and 25 c.c. of the barium sulphide solution allowed to flow in with shaking. The slight excess of barium sulphide is now titrated by iodine solution and starch. If more than a few tenths of a cubic centimeter are needed, the operation should be repeated, using a correspondingly greater amount of iodine solution in the first instance. One c.c. of N/10 iodine = 0.0084745 gram of barium sulphide.

Manufacturers of Barium Products.—In the following table are given the names and addresses of all the consumers of crude barytes in the United States known to us, together with a note of the operation conducted by each:

Name.	Address.	Operation.
Nulsen, Klein & Krausse Mfg. Co	St. Louis, Mo	Bleach, grind, float.
Finck Mining and Milling Co	St. Louis, Mo	Bleach, grind.
Point Mining and Milling Co	Mineral Point, Mo	Bleach, grind, float.
Nulsen, Klein & Krausse Mfg. Co	Lynchburg, Va	Bleach, grind.
Pittsburg Baryta and Milling Corp	Richlands, Va	Bleach, grind.
Clinch Valley Baryta Co	Honaker, Va	Beach, grind.
Commercial Mining and Milling Co	Knoxville, Tenn	Bleach, grind.
William D. Gilman Co	Sweetwater, Tenn	Crush, jig, roast, salts.
John T. Williams & Son	Bristol, Tenn	Bleach, grind, roast, salts.
Carolina Barytes Co	Stackhouse, N. C	Bleach, grind.
Hot Springs Mfg. Co	Hot Springs, N. C	Bleach, grind.
Delaware Barytes and Chemical Co	Dover, Del	Bleach, grind, salts.
Barium Production Co	New York, N. Y	Bleach, grind.
Dix River Barytes Co	Lancaster, Ky	Bleach, grind.
Hammill & Gillespie	Stamford, Conn	Bleach, grind.
Cawley, Clark & Co	Newark, N. J	Lithophone.
N. Z. Graves & Co	Philadelphia	Lithophone.
Harrison Bros. & Co	Philadelphia	Lithophone.
Excelsior Mfg. Co	Newport, Del	Lithophone.
Grasselli Chemical Co	Grasselli, N. J	Lithophone.
New Jersey Zinc Co	Palmerton, Pa	Lithophone.
Cheeseman Chemical Co	Scranton, Pa	Lithophone.
Becton Chemical Co	Becton, N. J	Lithophone.
American Paint and Pigment Co	East Alton, Ill	Roast, salts.
E. E. Dwight & Co	Webb City, Mo	Roast, salts.
Krebs Pigment & Chemical Co	Newport, Del	Lithophone.

BAUXITE.

By Edward K. Judd.

The only noteworthy events in the bauxite industry of the United States in 1907 were the beginning of active mining near Chattanooga, Tenn., by the National Bauxite Company, and the transfer of the mines in Arkansas, formerly owned by the American Bauxite Company, to the Republic Mining and Manufacturing Company. Elsewhere in Arkansas, and in the Georgia-Alabama field, the industry remained much the same as during 1906.

PRODUCTION OF BAUXITE IN THE UNITED STATES.
(In tons of 2240 lb.)

State	1897	1898	1899	1900	1901 (a)	1902	1903 (a)	1904	1905	1906	1907 (a)
Alabama	13,083	13,848	14,144	650	18,038	5,577	22,374	7,087	17,094	27,131	97,776 (b)
Georgia	7,507	12,943	19,619	20,715		19,000		16,909			
Arkansas	....	....	3,050	2,080	867	4,645	25,713	24,016	30,897	51,200	
Total	20,590	26,791	36,813	23,445	18,905	29,222	48,087	48,012	47,991	78,331	97,776

(a) Statistics of the United States Geological Survey. (b) Production of Tennessee included.

The following companies were in active operation during 1907: Aluminum Company of America at its numerous mines in Saline county, Ark., at the South Watters mine near Hermitage, Ga., and at the Monahan mine at Rock Run, Ala.; Republic Mining and Manufacturing Company at the Julia and the Holland Spring mines near Hermitage, Ga., at the Dykes and the "No. 142" mines at Rock Run, Ala., and at the England mine, six miles from Little Rock, Ark.; the National Bauxite Company at Cave Springs, Ga., and on Mission Ridge, near Chattanooga, Tenn.; John H. Hawkins, at Kingston, Ga. Some New England parties also shipped an experimental lot of ore from Cave Springs, Ga.

Only a small amount of new equipment was installed during 1907. The Aluminum Company of America built a spur track from its works to the Iron Mountain road. The National Bauxite Company erected a new hoisting plant at its Mission Ridge mine, and will probably soon build a drying plant there also. No new equipment was installed in the Georgia-Alabama district, so that the catalogue of the mining and drying plants of the Southern field, published in Vol. XV of The Mineral Industry, remains complete.

Winthrop C. Neilson[1], president of the Republic Mining and Manufacturing Company, said, regarding the future of the Southern field: "It is impossible to form any correct estimate as to the amount of bauxite still in the Georgia-Alabama section, as the ore occurs in round deposits of considerable depth, some of which are very small and some of which are quite large. Unlike the Arkansas bauxite, which is of blanket formation, and can be estimated upon with some degree of accuracy, the Georgia-Alabama field is poorly adapted to give any data. However, since the bauxite industry started in the South in 1889, there have probably been shipped some 300,000 long tons, and I feel quite sure there is as much ore still in that section as has been taken out. The Georgia-Alabama bauxite belt runs practically from Chattanooga, Tenn., to Anniston, Ala., going directly through Rome, Ga. This is approximately 140 miles in length, but it is not wide—probably 10 miles, as an outside estimate, would cover its width. The ore occurs in certain sections, but with no regularity, and is very fickle."

Alabama (By Eugene A. Smith).—At this time bauxite is known to occur in this State only in Cherokee, Calhoun, Talladega and De Kalb counties. Near Rock Run in Cherokee county are situated some of the largest mines of this mineral in the South; they lie mostly in Alabama, though partly in Georgia. This locality furnishes practically all the bauxite now mined in this section. It is in very close association with a bank of brown iron ore and a deposit of kaolin, or white clay. In Calhoun county several occurrences of bauxite have been discovered, but they have not been opened up and the quality of the bauxite is therefore unknown; similarly near Fort Payne in De Kalb county an occurrence of bauxite has been reported.

Most of the bauxite sent from this State is used in the manufacture of alum, and not for the manufacture of aluminum metal. During 1907 shipments of bauxite were made by two companies, the Republic Mining and Manufacturing Company, and the Aluminum Company of America, the shipping point being Rock Run, and the amount being about 11,000 long tons. A couple of carloads, however, for experimental purposes were shipped from the Rock Run section by some explorers. The Alabama situation might be summed up at present as being practically the same as it has been for some years past.

Georgia (By Otto Veatch).—A discovery of bauxite was made in Central Georgia early in 1908. The occurrence has not yet been thoroughly prospected and it is too early to make any predictions as to its future commercial importance, although the outcrops of the mineral present a very favorable prospect.

The deposit is situated three miles east of McIntyre, Wilkinson county,

---

[1]*Manufacturers' Record*, June 13, 1907.

and about 1½ miles north of the Central of Georgia Railway. The deposit is in no way connected with those of northwest Georgia, either geographically or geologically, being located about 150 miles distant, and in the Coastal plain. It lies near the northern margin of the Coastal plain and occurs in strata of probably lower Cretaceous age, the Tuscaloosa formation. It is overlaid directly by red sands and impure clays of Eocene age. The strata in this vicinity are for the most part unconsolidated sands and clays, having no pronounced structural features; the beds lie almost horizontal, having only a slight dip southward.

The bauxite occurs in blanket form, its lateral extent greatly exceeding its thickness. The position of the bed can be traced by small fragments at the surface for a distance of about ½ mile, and it maintains a rather uniform level. It lies near the contact of the red argillaceous Eocene sands and a bed of white and stained massive clay. I offer the opinion, tentatively, that the bauxite is an alteration product of the bed of white clay, or kaolin, and has resulted from desilication of the kaolin.

The mineral is pisolitic and varies in color from bright red to yellow and cream color. The pisolites are generally quite small, the majority varying from 1/10 to ½ in. in diameter; they may either make up the entire mineral, or may be scattered in an amorphous matrix. The bauxite is hard at the surface, although it will doubtless be found much softer where not exposed to atmospheric action.

The writer collected three samples of the pisolitic mineral from the outcrops. These were analyzed by Dr. Edgar Everhart, chemist for the Geological Survey of Georgia, with the results shown in the accompanying table.

BAUXITE FROM CENTRAL GEORGIA.

	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	H_2O Comb. %	H_2O hygros. %	Total %
I	57.58	9.38	0.96	2.76	29.12	0.35	100.15
II	52.92	10.17	7.66	2.30	26.55	.35	99.95
III	55.21	12.40	.96	2.15	29.10	.33	100.15

The second analysis represents the most ferruginous of the samples, which was a bright red in color. The clay underlying the bauxite approaches a kaolin in composition, except that it is in places highly stained with iron oxide.

The lateral extent of the deposit has not yet been determined, and the total amount of the bauxite cannot yet be estimated. However, it is known to reach a thickness of 10 ft., and from the outcrops and a small amount of prospecting, 100,000 long tons are known to occur at the point of discovery; this amount will certainly be increased by further

prospecting. The overburden consists of unconsolidated sand and will vary from a few feet, with a gradual increase, to 40 ft., provided that the bed is extensive laterally. The mining of the ore would present no special difficulties, and water for power and other purposes is abundant.

CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.			Imports.		Exports.		Consumption.	
	Long Tons.	Value.	Per Ton.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1896	17,096	$42,740	$2.50	2,119	$10,477	....	....	19,215	$53,217
1897	20,590	51,475	2.50	2,645	10,515	2,537	$5,074	20,708	56,916
1898	26,791	66,978	2.50	1,201	4,238	1,000	2,000	26,992	69,216
1899	36,813	101,235	2.75	6,666	23,768	2,030	4,567	41,449	12,436
1900	23,445	85,922	3.66	8,656	32,968	1,000	3,000	31,101	115,889
1901	(a)18,905	97,914	4.23	18,313	66,107	1,000	3,000	36,218	144,021
1902	29,222	128,206	4.39	15.790	54,410	*Nil.*	....	43,112	175,875
1903	(a)48,087	171,306	3.56	14,889	49,684	*Nil.*	....	62,976	220,990
1904	48,012	166,121	3.46	15,475	49,577	*Nil.*	....	63,487	215,698
1905	47,991	203,960	4.25	11,726	46,517	*Nil.*	....	59,717	250,477
1906	78,331	352,490	(e)4.50	17,809	63,221	*Nil.*	....	96,140	415,711
1907	....	....	....	25,065	93,208	*Nil.*	....	....	....

(*a*) Statistics of the United States Geological Survey. (*e*) Estimated.

Bauxite Mining in Foreign Countries.

Austria.—The British Consul at Trieste reports that several bauxite deposits have been discovered in Lesina, which is one of the islands in the Adriatic, forming part of the province of Dalmatia.

India.—W. R. Dunstan, director of the Imperial Institute, reports that samples of bauxite from the Bailier tahsil, Balaghat district, range from 52 to nearly 59 per cent. alumina, with silica from 0.58 to 2.65 per cent. In this area high proportions of titanium oxide occur in the bauxite, as much in one case as 13.76 per cent. One of the samples contained only 2.70 per cent. of ferric oxide, and might be used for the preparation of aluminum salts as well as for the manufacture of pure alumina suitable for reduction to the metal.

A test made in 1906 on a large scale in one of the factories in Europe showed that the Indian bauxite is suitable for the manufacture of pure alumina by Bayer's process. Indian bauxite might be developed in three ways: Export of the raw or calcined material to Europe or America for use in the alumina factories; manufacture of pure alumina locally by extraction with alkali, and export of the pure oxide to European or American aluminum works; manufacture of the metal in India. The first proposal is considered to be impracticable on account of the low prices of crude bauxite at European ports, while the third proposal would involve a heavy capital outlay under untried conditions and an elaborate preliminary investigation before power works could be erected. The second proposal involves smaller risks, and appears to be the one most practicable as a beginning in India.

Ireland.—Some attention is being paid to the iron and bauxite deposits in the north of Ireland at present. A good deal of the bauxite used in the manufacture of aluminum comes from this district, and owing to the increased demand for aluminous ores, extensive prospecting operations have been conducted throughout the counties of Antrim and Derry. A company called the Derry and Antrim Ore Company has recently been formed by Scotch and north country people to develop several groups of deposits thus discovered. They are in the neighborhood of Portrush and Bushmills. Some of the ore runs to 56 per cent. of alumina. The leading companies already in operation in this vicinity are: The Antrim Iron Ore Company, at Cushendall; the Crosumellin Mining Company, at Cargan; and The Bauxite Company, at Ballymore.

Progress in the Technology of Bauxite.

Magnetic Phenomena.—A chance observation led Köbrich[1] to establish the fact that the bauxites of the Vogelsberg are possessed of magnetic properties, and he regarded the discovery as of practical importance, inasmuch as it may be found practicable to separate magnetically the bauxites rich in iron from those that are comparatively free from it. The bauxites that possess a deep red color are characterized by magnetic properties. Köbrich found that the seat of the magnetism lay in certain red grains, which were tiny crystals of olivine enveloped in a crust of iron oxide of varying thickness.

Extraction of Alumina from Bauxite.—Several European patents have been granted during the last two years for the refining of bauxite. By the process of A. Clemm,[2] bauxite, or other aluminous ore, is mixed with a large excess of an alkali sulphate and carbon, and the mixture is furnaced. The solution obtained on lixiviating the mass, containing alkali sulphide and aluminate, is treated with sulphurous acid to precipitate aluminum hydroxide, and to form a thiosulphate, and the filtrate therefrom is concentrated to crystallize out the thiosulphate. The residue left on lixiviating the roasted mass is exposed to the air with addition of milk of lime, and the solution obtained by further washing is concentrated to recover the alkali, etc., as a thiosulphate. Hydrogen sulphide may be used as the precipitant, in place of sulphurous acid, in which case the filtered solution contains sodium sulphide and hydrosulphide.

Instead of using alkali sulphate and coal, H. Arsandaux[3] mixes bauxite with sodium carbonate and roasts. The product, containing silicates, is treated with sodium carbonate solution and then with an equivalent quantity of milk of lime or with finely divided or recently precipitated

[1] *Zeit. f. prakt. Geol.*, 1905, XIII., 23-36.
[2] French patent No. 363,893, Mar. 6, 1906.
[3] French patent No. 361,766, Sept. 13, 1905.

calcium carbonate, in order to form silico-aluminate, which is separated by filtration. Aluminum hydroxide is then precipitated from the cleared solution by carbon dioxide, in the usual manner.

In the process patented by the Chem. Fabr. Griesheim Elektron[1], bauxite is heated to a temperature between 180 and 200 deg. C. in an open vessel, and a strong, boiling hot solution of potassium hydroxide (in the proportion indicated by Al_2O_3:Na_2O, or $K_2O = 1:1.3$ or 1.5) is gradually added. The heating is continued for half an hour or more, after which the potassium aluminate thus formed is extracted and treated as usual. Sodium hydroxide may be substituted for the potassium hydroxide, but in this case a final heating to about 500 deg. C. is necessary.

[1] French patent No. 367,728, July 4, 1906.

BISMUTH.

So far as we have been able to learn there was no production of bismuth ore in the United States in 1907. In 1906 Leadville, Colo., produced about 8300 lb. of ore, but none was shipped. In 1907 also there were no sales, the producers being unable to secure any satisfactory offer for the ore that has been in warehouse for over two years. In view of the difficulty of marketing bismuth ore as such the miners have let it go with their ordinary product to the silver-lead smelters. Up to 1907 metallic bismuth had never been produced in the United States, wherefore the consumption is represented by the imports, which are given in the accompanying table.

IMPORTS OF BISMUTH INTO THE UNITED STATES.

Year.	Pounds.	Value.	Av. per lb.
1896	124,263	$ 90,950	$0.73
1897	151,374	172,236	1.14
1898	137,205	162,846	1.19
1899	176,668	208,197	1.18
1900	180,433	246,597	1.37
1901	165,182	239,061	1.45
1902	190,837	213,704	1.12
1903	147,295	235,199	1.60
1904	185,905	339,058	1.82
1905	148,589	318,007	2.14
1906	254,733	318,452	1.25
1907	259,881	325,015	1.25

The production of metallic bismuth as a by-product from the electrolytic refining of lead has been contemplated by the United States Metals Refining Company, of Grasselli, Ind., for a year or more, but has not yet been begun. However, there was a production in the United States in 1907 of a little more than 10,000 lb. of metallic bismuth by another company which treated ore from Sonora, Mexico, and refined the crude metal by an electrolytic process, the details of which are kept secret. The refined bismuth was 99.85 per cent. pure. The outlook for an expansion of this industry in 1908 is said to be bright.

Heretofore the production of metallic bismuth has been in the hands of Johnson, Matthey & Co., of London, the Royal smelters at Freiberg and Oberschlema, in Saxony, and the Deutsche Gold and Silberscheideanstalt, of Frankfurt. The Anhaltische Blei-und Silberwerke, of Anhalt, Germany, attempted to enter the business in 1905, but was driven out by the combination of the three older producers which made a drastic cut in the price of the metal for that purpose. Even at the price ruling in 1906 and 1907 the production must have been very profitable to the combination. Ore

containing 10 per cent. bismuth is paid for at the rate of about 10c. per lb. of bismuth contained. A few years ago the price of bismuth was $2.25 @$2.50 per lb., the metal in ore was paid for at the rate of 25c. per lb. for 10 per cent. ore, and as high as $1 per lb. in ore containing 30 per cent. bismuth and upward. Further data as to the value of bismuth ore are given in the article by Mr. Darroch, which is appended hereto.

The usual number of new bismuth discoveries was reported in 1907, but in general these are untrue. Thus, a mine in Ouray county, Colo., was reported to be producing rich ore, but the owner of the property informs us that there are no bismuth-producing mines in the county. The Euclid Mining Company, in the Whetstone mountains, Arizona, is said to have found bismuth ore, and quartz containing bismuthinite and native bismuth is said to have been found on Iron creek, Custer county, S. D., but we have been unable to verify the reports as to either of these occurrences. The discovery of native bismuth in alluvial gravel at the mouth of Charley creek, a tributary of the Sinrock river, in the Seward peninsula, 35 miles north of Nome, was also reported. We have seen samples of native bismuth alleged to have come from alluvial deposits in the vicinity of Nome, but have been unable to verify the occurrence. According to H. C. Beeler, State Geologist of Wyoming, ore assaying 50 to 65 per cent. bismuth has been mined at Jelm Mountain in that State. Nothing further has yet developed from the previously reported occurrences in California and Idaho.

Bismuth ore occurs at Carbo, Sonora, and a company was formed in 1907 to work it. Samples of ore that we have had from that region have been oxidized material, rather low in grade.

Arizona (By Wm. P. Blake).—Bismuth, chiefly as oxide and carbonate, occurs at several places in the mountains bordering the Salt River valley, but although under exploitation for nearly a year by a German company it has not yet become a commercial product.

Market.—The price of bismuth, which is established at London, experienced only one change in 1907. Up to July 4 it was 5s. per lb. On that date it was changed to 6s. 6d. per lb., which continued during the remainder of the year. In the New York market for bismuth preparations prices ruled steady throughout the year at $1.50@1.55 for subnitrate, $1.75 @1.80 for subcarbonate and $1.85@1.90 for subgallate. There were no important developments in the market for the metal, and, with manufacturers of the various preparations working along harmonious lines, the situation was devoid of new features. Business was of fair average volume.

Assay of Bismuth.—According to L. Moser (*Zeit. anal. Chem.*) the only volumetric method that has proved satisfactory is that of Rupp and Schaumann (*Analyst*, 1903, xxviii, 17). The accuracy of this method

depends largely upon the concentration of the different solutions; for, if a certain degree of dilution be passed, the bismuth salt undergoes hydrolysis, forming a basic bismuth chromate, with the result that too large an excess of chromic acid is found in the filtrate. Good results are obtained when 20 c.c. of $\frac{N}{10}$ bismuth nitrate solution are treated with 50 c.c. of N-potassium chromate solution, and $\frac{N}{2}$ thiosulphate solution used for the final titration. Riederer's method of precipitating the bismuth as molybdate (*Analyst*, 1903, XXVIII, 326-327) will also yield accurate results if the permanganate solution be standardized on a bismuth solution of known strength. It is, however, so tedious that in practice it offers no advantage over the gravimetric methods.

The Occurrence and Metallurgy of Bismuth.

By James Darroch.

Ores of bismuth are not numerous, nor do they occur in large deposits, while unlike many other minerals they do not appear to be widely distributed. Possibly, the fact of their being found in so comparatively few localities is due to the small amount of interest taken in the metal by prospectors, who are mostly engaged in searching for other minerals. Several of the bismuth minerals resemble ores of the better-known metals in appearance, and since prospectors are accustomed to the large masses in which the latter occur, they might pass over as unworthy of attention a small deposit of bismuth discovered by them. Thus, occurrences of bismuth minerals are likely to escape notice altogether, or if met with, they may be allowed to remain undeveloped, to become eventually forgotten by the discoverer, who, ignorant of the nature and value of his find, would consider it a waste of time to trouble about such an apparently unimportant deposit. In any case, it is certain that bismuth does not occur in many regions noted for their mineral wealth, while in only a few districts is it found in payable quantities.

The Ores of Bismuth.—The ores from which the larger part of the world's supply of bismuth is derived are native bismuth and bismuth glance, the oxide and carbonate ranking next in importance. Minerals of less importance are bismuth oxychloride, tetradymite (telluride of bismuth), copper-bismuth glance, silver-bismuth glance, and the corresponding compounds of bismuth with cobalt and nickel.

Native bismuth, the most valuable bismuth mineral, is of a grayish black color, sometimes tarnished yellow. Freshly broken faces exhibit a white metallic luster with a characteristic pinkish tinge. It occurs occasionally in rhombohedral crystals, but is more commonly found in lamellar or granular masses. Native bismuth is an extremely brittle and sectile mineral, having a specific gravity of 9.6 to 9.8.

Bismuthine, or bismuth glance (Bi_2S_3), is a very brittle, sectile mineral, which occurs usually in prismatic crystals, often very delicate and strongly striated; the massive and granular forms also occur, but more rarely. Bismuthine is of a light lead-gray color, yellow when tarnished (bismuth ocher); its specific gravity varies from 6.4 to 6.6. Bismuthine generally occurs associated with other ores of the metal, native bismuth being the ore most frequently found accompanying it.

The oxide, bismuthite, bismuth ocher (Bi_2O_3), occurs as an earthy mineral, the color of which varies from white to yellow. It may be massive, disseminated or pulverulent in form, and is sometimes found foliated. The natural mineral is not crystallized. The specific gravity of bismuthite is about 4.36.

Bismuth carbonate (Bi_2CO_5) is a heavy mineral of a bright yellow to dark gray or blackish brown color, sometimes almost black. It occurs massive, and in spherical forms with a concentric and fine fibrous radiated structure; sometimes also it is pseudomorphous after stibnite. The mineral is soft and easily crushed; its specific gravity varies from 7.3 to 7.4. The native metal is sometimes found accompanying the carbonate.

Bismuth also occurs as a secondary deposition, replacing antimony in tetrahedrite.

The production of bismuth from the oxidized ores is small in amount compared to that derived from the native metal and the sulphide, while the quantity obtained from the other minerals enumerated above is insignificant.

Occurrence.—The ores of bismuth are rarely found alone, being usually associated with other minerals, more especially with those of iron, antimony, arsenic, lead, copper, nickel and cobalt. They occur in quartz and feldspar veins, contained in slates, porphyry, granite and gneiss. The mineral is also found disseminated in slates, gneiss, granite and other crystalline rocks.

By far the largest quantity of bismuth is derived from ores mined in Bolivia. Germany is the next most important producer, while Australia takes third place, New South Wales and Queensland furnishing most of the ore. Small quantities of bismuth ore are produced by the United States and by Austria-Hungary. Elsewhere, the production is intermittent and of little moment. The principal smelting works are in England and Germany.

Mechanical Concentration.

The chief difficulty that presents itself in the metallurgy of bismuth is the great volatility of the metal, which makes it essential that the slags be fusible at a very low temperature. This condition necessitates the use of expensive fluxes, and precludes the employment of the blast furnace

for the reduction process. The calcination of bismuth ore is also rendered more difficult by the volatility of the metal and its sulphide, for unless considerable care is exercised, excessive volatilization losses inevitably take place during the roasting process. Fortunately, however, the oxide is not nearly so volatile, and as it is quite stable at all temperatures, there is practically no danger of loss after the bismuth has been converted to that state. The difficulty in dissolving bismuth and its compounds and the necessity for using strong acid solutions, restricts the applicability of wet methods of extraction, for not only do these properties increase the cost of treatment, but the first makes the use of large leaching vats impracticable, it being an essential condition that the solutions should be kept hot while they are in contact with the ore.

Dressing.—Another obstacle is the difficulty of dressing the lower-grade ores to a sufficiently high standard to make their subsequent treatment profitable. Although it might appear from the great density of bismuth minerals that they would be eminently suitable for concentration by the ordinary mechanical processes, this unfortunately is not the case. Owing to the exceeding brittleness of the minerals, they slime very readily when crushed, and being reduced to a much finer state of division than the gangue, a very heavy loss of metal inevitably occurs in the dressing process. The adoption of slime concentration would not do much to remedy matters, as fine pulp is very hard to dress, and besides, a further objection presents itself, viz., the excessive loss by dusting and volatilization that will take place during the calcination of such fine material. For these reasons, and because of the small size of most of the mines, the mechanical concentration of bismuth ores has been neglected in the past. During recent years, however, great advances have been made in the art of mechanical dressing and possibly some of the newer processes, which are securing good results with other ores that previously could not be worked advantageously even by the best methods of concentration in vogue, might be applied to the treatment of bismuth ores. When the gangue is of a suitable nature, magnetic separation is quite effective with bismuth minerals, and that method of dressing has already been successfully applied in some instances.

Concentration by Liquation.—At one time low-grade ores containing bismuth sulphide and the native metal were concentrated by liquation; indeed in the case of native bismuth this was the only means adopted for its recovery, but the method was a wasteful one, and it has now been entirely abandoned.

Hand-Picking.—It is desirable, however, that the ore should be concentrated to as a high a degree as possible before it reaches the smelting furnace, and generally the best way of accomplishing this is to adopt an efficient system of hand-picking. If the mineral is thoroughly dissemi-

nated through the gangue, a mere rejection of the larger lumps of barren silicious material that may have become mixed with the ore will do much toward raising its average value, while the saving in cost of fluxes will more than pay for the additional labor required. The sorting must be performed carefully, because the value of the metal makes it of importance that no ore which would pay to treat should be thrown away, while on the other hand it is imperative that all unpayable stone, especially if it is silicious, be discarded. The large lumps which, from their appearance, contain no mineral should therefore be broken up and examined; if proper care has been observed in sorting, the smaller pieces may generally be rejected without subjecting them to a second inspection.

The best plan is to sort on a belt conveyer, which must travel slowly; otherwise the workers cannot discern the unpayable portions as they are carried past. For the same reason the ore should be spread in a very thin layer on the belt, and all pieces under $\frac{3}{4}$ in., which is the smallest size that can be distinguished with any degree of accuracy, should be screened out before it is fed to the conveyer. The sorters should confine themselves to picking out lumps of certain sizes, those stationed at the head taking the largest fragments, and so on until the discharge end of the belt is reached. Here the finest material is sorted, and as much experience and sharpness are required to detect the unpayable pieces among it, the best workmen should be placed at this point. When the larger lumps are separated first, the workers are given a better opportunity of discharging their duty properly, because not only are the smaller fragments better exposed to view, but the labor of sorting is greatly lightened, for it is easier to concentrate the attention on pieces of one size than on material of varying size.

The ore should be sorted whether it is intended to treat it on the spot, or to ship it for treatment or sale; in the latter case not only are better prices per unit realized, but a considerable saving in freight charges is effected.

It is assumed that the proportion of payable material exceeds that of waste, but should the reverse be the case, it would, of course, be more economical, and at the same time much better practice, to sort out the mineral-bearing portion, leaving the remainder on the belt to be conveyed directly to the dump.

With suitable ore, and when not performed in a perfunctory manner, as is too often the case, hand-picking is a very effective method of separating the valueless portions; and where operations are conducted on a small scale, it is probably less expensive than any process of mechanical concentration.

The following hypothetical case will illustrate the benefit that may be obtained from sorting when handling suitable material: 100 tons of ore aver-

ages 5 per cent. metal, but it is possible to separate 40 tons of which the average value is 0.75 per cent., thus raising the average grade of the remaining 60 tons to $7\frac{5}{8}$ per cent. Assuming that it is possible to obtain an extraction of 85 per cent. in the case of the unsorted ore, and of 90 per cent. when dealing with the richer stuff, the amount of metal recovered is practically the same in both instances. That is to say, there is no gain from treating the 40 tons of poor ore. It might seem almost unnecessary to give an illustration, but in many cases the reduction in expenses per ton that is secured by increasing the scale of operations, combined with the desire to handle a large tonnage, leads to the treatment of unprofitable material. This apparent saving is very deceptive, therefore it must be guarded against; for unless the low-grade ore of itself contains sufficient values to yield a profit after paying the cost of treatment, besides providing interest, sinking fund and depreciation on the additional plant required to effect its reduction, the reduced operating cost per ton is bought very dearly indeed.

When small tonnages are handled it is best to do the preliminary breaking by manual labor, for a crushing plant is at great disadvantage, unless it is run at something approaching its rated capacity. Where the cost of labor is low and fuel is dear, hand spalling is certainly no more expensive than breaking by machinery except when operations are conducted on a large scale. As practically every fragment of stone comes under the observation of the spall-men as it passes through their hands, a good deal of sorting can be done at this stage; in fact, under certain conditions, the whole of the sorting might be done in the spalling shed. Certainly the men should be instructed to knock off and pick out every piece of barren rock observed by them. Of course if sorting is done the cost of breaking will be considerably greater, but the saving in fuel and fluxes will more than offset the additional expenditure; moreover the smaller quantity of slag produced carries away less metal in it.

Roasting.

To prepare the ore for roasting it must be further reduced in size. This can be accomplished most economically by passing it first through a light breaker of the reciprocating roll-jaw type, and finally through rolls or some similar machine. The machine installed for the final pulverization should be one that produces a minimum proportion of fines, as these are troublesome, and cause loss of metal, both mechanically and chemically, during the calcination process. Of course, if the ore does not require roasting, the proportion of fines in the final product is of no consequence, in fact their presence may even be advantageous, because it allows of a more intimate mixture of ore and fluxes in the smelting charge. Only by actual experiment is it possible to determine the degree

of fineness to which a given material requires to be reduced when intended for calcining purposes. The factors which should be considered are: The cost of crushing; state of oxidation of the iron present; the amount of residual sulphur in the product; the loss of metal by dusting and volatilization; and the degree to which arsenic and antimony have been removed during the calcination. The last three are of the greatest importance; the first two for obvious reasons, and the other because in the smelting process arsenic and antimony are reduced and alloy with the bismuth, thus considerably detracting from its value.

When the ore to be roasted is of a silicious nature it may be necessary to crush to 12-mesh or even finer, but if the proportion of sulphides is large, or the gangue porous or friable, 4-mesh is probably fine enough, while if the material is one that decrepitates when heated no reduction in size beyond that effected by the breaker may be required, in which case the fine crushing plant may be dispensed with.

Roasting Furnaces.—Bismuth minerals, like those of lead, do not lend themselves to roasting in heaps and kilns, consequently the operation must be conducted in a reverberatory furnace. The ordinary long hearth reverberatory, which is cheap and can work any sulphide charge successfully without requiring very highly skilled labor, is the most suitable furnace. The quantity of ore to be calcined of course determines the size of the roasting furnace. One with a hearth 16 or 17 ft. long and 9 ft. wide will treat from five to six tons per day; if the hearth is 16 ft. wide 10 to 12 tons can be roasted. When the quantity of sulphur in the ore is large, two or three additional hearths will be required. A long furnace is best if large quantities of arsenic and antimony have to be eliminated, as the low initial temperature greatly favors their expulsion.

The advantages of multiple hearth reverberatory calciners are so obvious, and have been so often and so ably enumerated in works dealing with the metallurgy of copper and lead, that their recapitulation here is unnecessary, and the same may be said of their construction, which has been described by many competent authorities. The roasting process has also been frequently described, and only one or two special points need be referred to. These are the constant rabbling that is required to prevent the very fusible charge from agglomerating, and the necessity for getting rid of arsenic and antimony. The reduction of arsenates and antimonates is effected in the usual manner, i. e., by adding small quantities of crushed coke or charcoal to the charge during the final stage of the roasting process, the temperature being at the same time raised to the highest practicable limit. When all the added carbonaceous matter has burnt off the charge may be withdrawn from the furnace. Bismuth trioxide is somewhat less fusible than lead oxide, consequently if the temperature is kept within the limits observed when roasting ores of

the latter metal the charge will be safe from sintering, which it must not be allowed to do, as the final product is required in the pulverulent state.

Chemistry of Roasting.—No detailed description of the chemistry of the roasting process is necessary. Arsenic and antimony are volatilized partly as unaltered sulphides, and partly as oxide, while a third portion is converted to arsenates and antimonates of the different metals present. By the addition of carbonaceous matter to the charge the latter compounds, which are stable at high temperature, may be reduced to the lower oxides and volatilized. The presence of steam during the roasting process is of great advantage in furthering the elimination of arsenic and antimony, as it reacts upon their sulphides in accordance with the following equation: $As_4S_6+6H_2O=2As_2O_3+6H_2S$, the elements being left in the most favorable condition for volatilization, i.e., as arsenious and antimonious oxides. Part of the hydrogen sulphide may burn to sulphur dioxide and water, in which case a further supply of steam is available. As the reaction is endothermic, the assistance of external heat is essential to its taking place. The cheapest and most certain method of ensuring the presence of water vapor in the atmosphere of the furnace is to dampen the fuel, or occasionally to throw a pailful of water into the ash-pit.

When subjected to an oxidizing roast bismuth sulphide is converted partly to oxide, and partly to sulphate; as the latter is only imperfectly decomposed by increasing the temperature, the result of calcination is invariably a mixture of these compounds of the metal. If water vapor is passed over red-hot bismuth sulphide, a reaction similar to that given above for arsenic results.

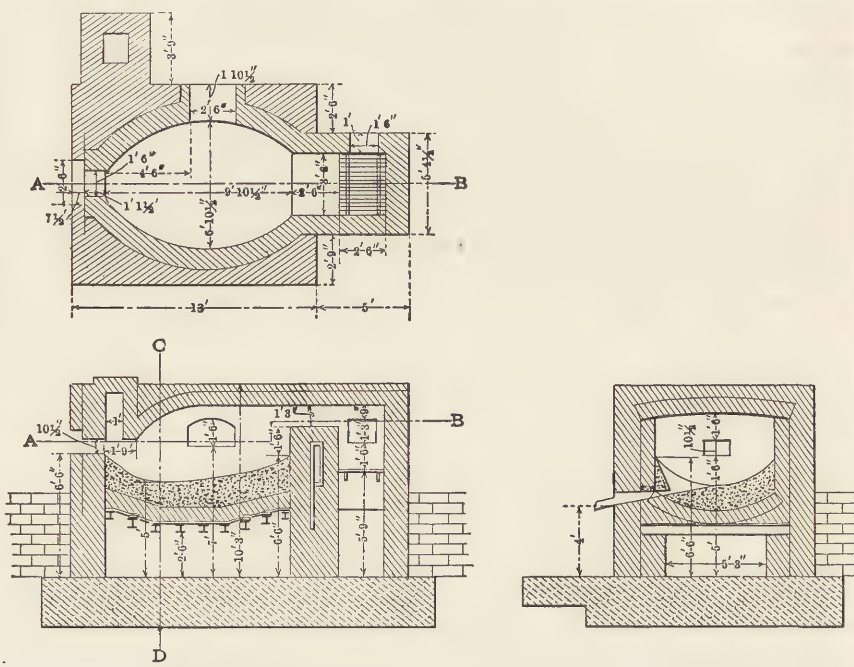
Unless operations are conducted on a very limited scale indeed it will be found profitable to adopt means for the recovery of flue dust and fume formed during the process of calcination. As this material will contain much arsenic and antimony, if those elements are present in the ore, it should not be mixed with the roasted product, but should be collected until a sufficient quantity accumulates to form a charge for the smelting furnace.

Smelting.

Crucible Method.—Bismuth ores are smelted in crucibles and in reverberatory furnaces. The former are to be recommended only where very small quantities of ore are handled, as they labor under several disadvantages, the most important of which are the small output in comparison to the cost of plant and number of men employed, and the necessity for skilled labor in their manufacture, or if they are purchased, their high cost, which is considerably augmented by transportation charges

and breakage. Those who are interested in crucible smelting will find in THE MINERAL INDUSTRY, Vol. VIII, a fully illustrated description of a gas-fired furnace specially designed by W. Borchers (the writer of the article referred to) for the smelting of bismuth ores. This furnace, which seems to be admirably adapted for its purpose, allows of operations being conducted on a fairly large scale, and without interruption, and its construction is such as to economize fuel to the greatest possible extent. Altogether it appears to meet the requirements very well indeed.

Reverberatory Smelting.—However, when the scale of operations permits it reverberatory furnaces are to be preferred to crucibles, owing to the



Longitudinal Section Cross Section

FIG. 1.—BISMUTH SMELTING FURNACE.

simplicity of their construction, and the saving effected in initial cost and labor charges. Any small reverberatory furnace of the types employed in lead or copper smelting may be adapted to the treatment of bismuth ores. Because of the great fluidity of the molten metal, however, and the readiness with which it penetrates the walls and hearth of the furnace, it is advisable to employ a furnace specially designed for the purpose, as considerable (temporary) loss of metal, and much unnecessary expense may be obviated in this way. Either of the furnaces shown in the diagrams will be found suitable for the smelting of bismuth ores. Their capacity would be about 10 or 12 tons of charge per day, but the dimen-

sions may be considerably altered to suit varying conditions without impairing their usefulness.

The illustrations show the construction of the furnace sufficiently well to make a detailed description unnecessary. The walls supporting the roof are built of fire-brick, strengthened by an outside casing of red brick, the whole being stayed by iron buckstaves and tie-rods. Inside the walls is a lining of fire-brick. It will be observed that both the lining and hearth are quite independent of the walls, thus enabling them to be renewed without interfering with the latter. The hearth is supported by iron plates at each end, and by iron plates and girders below. Every part of the furnace exposed to the heat must be constructed of very refractory materials, laid in silicious mortar.

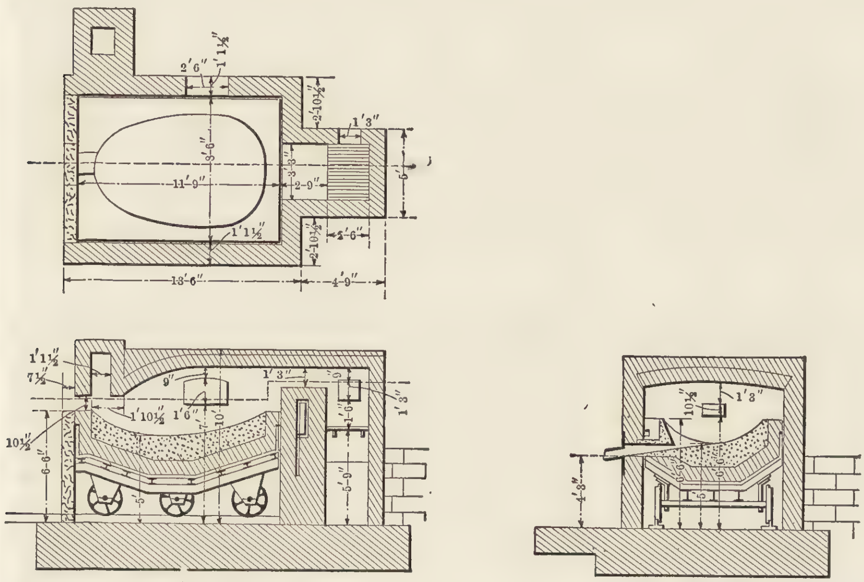


FIG. 2.—MOVABLE HEARTH FURNACE.

After the brickwork has thoroughly dried a working bottom is put in. This consists of successive layers of sand, which are spread over the hearth and fritted together, each layer being more refractory than the preceding one. The final layers must be of such refractory material that it just vitrifies at the highest temperature of the furnace. When the bottom has been completed it is saturated with slag, and is then ready for work. The taphole aperture is filled with ganister from the inside, a hole being cut from the outside when the bottom is nearly completed.

The movable hearth is an invention of W. Borchers. As before, the hearth is supported by iron plates and girders, only these are set upon

wheels, so enabling the whole to be removed from the furnace when worn out and another substituted. The rails should be laid level as shown in the illustrations, otherwise the hearth will have a tendency to run out of the furnace. After the hearth has been placed in position, the front of the furnace must be built up to exclude currents of air from the interior. A working bottom is smelted in, and the taphole opening filled up and the hole cut as previously described. If this were not done a current of cold air, which would be likely to cause cracks in the brickwork and bottom of the furnace, would be admitted at the taphole.

Composition of Charge.—Owing to the readiness with which bismuth volatilizes at high temperatures, the smelting mixture must be composed of very fusible materials. It is therefore necessary to employ fluxes which when combined with silica produce slags that melt at a low temperature. The silicates of the alkaline metals are exceedingly fusible, and this property is utilized in the smelting of bismuth ores. As is well known, however, a combination of silicates is generally much more fusible than a single silicate, while the addition of old slags facilitates the fusion of materials which have not been previously melted. Therefore by a judicious mixture of fluxes and old slag, a much more economical slag may be produced, while its metallurgical qualities need not be impaired in the least. The fluxes employed in bismuth smelting are sodium carbonate, oxide of iron, oxide of manganese, lime, and fluorspar.

Sodium carbonate is an essential flux in bismuth smelting because of the great fusibility and fluidity of the silicates of sodium, whether alone or in combination with silicates of other metals. As soda is expensive it must not be used too lavishly, but unless a sufficient quantity is present in the charge, the loss of bismuth will be excessive, especially if lime is resorted to to make up the deficiency. The smelting mixture should contain from 12 to 20 per cent. of this flux. Oxide of iron can replace soda in all proportions, but it is impracticable to run a purely iron slag, since heavy losses of bismuth would result from the adoption of that course. Iron silicates are very fluid when molten, but they possess the disadvantage of high specific gravity; therefore they should not be present in too large a proportion. As manganese silicates are even more fluid than ferrous silicate, manganese oxide may be advantageously substituted for iron oxide.

The cheapness of lime, its efficiency as a flux for silica, and the fact that it reduces the density of the slag make it a desirable flux to employ. Unfortunately, however, calcium silicates are fusible only at high temperatures, consequently only a relatively small proportion of lime can be used. As silicates of magnesia are most refractory, the limestone flux must not be dolomitic in character. The efficiency of fluorspar as a flux for silica, and its power of dissolving barium and calcium sulphates,

and also its fusibility, make it a desirable addition to the charge. Unless it is easily obtainable, however, it may be dispensed with in most instances.

In the case of very basic ores, silica is the flux employed, soda being added for the purpose of reducing the fusion temperature of the slag.

Old slags are added for the purpose of recovering metal contained in them, and to assist in the fusion of the charge. When rich ores are being treated only small quantities of slag are produced and there is a liability of losses of metal occurring by volatilization. In such cases old slags may be employed in considerable quantities to form a protective covering for the bismuth.

The Smelting Operation.—The furnace charge will therefore consist of a mixture of ore, soda, oxide of iron, lime, old slags, and from 3 to 5 per cent. of crushed coke or charcoal. The last material is employed as a reducing agent. Before it enters the furnace the whole must be thoroughly mixed. When the furnace is ready the charge is shovelled in, covered with a layer of old slag, and the temperature is brought up to a bright red heat, more ore and flux being added to fill the hearth as the charge melts down. Shortly after fusion bubbles of gas begin to make their appearance on the surface of the charge. At first the gas is composed almost wholly of carbon dioxide derived from the decomposition of sodium carbonate, but during the later stages it consists partly of carbon monoxide and carbon dioxide, formed by the interaction of bismuth and other metallic oxides with carbon, and partly of sulphur dioxide from the oxidation of sulphides. At the temperature of the furnace, sulphuric acid is unable to exist, and that derived from the reduction of sulphates is decomposed into sulphur dioxide and oxygen. The escaping gas gives the charge the appearance of boiling, and globules of metal may be seen coming to the surface, swirling around and again disappearing. When the reduction of bismuth oxide is complete, the boiling subsides and the contents of the furnace become tranquil. The reaction is now over, and the temperature is raised as rapidly as possible to a white heat. As soon as the contents of the hearth are perfectly fluid, the taphole is forced open and the molten material runs into iron molds in which the different constituents separate according to their specific gravities. The molds are best made of a conical shape, and they may be provided with a taphole at the bottom to allow of the bismuth being run out after the slag has solidified.

During the whole period of fusion the atmosphere of the furnace should be kept as strongly reducing as possible.

The products of smelting depend largely upon the metals present in the charge. Lead and antimony are reduced and alloy with the bismuth. The greater part of the copper goes to form a matte, but part of it is

reduced and enters the bismuth. Arsenic forms a speiss in which any nickel and cobalt that may be present is found. A part of the arsenic is reduced and alloys with the bismuth. Gold and silver are concentrated chiefly in the bismuth. The results of fusion may therefore be bismuth, matte, speiss and slag. The matte and speiss contain appreciable quantities of bismuth.

The basic slags produced in the smelting process, especially if they are high in soda, attack the bottom very severely, consequently after every charge the bottom must be examined and repaired with sand wherever necessary. This takes a good deal of time, but unless it be done the bottom will be rapidly destroyed, and much metal will leak into the hearth and walls, the eventual result being the total destruction of the furnace. This would appear to indicate the desirability of employing a basic lining, which, so far as I am aware, has not been tried in bismuth smelting. There is no apparent reason, however, why it should not prove successful. Either magnesite or calcined dolomite might be employed. It would be necessary to make a more basic slag than is produced in the silica lined furnace, but this would be no disadvantage, and could easily be accomplished. With suitable slags a well made bottom of magnesite or dolomite would be very durable, and consequently much less expensive to maintain than those at present in use.

Reactions in the Furnace.—The smelting process is essentially one of calcination and reduction, and if pure ore is being treated, the reactions that take place in the furnace resemble those occurring in lead extraction under similar conditions. When complex ores are being treated, however, the chemistry is greatly modified, as other metals and their compounds exercise considerable influence in the process of reduction. The bismuth oxide is reduced by carbon contained in the charge, by carbon monoxide from the furnace gases, by interaction with bismuth and other sulphides, and by metallic lead. Any bismuth sulphate present is reduced to sulphide, of which a part is reduced to metal by metallic oxides, and by copper and iron, the remainder going into the matte along with copper and iron sulphides. Lead oxide is reduced to metal, which in turn reduces bismuth oxide, the lead oxide formed being again reduced by carbon or by lead sulphide. Copper oxide is reduced to metal, the most of which immediately combines with sulphur and forms matte, but a small quantity invariably enters the bismuth. Oxides of arsenic and antimony are reduced, the former combining with iron to form speiss, in which any nickel and cobalt that may be present is found, while the latter element mostly passes into the bismuth. In the presence of carbon, ferrous oxide to a certain extent acts as a reducing agent, being itself converted to sulphide, in which condition it goes with the copper sulphide into the matte. The gases evolved by bubbling through the charge probably assist the

metal and other products of fusion to settle out, and possibly the carbon dioxide may exert some influence as an oxidizing agent.

Precipitation Smelting.—The precipitation process formerly employed in lead smelting is quite suitable for the treatment of sulphide ores of bismuth, which in this case are smelted without previous calcination, metallic iron being used as the reducing agent. The process possesses several advantages, among which are its rapidity, and the saving in cost of roasting furnaces and labor charges. Unfortunately, the very basic slags that are made, and the large quantity of ferrous sulphide formed during fusion attack and quickly destroy the hearth and lining of the furnace. This has caused the process to be abandoned.

The slags produced in the reduction process consist chiefly of polybasic monosilicates, mixtures of monosilicates and sesquisilicates, and sometimes sesquisilicates. The last type is the most silicious that can be employed with advantage, as the fusion point of more acid slags is so high that loss of bismuth by volatilization is certain to occur if the temperature is raised to a sufficient degree to melt them. In the precipitation process a subsilicate of indeterminate composition is made.

Refining.—Only under exceptional conditions is the bismuth produced in the smelting process sufficiently pure to make refining unnecessary. However, since bismuth possesses very little affinity for oxygen, its purification presents no great difficulty. The method adopted to purify the metal is exactly similar to that employed in the softening and refining of lead, namely, exposure of the molten bismuth to the oxidizing influences of the atmosphere. Tin, arsenic, antimony, sulphur, zinc, copper, nickel, cobalt, iron and lead all have a greater affinity for oxygen than bismuth, and this property is taken advantage of in the refining process. Lead is partly separated from bismuth by the Pattinson process, the latter metal being concentrated in the residual silver-lead alloy, while a complete separation is effected by cupellation. The process is as effective in recovering gold and silver from bismuth as from lead bullion.

Hydrometallurgy.

The wet process that has been most generally adopted for the extraction of bismuth is solution of the oxidized ores in hydrochloric acid. The finely crushed ore is treated in earthenware vessels with acid, heat being applied to expedite solution. Afterward, the bismuth is precipitated by iron, or by diluting the concentrated solutions with a large excess of water, when it is obtained as oxycholoride, from which metallic bismuth is thrown down by iron or zinc. In other cases the oxychloride is mixed with lime to prepare it for smelting, the products being bismuth oxide and calcium chloride. The method is applied also for the recovery of bismuth from mattes and other sulphurous products, including

occasionally sulphide ores; these materials, of course, require calcining before treatment.

A wet method is employed in the treatment of alloys for bismuth. Hot concentrated sulphuric acid, or aqua regia, is used to effect their solution, the bismuth afterwards being precipitated by one of the methods already mentioned.

Market Conditions.

The principal European bismuth smelters are members of an association which regulates the production and price of the metal. This association, which is said to control several of the most important Bolivian mines, is very strong, and at present completely governs the market both for metal and ore. As might be expected in the circumstances, there is practically no competition among buyers, and, of course, the market for ores is a very restricted one. The lower grade ores are not in demand, and although under certain conditions a 5 per cent. ore is saleable, those containing less than 10 per cent. of the metal do not readily find a market. It is therefore advisable to dress the ore to 10 per cent. as a minimum, and to 20 per cent. if at all feasible. The prices for ore are fixed by assay, and they are based upon the market price of the metal. Of course the price per unit is greater in the case of high grade ores. Taking the price of a 10 per cent. ore as unity, that of a 15 per cent. ore is represented by 1.7, of a 20 per cent. ore by 2.3, of a 30 per cent. ore by 3.7, of a 40 per cent. ore by 5, and of a 50 per cent. ore by 6.7. At the present price of the metal, viz., $1.60 per lb., unity is equivalent to about $90 to $100 per metric ton. These prices are net, there being no charge for treatment, but sellers must bear the cost of transportation to the smelting works. Very complex ores are much more costly to treat than those that are comparitively free from impurities; consequently they are not so valuable as the latter. Ores containing gold and silver are not attractive to the smelter unless an additional profit accrues from the treatment of those metals. Sellers must therefore be prepared to pay a treatment charge, or to accept less than the full value of the gold and silver contents of the ore. In a limited market the allowance to be made for gold and silver in the purchase of ores is a matter for arrangement between buyer and seller, but the terms offered are generally as favorable as could be obtained elsewhere. In this connection it has to be considered that the bismuth also is paid for, and that the value of that metal would be lost if the ore were sold to a copper or lead smelter.

The extraction secured when treating ores by the fusion method varies from 85 to 95 per cent. of the bismuth content. Much depends upon the nature and richness of the ore, and on the state in which the bismuth is present. The conditions being equal, it is obvious that a better extraction

can be obtained from an oxidized ore than from a sulphide, since the latter requires to be calcined before treatment, and it is during the roasting process that a considerable part of the volatilization loss occurs. Again a material which is almost self-fluxing, requiring only the addition of sodium carbonate to form a perfect smelting mixture, is more favorable to a high extraction than a very silicious ore, or one containing a large excess of bases. In the first case a smaller quantity of slag is produced, consequently less metal is carried away by that material. As a rule, however, an extraction of 90 per cent. may be expected from ore containing 10 per cent. of bismuth, while in the case of richer ores an even better saving can be made. The metal contents of the ores usually smelted range from 10 to 50 per cent., the average being about 20 or 25 per cent. Exceptionally ores of lower grade than 10 per cent. are smelted, but smelters prefer the richer ores, since greater profit accrues from their treatment. The German ores probably average much less than 10 per cent. of bismuth, but they contain considerable cobalt and nickel which are also recovered.

Bismuth, like lead, acts as a collecting agent for the precious metals, and since the conditions are especially favorable to a high recovery of these metals, the extraction of gold and silver is very perfect, particularly if neither matte nor speiss is made. Practically all of the gold, and 97 per cent. or more of the silver can be secured. Copper and arsenic in themselves do not adversely affect the extraction of gold and silver; it is during the re-treatment of the matte and speiss that a loss occurs. If copper and arsenic are present, there is a greater liability of slag losses taking place, since the density of speiss, and especially of matte, is less than that of bismuth; consequently these materials do not settle out from the slag so readily as does the latter. In any case the recovery of gold and silver should not be less perfect than at a copper or lead smelter.

BORAX.

By Edwin Higgins.

The production of borax in the United States is confined almost entirely to Inyo, San Bernardino and Ventura counties, California. Although there is enough borax developed in these counties to supply the United States for many years to come, competing companies have paid good prices for proved deposits; consequently there was much activity in prospecting during 1907. Discoveries of borax were reported from various points in the State, but little importance is attached to most of them. The presence of ludwigite, a borate of iron and magnesia, associated with magnetite and manganese ores, was reported in the vicinity of Phillipsburg, Mont. The only important occurrrence of this rare mineral, heretofore recorded, is at Morawitza, in the Banat, Hungary. The accompanying table shows the production of borax in the United States for a series of years.

PRODUCTION OF BORAX IN CALIFORNIA. (a)
(In tons of 2000 lb.)

Year.	Tons.	Value.	Year.	Tons.	Value.	Year.	Tons.	Value.
1896	6,754	$ 675,400	1900	25,837	$ 1,013,251	1904	45,647	(c) $ 698,810
1897	8,000	1,080,000	1901	7,221	982,380	1905	46,334	1,019,158
1898	8,300	1,153,000	1902	(b) 17,202	2,234,994	1906	58,173	1,182,410
1899	20,357	1,139,882	1903	34,430	(c) 661,400	1907	....	....

(a) Reported by the California State Mining Bureau. (b) Mostly refined borax whence the apparent discrepancy in value. Output of the other years is given as crude material. (c) Spot value.

The principal refiners of borax in the United States are: Pacific Coast Borax Company, operating a refinery at Bayonne, N. J., and one at Alameda, Cal.; Sterling Borax Company, operating refineries at San Francisco, Chicago and Pittsburg; Charles Pfizer & Co., Brooklyn, N. Y.; and M. Calm & Bro., Jersey City, N. J.

The Sterling Borax Company is a recent consolidation of the American Borax Company and the Frazier Borate Company, of California, the Stauffer Chemical Company, of San Francisco, Cal., the Thomas Thorkildsen Company, of Chicago, Ill., and the Brighton Chemical Company, of New Brighton, Penn.

A newly organized company, the Borax Properties Limited, has entered the California field. This company was organized to acquire and operate the properties of the Palm Borate Company, in San Bernardino county.

The crude mineral on this property is said to average 9.72 per cent. boric acid. It is the intention to build a refinery at the mine. The main office of the company is 68 Palmerston House, Old Broad street, London, E. C.

Industrial Conditions and Prices.—The bulk of the borax mined in California is shipped to Eastern points to be refined. The Pacific Coast Borax Company, which is the largest producer, treats 80 per cent. of its product at its refinery in Bayonne, N. J.; the remaining 20 per cent. is treated at its works in Alameda, Cal. Borax of commerce contains approximately 37 per cent. anhydrous boric acid, 47 per cent. water of crystallization and 16 per cent. soda. The crude material now shipped to Eastern points for treatment contains not less than 30 per cent. anhydrous boric acid, and when the freight charges across the country are considered, it is doubtful if there is much, if any, profit to be realized on material so handled, with refined borax as low as 4½ to 5c. per ℔. The impurities to be met with in much of the California crude material are soluble and insoluble silica, iron, alumina and magnesia; with the exception of the soluble silica, these must be removed by special processes, which of course increase the cost of production. In this respect the borax of Asia and South America is far superior to the California product, containing little or no impurities and running high in anhydrous boric acid.

Under existing conditions, with respect to the attitude of the different companies and the price of the finished product, the independent owner of a borax property is placed in an unenviable position. He must either sell his property outright or build his own refinery, for there is no market for the crude mineral; the companies operating refineries are adequately supplied with crude material from their own properties. The great decline in the price of borax toward the end of 1907 was the result of a struggle for the control of the borax market. The strong competition which was developed also brought about many changes in the control and operation of both borax properties and refineries. The price of borax at New York and Chicago up to the end of August was steady around 7¼c. per ℔. In September the quotation was 6½c. and in December 5½c. per ℔. The decline in price continued into 1908, the quotation in February and March being 4½c. per pound. The prices are quoted on the basis of anhydrous boric acid.

Borax in California.

The mineral colemanite (calcium borate), a white, crystalline substance with a vitreous luster, resembling gypsum somewhat but distinguished from it by a greater hardness, is the most important form in which borax occurs in California. However, other combinations of boric acid, lime and soda are of common occurrence.

Inyo County.—South of the Funeral range, on the Amargosa side of the mountain, occur immense deposits of calcium borate, conformable with the stratification of the country shales, clays, sandstones and thin sheets of gypsum. Borax is found as marsh deposits, and the Amargosa river, flowing southerly across the desert of the same name, is doubtless responsible for most of them.

For many years the Amargosa valley produced great quantities of borax, but now the chief source of supply of the mineral is the Lila C mine, 18 miles west of Greenwater in the foothills of the Funeral range of mountains. Eight miles to the east is Death Valley junction, from which point a spur of the Tonopah & Tidewater Railroad has been built to the mine. Here there is a deposit of calcium borate from 3 to 17 ft. thick, dipping 45 deg. easterly, which has been opened on the surface for about a mile. This property is operated through adit levels. The borax deposits of this locality are noted for their great size and high degree of purity. The completion of the Tonopah & Tidewater Railroad, which is now being constructed from Ludlow, on the Santa Fe, will greatly benefit the borax industry in this section.

SOME OF THE PRINCIPAL SUPPLIES OF BORAX PRODUCTS.

(In metric tons.)

Year.	Chile. (*a*)	Germany (*b*)	Italy.			United States. (*d*)	Total. (*e*)
			Borax Refined.	Boric Acid.			
				Crude.	Refined.		
1897....	3,154	198	990	2,704	260	7,257	14,563
1898....	7,028	230	702	2,650	166	7,529	18,305
1899....	14,951	183	709	2,674	129	18,466	37,112
1900....	13,177	232	858	2,491	283	23,437	40,478
1901....	11,457	184	544	2,558	347	6,550	21,640
1902....	14,327	196	....	2,763	....	15,512	32,798
1903....	16,879	159	....	2,583	....	31,232	50,853
1904....	16,733	135	569	2,624	314	41,407	61,782
1905....	19,612	183	(*f*)1,007	2,700	(*f*)749	42,036	66,287
1906....	(*c*)	161	1,062	2,561	562	52,774	57,120

(*a*) Prior to 1903, figures are for borate of lime exports. (*b*) Boracite. (*c*) Statistics not yet available. (*d*) Crude borax. (*e*) The total falls short of the World's supply, particularly because it fails to include the important production of Turkey. (*f*) Obtained by treating a part of the crude boric acid reported for 1905.

San Bernardino County.—In this county the principal deposits are found in the neighborhood of Daggett, but the supply from these mines has not kept pace with the demand. Here a great deal of the borax occurs in what is locally called the low-grade "muds." It is reported that the American Borax Company, of Daggett, has secured a new and extensive borax deposit in Soledad cañon, near Langs Station.

Ventura County.—There are at present two concerns mining borax on a large scale in the northeastern part of this county, the Columbus and the Frazier companies, each employing 65 men. The mineral, principally

colemanite, is mined and hauled by wagons to Bakersfield. The industry in this section is prosperous and gives promise of growing to larger proportions.

Tulare County.—Four borate claims were located on Deer creek, east of Porterville, and development is progressing.

Uses of Borax.

Of late years borax has been extensively used in the manufacture of porcelain-coated ironware, tough grades of glass, pigments for staining glass, household soaps and preservatives for canned foods. When melted at a high temperature it has the property of dissolving metallic oxides and for this reason is used in assaying and also in soldering, brazing and welding metals. The borate of chromium makes a beautiful green pigment which is used in calico printing. Borate of manganese is sometimes used as a dryer in paints and oils. Borate of ammonia renders cotton goods uninflammable to a certain extent.

Borax in solution is useful in the laundry and for toilet purposes; it is also a constituent of numerous cosmetics. It is effective in cleaning, softening and preventing the hair from falling out of woolen goods and furs.

In addition to the above, borax is extensively used in medicine and for various other purposes in manufacturing.

BROMINE.

The production of bromine in the United States in 1907 was about 1,062,000 lb., against 1,229,000 lb. in 1906. It is impossible to report the production in 1907 by States without disclosing the business of individual producers. However, it may be stated that there was a decrease in the production of Michigan and an increase in that of West Virginia. The product of West Virginia, Pennsylvania and Ohio is sold in liquid form. That of Michigan is sold chiefly in the form of bromides, chiefly the bromide of potassium. Bromide of sodium is also produced, being sold chiefly in the domestic market. Bromide of ammonia, bromide of barium, bromide of iron and bromide of lithium are also made in considerable quantities. The chief producer in Michigan sells a considerable quantity of what is called "mining salt", which is a mixture of bromide and bromate of potash and soda.

PRODUCTION OF BROMINE IN THE UNITED STATES.
(In pounds.)

Year.	Michigan. (a)	Ohio and Penna.	West Virginia.	Total. (a)	Metric Tons.	Value.	
						Total.	Per lb.
1897	147,256	241,939	97,954	487,149	221	$136,402	28c.
1898	141,232	226,858	118,888	486,978	220	136,354	28
1899	138,272	193,518	101,213	433,003	196	125,571	29
1900	210,400	196,774	114,270	521,444	237	140,790	27
1901	217,995	227,062	106,986	552,043	250	154,572	28
1902	226,452	194,086	93,375	513,913	233	128,742	25
1903	320,000	180,000	97,000	597,000	271	170,145	28½
1904	646,249	147,807	85,256	879,312	399	215,431	24½
1905	579,434	223,000	97,000	899,434	408	139,492	15½
1906	955,000	203,000	71,000	1,229,000	553	184,350	15
1907	(b)	(b)	(b)	1,062,000	482	138,060	13

(a) Includes the bromine equivalent of the bromides produced directly. (b) Not reported separately.

Market Conditions.—The average value of the bromine product in 1907 is estimated at 13c. per lb., against 15c. per lb. in 1906. The further decline in price was due to over-production and the continued competition of German bromides, which were imported into the United States in rather large quantity. The contest between the American and German producers appears to be an unending one, the Americans exporting bromine products to Europe and the Germans shipping their products to the United States. At the end of 1907 bromine was sold as low as 10c. per lb. in carload lots, f.o.b. works, which is believed to be below the cost of production to all the makers except the Dow Chemical Company. So low a price enabled

bromides to be manufactured very cheaply. The manufacturing cost of making bromides is said to be from 5 to 9c. per ℔. exclusive of the bromine. The manufacture of bromide of potassium requires about 0.7 ℔. of bromine, while bromide of sodium requires 0.8 ℔. of bromine.

The history of the bromine industry has been a series of agreements between the American and the German producers to control prices, the agreements leading to high quotations and the breaking of them being signalized by a drop in American prices due to aggressive German competition. For several years back, the market has been an open one and the price has fallen from 28½c. in 1903 to the basis of 10c. established at the end of 1907.

Producers.—The producers of bromine in the United States are the following: Pomeroy Salt Association Company, Pomeroy, O.; Coal Ridge Salt Company, Pomeroy, O.; Buckeye Salt Company, Pomeroy, O.; Excelsior Salt Works, Pomeroy, O.; Syracuse Salt Works*, Syracuse, O.; Dow Chemical Company, Midland, Mich.; Saginaw Salt Company, St. Charles, Mich.; St. Louis Chemical Company*, St. Louis, Mich.; John A. Beck Salt Company, Allegheny, Pa.; J. Q. Dickinson & Co., Malden, W. Va.; Slagel Salt Company, Mason, W. Va., and Pomeroy, O.; Liverpool Salt and Coal Company*, Hartford, W. Va.; Hartford City Salt Company, Hartford, W. Va. The concerns marked with an asterisk were not in operation in 1907, but their plants are held in readiness to resume production.

Michigan (By Alfred C. Lane).—The continued low price of bromine took away all stimulus to further increase in production in Michigan, where the industry remains confined to the Dow Chemical Company, at Midland and Mount Pleasant, and to the production of small quantities at St. Charles, St. Louis, etc. New wells were put down at Midland during 1907. A brine about equally strong was struck by Solling, Hansen & Co., at Grayling, where it could be handled in connection with extensive saw-mills, but so far as known to me no one has been found to take up the business. The analysis of this brine is as follows: $CaCl_2$, 72.627 grams per kilogram; NaCl, 134.684; NH_4Cl, 3.07; $MgCl_2$, 20.128; KCl, 0.873; $MgBr_2$, 2.116; K_2SO_4, 0.248. Total, 233.746.

This is about as strong as the Midland water. Just about the same amount of bromine, i.e., 2000 mg. per liter, has been found by Fernekes in the deep water of the copper mines described by Fernekes and Lane. But those waters occur in very limited quantity and soon drain away. The ordinary Marshall sandstone brine along the Saginaw river will run from 0.35 to 0.97 grams of bromine per kilogram.

CADMIUM.

The manufacture of cadmium was begun, for the first time in the United States, in 1907 by the Grasselli Chemical Company at Cleveland, O. The process employed is fractional distillation in iron retorts. The production was small, the demand for the metal being small. It is used principally by the manufacturers of silverware; also there is a small consumption for the manufacture of cadmium yellow pigment; also for the manufacture of several alloys and an amalgam. Heretofore, the domestic consumption of cadmium has been supplied from Upper Silesia.

The manufacturers of silverware find that the addition of 0.5 per cent. of cadmium imparts malleability to the alloy and prevents to a certain extent the formation of blisters. Silver manufacturers now use cadmium in making sterling for rolling or for sand or plaster casting.

The Grasselli Chemical Company casts its cadmium in sticks 12 in. long and ¼ in. in diameter, weighing about ¼ ℔. each. The present price for 100-℔. lots is $1.25 per ℔. f.o.b. Cleveland. The production in 1907 was about 15,000 ℔.

Cadmium in noteworthy quantity has been detected in some of the fume from lead smelting furnaces collected by bag filtration in Colorado, and the utilization of this material as a source of the metal has been considered, but it was decided that the market is too small to bother about.

The Metallurgy and Uses of Cadmium.

By Paul Speier.

In late years many new uses for cadmium have been found, and interest in the metal has increased. Cadmium blende, or greenockite, is of rare occurrence and for the production of the metal is of no commercial importance. This mineral is pale yellow to brick red in color, and among other localities has been found at Przibram, in Bohemia, and also in Transylvania. Cadmium in small amount is almost always associated with blende and calamine.

Cadmium is commonly produced by fractional distillation. Hydrometallurgical and electrometallurgical processes have been proposed, but no extensive commercial application has been made of either. A valuable by-product has in recent years been obtained from the manufacture of lithophone. In the purification of the zinc sulphate solution by means of zinc, a metallic mud is obtained which contains from 30 to

70 per cent. cadmium. From this mud, which besides cadmium contains copper, zinc, lead, and arsenic, it is possible, observing certain precautions, to isolate crude cadmium; this crude metal serves as an anode for the electrolytic separation of commercial cadmium. This mode of extraction has for perhaps a year been adopted in Great Britain, but so far but little of the product has come into the trade.

During the last 10 years the bulk of the cadmium of commerce has been obtained by the old process of fractional distillation. The process depends upon the fact that metallic cadmium volatilizes at a far lower temperature than does metallic zinc and also upon the fact that cadmium oxide, by means of carbon and carbonic oxide, is reduced to metallic cadmium vapor at a lower temperature than zinc oxide. The starting material for the production of cadmium is the blue powder produced in the first two hours of the zinc distillation, which contains 5 to 8 per cent. cadmium. This portion of the blue powder is collected separately. At some smelting works the fumes escaping from the retorts are burned and collected in a flue, from the first portion of which a product enriched in cadmium is obtained. This also is used as a source of the metal.

The metal, which is malleable and ductile and has a lustrous tin white color, is cast into sticks usually from 20 to 24 cm. long and 6 to 8 mm. thick. It oxidizes in the air, and its luster becomes somewhat dull by exposure. The atomic weight is 111.6 and the specific gravity 8.6; the melting point is 320 deg. C.

Cadmium forms alloys with many other metals and possesses the property of reducing the melting point. The alloys most frequently made are those consisting of bismuth, lead and tin, which are distinguished by their great fusibility. A composition of 50 per cent. bismuth, 26.7 per cent. lead, 13.3 per cent. tin and 10 per cent. cadmium melts at 60 to 70 deg. C., and a composition of 50 per cent. bismuth, 25 per cent. lead and 12.5 per cent. each of tin and cadmium, melts at 60.5 deg. C. These alloys have a brilliant luster; they are malleable, but break when drawn out to a third or a half their original length. An alloy of 50 per cent. bismuth, 14 per cent. cadmium and 36 per cent. lead melts at 82 deg. C. If the lead content be increased and the tin decreased the alloy becomes softer. By the addition of 1 per cent. of mercury the melting point is still further lowered. A mixture of cadmium (26 per cent.) and mercury (74 per cent.) is employed by dentists for filling teeth. A completely saturated amalgam is obtained by mixing 21.7 per cent. cadmium and 78.3 per cent. mercury. An alloy consisting of 26.7 per cent. lead, 50 per cent. bismuth, 13.3 per cent. tin and 10 per cent. cadmium, becomes viscous at 60 deg. C. and fluid at 70 deg. A soft solder consisting of 37 per cent. lead and 63 per cent. tin melts at 136 deg. C. if 8 per cent. cadmium be added; if 25 per cent. cadmium be added the melting point is lowered to 132 deg. C.

The soft solder made from 25 per cent. cadmium, 25 per cent. lead and 50 per cent. tin is very tough and can be hammered and rolled; its melting point is 149 deg. C. The alloys of cadmium and tin are very ductile. For stereotype metal it is useful to alloy 50 per cent. lead, 27.5 per cent. tin and 22.5 per cent. cadmium; or 46.1 per cent. lead, 33.2 per cent. tin, and 20 per cent. cadmium. These alloys are readily fusible but are harder than the usual bismuth alloy. It has also been shown by trial for the above mentioned purpose that an alloy of 35.5 per cent. lead, 52.5 per cent. tin and 12 per cent. cadmium is satisfactory. A serviceable soft solder results from an alloy of 25 per cent. cadmium, 25 per cent. lead and 50 per cent. tin. While bismuth lessens the toughness and malleability of the alloy, cadmium promotes its fusibility. The alloy named after Lipowitz consists of three parts by weight of cadmium, four of tin, 15 of bismuth and eight of lead. This alloy is soft at 60 deg. C. and at 70 deg. is completely melted. It is silver white in color, in luster somewhat resembles polished silver, and is very malleable and ductile. In water heated to about 70 deg. C. tin, lead and britannia metal can be soldered with it. In soldering german silver, argentan, brass or zinc, a drop of hydrochloric acid should be added.

An alloy of 50 per cent. bismuth, 33.4 per cent. lead, 9.4 per cent. tin and 5.2 per cent. cadmium melts at 76.6 deg. C.; an alloy of 50 per cent. bismuth, 25 per cent. lead, 15 per cent. tin and 10 per cent. cadmium melts at 66 deg. C. The fusion points of the following alloys are about the same: (*a*) 16.66 per cent. cadmium, 33.34 per cent. tin, 50 per cent. bismuth; (*b*) 12.5 per cent. cadmium, 37.5 per cent. tin, 50 per cent. bismuth; (*c*) 25 per cent. cadmium, 25 per cent. tin, 50 per cent. bismuth.

The specific gravities and melting points of several alloys have been determined by von Hauer as shown in the accompanying table.

MELTING POINT OF ALLOYS.

Composition.	Specific Gravity.		Melting Point.
	Found.	Calculated.	
(*a*) in atomic proportions:			
Cd, Sn, Pb, Bi	9.765	9.624	68.5 deg. C.
Cd, Sn_2, Pb_2, Bi_2	9.784	9.698	68.5 " "
Cd_3, Sn_4, Pb_4, Bi_4	9.725	9.666	67.5 " "
Cd_4, Sn_5, Pb_5, Bi_5	9.685	9.652	67.5 " "
(*b*) in mixed proportions:			
7.1 Cd, 50 Bi, 42.9 Pb	10.529	10.330	88.0 " "
16.7 Cd, 33.3 Bi, 50 Pb	10.563	10.275	89.5 " "
15.4 Cd, 30.8 Bi, 53 Pb	10.732	10.341	95.0 " "

The alloys of cadmium with gold, silver and copper are also distinguished by their easy fusibility and are especially suitable for jewelry work. For the manufacture of novel ornaments and gold ware there are

employed the following alloys: (*a*) 75 gold, 16.6 silver, 8.4 cadmium; (*b*) 74.6 gold, 11.4 silver, 9.7 copper, 4.3 cadmium; (*c*) 75 gold, 12.6 silver, 12.5 cadmium. The last yields a green gold.

At normal prices cadmium is used extensively for cadmium plating. On tin the coating is very hard and takes a high polish. For this work a solution of cadmium carbonate is the most suitable. The cadmium carbonate is prepared by precipitating a solution of a cadmium salt with corbonate of soda.

An alloy consisting of 20 parts of zinc and 15 parts of cadmium is suitable for use as a soft solder for aluminum bronze. To clean this solder preparatory to using, it is only necessary to scrape off the surface. Röder makes an aluminum solder consisting of 50 per cent. cadmium, 20 per cent. zinc and 30 per cent. tin. An alloy which at high temperatures possesses a greater tensile strength and ductility than copper, brass or bronze contains, according to an American patent, 92 per cent. copper, 4.5 per cent. tin and 3.5 per cent. cadmium. By the use of a greater proportion of cadmium an increased ductility is secured.

Cadmium has been used for a number of years in the manufacture of cadmium sulphide, a compound varying in color from pale to brilliant yellow. It is incombustible, is not altered by air or light, and resists the action of sulphureted hydrogen, alkalis and weak acids. Its color does not appear to be altered by grinding in linseed oil, provided that the cadmium yellow is free from separated sulphur. Cadmium sulphide gives in combination with ultramarine a beautiful green color. A mixed color that finds employment as a paper pigment is obtained in the precipitate formed by mixing solutions of barium sulphide and cadmium sulphate. Cadmium sulphide is also employed in pyrotechny. The combination of 66.6 per cent. potassium nitrate, 17.5 per cent. sulphur, 3.3 per cent. carbon and 12.6 per cent. cadmium sulphide gives a beautiful white flame, edged with blue.

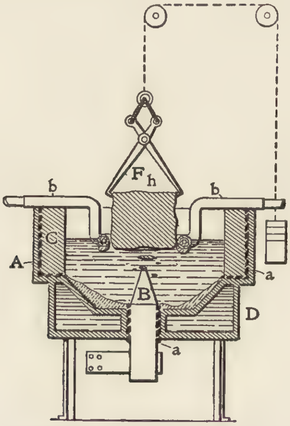
For commercial use cadmium should be at least 99.5 per cent. pure. Upper Silesia has remained supreme for many years in the production of cadmium. In 1882 the production amounted to 3521 kg., increasing in 1887 to 15,527 kg.; in 1897 the production was only 9840 kg., increasing to 27,561 kg. in 1906. For 1907 the production is estimated at 32,000 kg. In 1907 the United States became a producer, with an output of 7000 to 8000 kg.

The price of cadmium has been subject to extensive fluctuations. While in 1874 from \$3.55 to \$3.80 per kg. was paid, the price quoted in 1884 was \$2 to \$2.15 per kg. From 1888 to 1893 the quotation fell to 95c.@\$1.20 per kg. In April, 1896, the Prussian Pyrotechnic Laboratory and the Imperial Saxon Artillery Administration bought an important quantity of the metal and at the close of 1897 the price had risen to

$5 per kg. Soon afterward, with the above buyers out of the market, a quick downward movement again set in. The price up to the close of 1905 fluctuated between $1.31 and $1.55 per kg. At the beginning of 1906, under a very strong demand it advanced rapidly, the high point for the year being $3.57 per kg. The present quotation is $2.08 to $2.26 per kg.

CALCIUM.

This light, peculiar metal, which resembles silver in appearance, but is combustible like sodium, is now produced commercially by electrolysis of fused calcium salts. Experimentally the metal has also been prepared from calcium carbide. The high temperature of the electric furnace utilized for the synthesis of calcium carbide can also be employed to disassociate that compound, the calcium distilling off and being collected in a condenser. The electrolysis of a calcium salt is, however, the method commonly employed. Such a method is described in U. S. patent 880,760, granted to G. O. Seward and F. von Kügelgen, March 3, 1908, which may be assumed to represent practice at Holcombs Rock, Va., with which works the inventors are connected.



ELECTROLYTIC CALCIUM CELL.

Their patent specification shows a cell, illustrated in the accompanying engraving, which is intended for the electrolytic production of metals that are lighter than their molten electrolytes, and whose melting points are either higher or not much lower than the melting points of their electrolytes. A difficulty encountered in the production of such metals is their tendency to burn when they come in contact with air at the temperature at which the electrolysis is conducted. The cell is formed by a vessel *A* of cast iron and of circular form. The cathode *B* projects up centrally through the bottom, while a graphite ring *C* forms the anode. The electrodes are insulated from the cast-iron vessel by means of the

insulating compound *a*. The bottom of the vessel is protected by a chilled layer of the electrotyte; for this purpose the cold-water jacket *D* is provided. The calcium set free at the tapered cathode rises to the top of the bath and collects within the ring *E*, which is water-cooled, *bb* being the pipes through which the water is supplied. This ring confines the metal rising from the cathode and isolates it from the gases separated at the anode. The block of metal, *h*, which forms inside of the ring, is slowly raised out of the electrolyte by the tongs *F*.

Calcium is produced commercially in France, in Germany, and in the United States, in the last by the Virginia Electrolytic Company, at Holcombs Rock, Va. The Virginia Electrolytic Company also produces metallic sodium and magnesium by electrolytic methods, besides silicon and ferro-alloys. The metallic calcium is sold in the form of little sticks, 96 to 98 per cent. pure. Its average density is 1.72, i.e., it is nearly twice as light as aluminum.

Metallic calcium appears likely to become useful in the arts as its price becomes lower. More easily handled than sodium and less violent in its reaction, observes J. Escard in *L'Eclairage Electrique*, it seems particularly suitable for metallurgical processes which require the employment of a reducing agent to purify the molten metal at time of casting. Calcium decomposes slowly in dry air, and rapidly in air saturated with moisture, and burns with a brilliant white flame. Its hardness is greater than that of sodium, lead, tin, and somewhat less than that of zinc and magnesium. Its tensile strength is 0.61 kg. per square centimeter.

Its most interesting combinations are with hydrogen and nitrogen, forming the hydride and nitride of calcium. Calcium hydride, CH_2, was exhibited to the French Academy of Sciences in 1906 by G. Jaubert, under the name of "hydrolithe." Under the action of water at the ordinary temperature calcium hydride is decomposed with evolution of hydrogen. A kilogram of calcium hydride disengages 1143 liters of hydrogen at 20 deg. C. In the industrial manufacture of hydrolithe, metallic calcium is heated in horizontal retorts maintained at a high temperature, in which a current of gaseous hydrogen circulates; after some hours all the calcium is transformed into hydride. Hydrolithe appears in the form of irregular fragments, porous, white, or gray, of considerable hardness.

The avidity of metallic calcium for nitrogen will be utilized by metallurgy. We know what noxious influence the presence of nitrogen exerts on iron and steel. Hitherto, bismuth has been employed to purify molten pig iron, which also gives a nitrate; but elimination of the gas by bismuth is not perfectly satisfactory. On the contrary, it has been observed that calcium will meet all the requirements of the metallurgist. This subject, together with the properties of calcium, was treated in *Cosmos*, June 1, 1907.

CALCIUM CARBIDE AND ACETYLENE.

By A. C. Morrison.

The Union Carbide Company is now the only manufacturer of calcium carbide in the United States. During 1907 calcium carbide was manufactured to a limited extent by a small concern at Duluth, Minn., but recently it is reported to have discontinued operation. The Union Carbide Company is prosecuting suits for infringement of its patents against two of the users of the product manufactured by the concern above referred to.

The works of Union Carbide Company are situated at Niagara Falls, N. Y., and Sault Ste. Marie, Mich. The company has conducted an extensive propaganda for building up the calcium carbide and acetylene industry, and in assisting in placing on the market reliable and efficient acetylene apparatus.

The policy of the company has been to maintain non-fluctuating prices for calcium carbide, except for uniform reductions over large zones from time to time, to the end that consumers of the product shall have confidence in the stability of prices. Sales are made to a large extent directly to consumers, and large stocks of carbide are carried in warehouses at a large number of distributing points throughout the United States. Uniform prices for carbide are maintained throughout large zones, the difference in freight rates being absorbed by the company, in order to give the trade the benefit of uniform prices for the product over as large sections as possible. For instance, in the zone between the Atlantic Ocean and the Missouri river and from the Great Lakes to as far south as Charleston and Memphis, calcium carbide is sold at the many distributing points at $70 per ton, f.o.b. cars. Under these circumstances there is not the fluctuation in prices as in the case of most products.

The rapid strides with which the calcium carbide and acetylene industry has this year advanced in the United States has been paralleled by its progress in European countries. While in the United States there is but one company manufacturing calcium carbide, there are in Europe many establishments, though in no case do they parallel in magnitude the industrial organization built up in the United States.

Calcium carbide is still used chiefly as a source of acetlyene for artificial illuminiaton, although it promises to be employed extensively for the

manufacture of calcium cyanamide, a new fertilizer. It is a peculiar fact that acetylene in calcium carbide represents the most compact form in which an illuminant can be transported. The dream that liquefied acetylene would, if safe, conquer the world as a source of illumination is set aside by the fact that, considering the cylinder required, calcium carbide in 100 lb. drums will, weight for weight, produce as much acetylene as can be compressed into a similar package, even when compressed to a point of liquefaction. It is a further curious fact that the farther calcium carbide is carried in its relation to other illuminants, the cheaper it becomes relatively. Hence, it finds its way on mule back into the most inaccessible places.

When it is considered that the acetylene industry commercially has a history of only ten years, it is interesting to contemplate what the future has in store for it.

Uses Of Acetylene.

In 1907 the calcium carbide and acetylene industry experienced a continuation of the rapid advance which has been characteristic since 1900. The progress of the industry has perhaps been most distinctly marked by a change in public opinion respecting the safety of acetlyene. The Government ruling prohibiting the shipment of calcium carbide on passenger vessels, which went into effect in 1898, was removed, and the shipment of calcium carbide in metal drums is now permitted without any restriction. This was followed by the modification of the rules of the National Board of Fire Underwriters. The old ruling of the Underwriters (which, however, was not strictly enforced in some parts of the country) prohibited the installation of acetylene generating apparatus inside of insured premises. This was due to inexperience during the early history of the industry and some accidents which led to a prejudice against acetylene. Since then methods of use have been developed, which have caused acetylene to be declared safer than other illuminants. The Underwriters, after an exhaustive investigation, now permit the inside installation of generators which have passed satisfactorily the examining board of engineers.

Acetylene a Normal Illuminant.—During 1907 sharp attention was drawn to the character of the illumination produced by various means. This has brought to the front the well-known fact that the spectrum of acetylene is the nearest approach to that of daylight of any of the artificial illuminants. It is, therefore, the best possible light for the eye, and it is also apparent that acetylene has a special field where fine discrimination of colors is necessary, as in dye works, printing establishments, lithography, paint grinding, carpet mills, in testing syrups and molasses, and many other purposes.

Acetylene and Plant Life.—A long series of experiments has been conducted upon the growth of plants under artificial illuminants, and the result has shown that the only artificial illuminant under which plants will grow naturally to fruition is acetylene. The subtle differences in the spectrum of other illuminants are clearly detected by the elusive chemistry, one might almost say the alchemy, of plant life, and only under acetylene will vegetation grow without abnormalities.

Use in Photography.—It is now recognized that the actinic rays of acetylene give it a great utility in photography. The amount of acetylene used for such purposes it not yet great, but it is more economical than the illuminants now employed, and its use in this direction will doubtless increase. The time required for taking a negative and developing prints will be considerably less, and photographic effects will be possible with acetylene which cannot be obtained by other means.

Acetylene as a Standard.—Acetylene has been very prominently mentioned for use as a standard for illumination. While there are many standards in use, there is no entirely satisfactory unit of light. The basis of all illumination is the light emitted by a sperm candle. This has a number of disadvantages which have never been entirely overcome.

It seems rather audacious, perhaps, that a suggestion should be made that other standards of illumination for estimating candle powers be abandoned and that the adoption of acetylene as a standard should be advanced . The necessity is, however, well known. The use of acetylene as a standard by which other illuminants should be measured has been advocated by some of our greatest scientists.

Aids to Navigation.—The Canadian Government has begun the substitution of acetylene for lighting all its buoys, lighthouses and beacons. It possesses qualities of steadiness, the possibility of burning for months without attention in remote places, and it is said to have a penetrating power in fog that is unparalleled. With the light steadily burning, it is said that the heat of the generation of acetylene is sufficient to protect the generator against freezing, and acetylene in cylinders compressed in acetone and asbestos suffers no change under the lowest temperatures recorded. The United States Government has already adopted acetylene for many lighthouses and for many beacons, and is now considering its further use. Brazil is actively employing acetylene and rapidly installing buoys using acetylene wherever practical. The use of acetylene in all lighthouses where oil is now employed is one of the probabilities of the near future.

Headlights and Signal Lighting.—It has been found that acetylene headlights are economical and of great advantage for long distance trolley lines. Acetylene headlights have the advantage of burning steadily, whereas, whenever the electric current is cut off, the car is left in darkness

and may run for a long distance at great speed without illumination. The railways are experimenting with acetylene for signals. The penetrating quality of the light in fog is here a matter of importance.

Acetylene for Power Purposes.—The use of acetylene for power purposes has recently received a great stimulus by the enactments of the Government relating to denatured alcohol. It is found that denatured alcohol containing 20 per cent. of water, if atomized and blown through calcium carbide, will take up enough acetylene to enliven the alcohol and bring it up to the exact standard required for internal combustion engines.

Miners' Lamps.—During 1907 over one hundred thousand acetylene miners' lamps were sold and put into use in the mines of the United States. The result is a very large saving over the cost of any other illuminant, a more brilliant and penetrating light and total absence of smoke, all of which are very advantageous. Acetylene seems to have a great destiny in this field which will be of great advantage to the mining industry.

Autogenous Welding.—A combination of acetylene and oxygen under about one-half atmospheric pressure, used through a blow-pipe, produces a temperature approximating 6000 deg. F. The flame can be produced not larger than a pencil point, and through its means welding is successfully accomplished by literally melting the metal together. Great steel bridge girders can be melted apart with ease and facility. It is said that a 14-in. shaft has been welded. Where the metal is so welded, the file fails to disclose the joint.

Calcium Cyanamide.—The use of calcium carbide for the manufacture of cyanamide has been described in another article in this volume.

CALCIUM CHLORIDE.

Calcium chloride is obtained as a by-product in the manufacture of caustic soda by the electrolytic method; also as a by-product of the ammonia-soda process, and this is the chief source of supply in the United States. Moreover, a large quantity is obtained as a by-product from brine in the manufacture of common salt in the Ohio Valley. The output from the last source in 1907 was about 15,000 tons. The total production of calcium chloride in the United States in 1907 was about 45,000 tons, containing 40 per cent. $CaCl_2$.

The principal producers in the United States are the Solvay Process Company, of Syracuse, N. Y., and Rhodes & Co. and P. Van Schaack & Son, of Chicago, Ill., and the salt manfacturers of the Ohio Valley. The first named company is the largest producer; its output is sold through the Carbondale Chemical Company, with offices in New York, Chicago and Boston.

Prices.—At the end of 1907 the producers in the Ohio valley were realizing about $9, per ton f.o.b. cars. During 1907 the price of 40 per cent. calcium chloride, f.o.b. New York or Chicago, ranged from 78 to 98c. per 100 lb. for less than carloads; 63 to 78c. for carload lots. In April, 1908, carload lots were quoted at 68c., while smaller quantities brought 88c. per 100 lb. Although the 40 per cent. is the standard, there are on the market grades containing as high as 98 per cent. calcium chloride; these are sold at a higher price, depending on the percentage of calcium chloride present, the quantity and the form in which it is desired.

Grades and Uses.—Following is a description of the different grades of calcium chloride made by the Solvay Process Company:

Commercial fluid: contains 40 to 50 per cent. anhydrous calcium chloride, according to the specific gravity, and always less than 0.5 per cent. of impurities. During cold weather a solution stronger than 4 deg. B. is liable to crystallize somewhat, hence no shipments are made during the period from Nov. 1 to April 1. It is shipped in tank cars of 4500 gal. capacity, giving when diluted from 9000 to 12,000 gal. of solution sufficiently strong for most purposes, depending upon the strength of the original and finished solutions. It is also shipped in 110-gal. iron drums.

Kal crystal: contains 60 per cent. anhydrous calcium chloride, crystallized; practically chemically pure. Intended specially for automobile

service or other uses where a pure calcium product, easy to dissolve and absolutely neutral, is required. Put up in 10-lb. tin cans.

Commercial fused: contains not less than 75 per cent. anhydrous calcium chloride, not more than 25 per cent. water and less than 0.5 per cent. of impurities. This grade is used in ice making and refrigerating plants, canning and other establishments where absolute purity is not essential. It is shipped in 635-lb. iron drums painted with asphalt varnish.

Granulated: technically pure; contains not less than 75 per cent. anhydrous calcium chloride and not more than 25 per cent. water. This product is as dry as calcium chloride can be made without resorting to calcination which, for most purposes, would have no special value and adds considerably to the cost. It is applicable for drying air, vapors and gases, and is used in chemical operations because of the ease with which it can be dissolved and incorporated as an ingredient in compound products, either fluid or solid. Shipped in 350-lb. paper-lined barrels.

Calcined and powdered: contains 98 per cent. anhydrous calcium chloride; practically chemically pure; a desirable form for the use of cement manufacturers, color manufacturers and others who desire to mix calcium chloride intimately with other substances during the manufacturing processes without first dissolving it, or to incorporate it intimately just before shipment. It is carefully screened and can be furnished in various degrees of fineness. Shipped in 350-lb. paper-lined barrels or put up in small packages.

Granulated and calcined: contains 98 per cent. anhydrous calcium chloride; practically chemically pure. A desirable form for dying purposes in the chemical industries, being a powerful absorbent of moisture. Shipped in 350-lb. paper-lined barrels.

Although the chief use of calcium chloride is in artificial refrigeration, on account of its great heat absorbing power, it has of late years come into use for many different purposes. Among the most important uses may be mentioned the following: To form a non-freezing solution for use in water jackets of automobiles and gasoline engines and for preventing the freezing of chemical fire extinguishers; to improve the cementing power of lime and the setting of cement, and to prevent concrete from freezing; in the thawing of ground, as a dryer in cold storage warehouses and as a desiccator.

CALCIUM CYANAMIDE AND CALCIUM NITRATE.

As early as 1781 Cavendish attempted to cause the oxygen and nitrogen of the air to unite and form nitric oxide. This reaction, represented by the equation $N_2+O_2=2NO$, has been repeated many times since by other chemists who have thought of using the principle for the industrial production of nitrates. In 1898 Sir William Crookes predicted that the world would not continue to produce the breadstuffs it required unless some way were found to restore the nitrogen of the soil. Scientific research has lately demonstrated beyond a doubt that the fixation of atmospheric nitrogen can be economically accomplished wherever cheap electric power is available.

Two processes have been worked out and are now employed upon an industrial scale. In the first of these an electric arc is employed to produce nitric oxide gas by combustion of the nitrogen of the air, and this gas, after cooling, is passed through milk of lime to produce calcium nitrate. The Birkeland & Eyde is the most promising of the processes of this class and is now being operated upon a fairly large scale at Notodden in Norway. This process was described briefly in THE MINERAL INDUSTRY, Vol. XV.

The second type of process for the fixation of atmospheric nitrogen depends upon its absorption by carbides at a high temperature. The most successful process of this type is that of Frank & Caro and is based upon the use of calcium carbide as the absorbing medium.

CALCIUM CYANAMIDE.

The Società Generale per la Cianamide of Rome, Italy, holds all patents covering the manufacture and use of this product, and has had an experimental plant in operation for the last three years. Licenses for the manufacture of calcium cyanamide have been sold in all civilized countries, and many plants are projected in Europe. A recent list of those said to be in construction in Europe is given in the accompanying table. The tonnages indicated in the last column represent initial capacities; the ultimate capacities will be about twice the amounts shown. The plant at Piano d'Orta is now in operation producing at the rate of 4000 tons; the others were expected to begin about Jan. 1, 1908, or within a few months of that time.

CYANAMIDE WORKS IN COURSE OF CONSTRUCTION IN EUROPE.

Name of Company.	Location of Works.	Annual Capacity in Tons.
Northwestern Cyanamide Company of London	Odde, Norway	13,750
Ostdeutsche Kalkstickstoffwerke von Berlin	Muhlthal, Prussia	3,300
Société Suiss Probera Azotati	Martigny, Switzerland	4,400
Société Francais Pour Produits Azotati de Paris	Notre Dame de Brianon, Savoy, France	13,200
Cyanid Gesellschaft von Berlin	Trosberg, Bavaria	16,500
Società Italiana pel Carburo di Calcio di Roma	Callestatti, Italy	13,750
Società Italiana pel Carburo di Calcio di Roma	Sebenico, Dalmatia, Austria	27,500
Società Italiana pel Carburo di Calcio di Roma	Almissa, Dalmatia	50,000
San Marcel Compagana	Italy	8,250
Società Italiana per la Fabricazione di Produtti Azotati di Roma	Piano d'Orta, Italy	11,550
Società Italiana per la Fabricazione di Produtti Azotati di Roma	Fiume, Dalmatia	4,400

In the United States, the American Cyanamide Company, 100 Broadway, New York, has announced its intention to build a plant for the manufacture of calcium cyanamide at Mussell Shoals on the Tennessee river in northern Alabama. The initial capacity will be 40,000 tons per annum. Experimental work, preparatory to the building of the plant, is now being conducted at Nashville, Tenn.

The Frank & Caro Process.

The following is an abstract of an article, by John B. C. Kershaw, in *Electrician*, Jan. 24, 1908.

As now carried out at Piano d'Orta in Italy, this process consists in leading nitrogen into closed retorts containing powdered calcium carbide heated to a temperature of 1100 deg. C. The reaction between the carbide and the nitrogen is an exothermic one, and yields free carbon in the form of graphite, which remains distributed in the mass of material, and gives the product a black color.

$$CaC_2 + 2N = CaCN_2 + C.$$

At Piano d'Orta the carbide is obtained from a neighboring works, but Frank has patented a modification of the original process, which permits lime and coke to be employed as raw materials in place of calcium carbide. The chemical equation for the formation of calcium cyanamide then becomes

$$CaO + 2C + 2N = CaCN_2 + CO.$$

The furnace in which Frank proposed to carry out this reaction is shown diagrammatically in sectional elevation in Fig. 1 and in sectional plan in Fig. 2. It consisted of a brickwork chamber 6 m. in length by 3 m. in hight and 3 m. in width, containing an inner chamber constructed with a pigeon-holed wall, as shown most clearly in Fig. 2. This inner chamber was filled completely with the crushed and mixed raw materials, lime and

coke, in the required proportion, and electric heating was provided for by the carbon electrodes and core of finely crushed coke, running through the center of the mass. The nitrogen was passed under pressure into the annular space surrounding the pigeon-holed chamber of the furnace, and the carbon monoxide gas escaped by the vertical vent pipe from this inner chamber, as shown in Fig. 1.

Although this method of producing calcium cyanamide is practicable, it has not been adopted at Piano d'Orta, the carbide of calcium required for this works being produced very cheaply by a neighboring carbide works. The nitrogen required for the process can be obtained either by passing air over heated copper (contained in retorts as shown diagrammatically in Fig. 3), or by the fractional distillation of liquid air produced

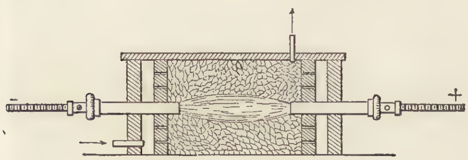


FIG. 1

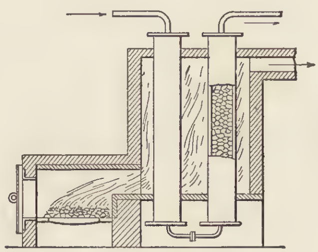


FIG. 3

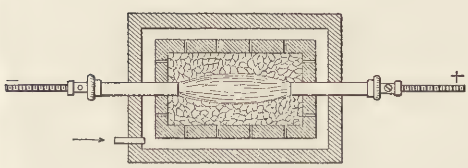


FIG. 2

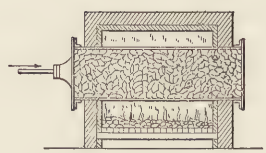


FIG. 4

FORMS OF CALCIUM CYANAMIDE FURNACES.

by the Linde process. The latter method has been found the most successful and has been adopted at Piano d'Orta.

The air is freed from moisture and is then compressed and liquefied at a temperature of —194 deg. C. This liquid air is subjected to repeated fractional distillations in apparatus similar to that used in the spirit industry, and is separated into its constituents, oxygen and nitrogen. The oxygen gas is compressed and sold in steel cylinders for various purposes. The furnace in which the absorption of nitrogen occurs is shown diagrammatically in Fig. 4. The carbide broken to a suitable size is packed into an iron retort provided at each end with removable covers for charging and discharging purposes. This retort is placed horizontally in a furnace, and is raised to the requisite temperature either by the combustion of the solid fuel or by electric heat.

Nitrogen gas is then passed under pressure into the closed retort and the absorption commences with a marked rise in the temperature. When no more nitrogen is absorbed, the contents of the retort are withdrawn and are allowed to cool in an inert atmosphere. The lumps of calcium cyanamide are finally crushed to a fine powder, which is packed in sacks for sale.

The exact details of the method of manufacture and of the plant employed are at present withheld from publication, and the above are only the broad lines of the process as carried out at the large works erected at Piano d'Orta in Italy and put into operation in December, 1905. It is probable that the retorts and furnaces are of more complicated design and structure than shown in Fig. 4 and that mechanical methods of charging and discharging the retorts are employed.

Proposed Modifications.—Polzeniusz has suggested the addition of calcium chloride ($CaCl_2$) to the carbide used in the retorts. A mixture of 23 per cent. calcium chloride with 77 per cent. carbide has been found to absorb the nitrogen at a much lower temperature (700 deg. to 750 deg. C.). The disadvantage of this addition is that the calcium chloride remains unchanged, and being hygroscopic, it confers the same character upon the finished product. It is therefore impossible to keep the cyanamide made with this addition dry when stored unless packed in air-tight drums or kegs. Carlson has however improved upon the suggestion of Polzeniusz by substituting calcium fluoride (fluorspar) for the chloride. This has the same effect in lowering the temperature at which the absorption of nitrogen occurs, while it is free from the hygroscopic defect of calcium chloride.

It is a curious fact that pure calcium carbide does not absorb nitrogen, and that the presence of some other compound of lime is necessary to start the absorption process. In commercial calcium carbide this other compound is supplied by the lime (CaO), which always exists as an impurity in the commercial product. Foerster and Bredig have sought to explain why these additions are necessary and assume that the chloride or fluoride acts as a diluent, enabling the nitrogen to penetrate more easily the whole of the mass of carbide.

It has also been suggested that producer-gas, which contains up to 50 per cent. nitrogen, would be cheaper than nitrogen obtained by the Linde process for manufacture of calcium cyanamide. The presence of 12 per cent. of carbon monoxide in this gas, however, renders it unsuitable for the purpose, and there is no convenient or economical process known which would permit of the complete removal of this constituent of producer-gas.

Composition and Cost.—Calcium cyanamide is a compound of the cyanide type, and its chemical formula shows that if it retained the atom of carbon within the molecule, which is set free as graphite during its

formation, it would be a true calcium cyanide, $Ca(CN)_2$. On fusing calcium cyanamide with sodium carbonate this change is in fact brought about. As placed upon the market calcium cyanamide is a fine black heavy powder, containing from 55 to 57 per cent. of the cyanamide and from 19 to 21 per cent. of nitrogen. It is about equal to sulphate of ammonia with respect to its nitrogen content.

As regards costs no very reliable figures based on actual work have yet been published. The cost of water-power is in fact the determining factor in the cost of the new product, and where this can be very cheaply developed, say for £2 per electrical horse-power-year, the prospects for the new industry appear to be promising. Dr. Erlwein has stated that one ton of nitrogen can be obtained in the form of calcium cyanamide by the Frank & Caro process for an expenditure of 2 kw.-years (equal to 330 kg. nitrogen per electrical horse-power-year), but whether this estimate covers all the power requirements of the works is not clear. On the same authority the cost of producing nitrogen from liquid air at Piano d'Orta is stated to lie between 3 pfg. and 5 pfg. per cubic meter when using electric power, costing 2 pfg. to 3 pfg. per kilowatt-hour.

At present the chief aim of the promoters of the calcium cyanamide industry is to obtain a trial upon a widely-extended scale for the new artificial manure, and to this end the selling price has been fixed so that it may be slightly cheaper than either sulphate of ammonia or nitrate of soda per unit of available nitrogen. No sales of the product are, however, being made in the United Kingdom, as there is no supply yet available. It is hoped that the Odde factory will be producing early in 1908, and the price of the product in the United Kingdom is to be 10s. per ton lower than ammonium sulphate of the same nitrogen content.

Calcium Nitrate.

The manufacture of calcium nitrate from atmospheric nitrogen on a commercial scale began in Nottoden, Norway, where 2000 h.p. produced about 1000 tons of nitrate per year. When the success of this small plant was demonstrated a larger works employing 40,000 h.p., supplied by the Tinfos waterfalls, was erected. This plant began operations on Sept. 1, 1907. The Norwegian Storthing some time ago authorized the construction of a dam at the Rjukanfos, one of the greatest waterfalls of the country, and the erection of a power plant to develop 250,000 h.p. for nitrate manufacture.

The production of nitric oxide requires the high temperature of the electric arc. The thermal action which produces the union of the nitrogen and oxygen by sparking is a reversible one, and unless the nitric oxide is speedily removed from the influence of the electric spark, it will be disassociated by the same heat energy which produced it. The process

of Bradley and Lovejoy, developed at Niagara Falls, depended upon the use of point discharges. The special feature of the Birkeland & Eyde process is the employment of a disk of flame. This process is more satisfactory and efficient than any other which has preceded it.

In connection with the Birkeland & Eyde process the following, condensed from an article by J. B. Van Brussel (*Engineering News*, Feb. 7, 1907, through *Technical Literature*, March, 1907), is of interest:

The disk of flame is produced as follows: The arc is struck between horizontal electrodes in series with an inductance coil, and the magnetic field is applied by a powerful electro-magnet, the poles of which are brought close to the arc, at right angles to the electrodes. The arc is rapidly driven outward, traveling back along the electrodes until it breaks, when a new arc is instantly formed between the tips of the electrodes and is driven outward in its turn, the successive arcs following each other so rapidly that thousands may be formed in a second. In practice, the frequency is a few hundreds per second.

When the arc and the magnet are both fed with direct current, the arc is always deflected to the same side, but when alternating current is used for one and direct current for the other, the deflections are alternately on opposite sides, and the rapid succession of arcs gives the appearance of a disk.

In the furnace or "oven" first employed, which has since been modified in the light of later experience, air is forced through two channels into a chamber, where it is brought into intimate contact with the electrodes before passing out by way of another channel, carrying with it a proportion of oxidized nitrogen. In the larger units first experimented with, from 75 to 200 kw. were used between one pair of electrodes, the arc being struck with alternating current at 500 volts pressure and 50 cycles per second frequency. The electrodes can be made of copper or iron, are easily cooled with air or water, and last several hundred hours.

The Notodden factory has been in constant operation about a year. Three furnaces of 700 h.p. each have been erected, with a combined capacity of about 2648 cu.ft. of air per minute. The furnaces work with a potential of 5000 volts, upward of 3500 volts being obtained across the electrodes. The cost of these furnaces, including inductive resistance, is $5000.

The gases come from the furnaces at a temperature of 600 deg. to 700 deg. C. and pass under a steam boiler, the steam of which is employed in the further manufacture of the ultimate product—calcium nitrate. On leaving the boiler, the temperature is 200 deg. C., and they are then reduced to about 50 deg. C. by means of a cooling apparatus, in order to be more easily absorbed by water.

The gas next enters two large oxidation chambers with acid-proof

linings, where the nitric oxide becomes nitric peroxide, and then flows into an absorption system, where the gas is converted into nitric acid. This consists of two series of stone towers having internal dimensions of $6\frac{1}{2}$x$6\frac{1}{2}$x$32\frac{1}{2}$ ft. Each series contains five towers, two of granite and two of sandstone, filled with pieces of quartz over which water and the nitric acid formed are made to trickle; the fifth tower in each series is filled with ordinary bricks, over which trickles milk of lime. The milk of lime quickly absorbs the rarefied nitrous gases remaining, and is converted into a compound of calcium nitrate and calcium nitrite.

The first tower yields nitric acid concentrated 50 per cent., the second about 25 per cent., the third 15 per cent., and the fourth 5 per cent. The liquids are raised from tower to tower by compressed air, thus gradually increasing their concentration up to 50 per cent., at which density the acid is conducted into a series of open granite tanks where it is temporarily stored.

Some of this acid is employed in the decomposition of the nitrate-nitrite combination obtained from the absorption by milk of lime. By the addition of nitric acid, the nitrous anhydride contained in the nitrite is driven out and carried back to the system of towers. The solution resulting from this, containing pure calcium nitrate, is carried, together with the rest of the stored-up acid, into another series of granite tanks, where this mixture of acid and calcium nitrate lye, reacting on ordinary limestone, is converted into a solution of neutral calcium nitrate. This neutral lye is carried farther into vaporization chambers of iron, where it is vaporized to a boiling point of 145 deg. C., which corresponds to a concentration of from 70 to 80 per cent. of calcium nitrate, containing about 13.5 per cent. of nitrogen. This substance is then run into iron drums, where it congeals, and in that form appears on the market.

It is said that half a ton of 100 per cent. nitric acid is produced per kw.-year, and that at Notodden this amount of electric power costs only $3.80. Each furnace therefore produces the equivalent of 250 tons of 100 per cent. nitric acid per year, equivalent to 325 tons of calcium nitrate, or to 337 tons of nitrate of soda per year. The present plant at Notodden is therefore capable of producing nitrates equivalent to 1000 tons of Chile saltpeter per year.

CARBONADO.[1]

.By J. P. W. Rowe.

Ever since the discovery of these precious stones in Brazil, this business has been a monopoly in the hands of native firms, who have associated themselves with French and German Jewish firms in Paris and London. Because of the fact that this is an article that can be easily smuggled out of the country without paying the export duty, which was 7 per cent. on the official value declared by the state customs house, nearly all the parties interested in this business evaded the duties, with the result that the governor of Bahia was forced to form a new law under date of Aug. 25, 1905, which reads: "Fifteen thousand milreis (three milreis=$1) for each firm, office, or merchant who deals for his own account or for account of a third party in diamonds or carbons in their rough state." Besides this State tax, interested parties must pay the municipal tax, which amounts to 7000 milreis ($2333). Complying with this regulation diamond merchants can ship their goods free of export duties.

It is generally calculated that the port of Bahia ships annually 12,000,000 to 14,000,000 milreis worth of diamonds, and the State should receive an annual revenue therefrom of 84,000 to 98,000 milreis, and the governor thought that by taxing the four or five diamond merchants with an annual tax of 22 contos each (1 conto=1,000,000 reis or 333\frac{1}{3}$) irrespective of the amounts they shipped, he would secure the yearly revenue of which the State was being deprived. When this law came into force the diamond merchants at once combined to form one firm, permitting it to be the only shipper, thus prorating the cost of the annual tax of 22 contos. Five firms joined the combination, and one remained out, refusing to comply with the new regulation. Although each firm works independently, they all combine as to the price limits they send to their buyers, who live in the interior diamond district. This district is about 267 miles from the city of Bahia, and can not be reached in less than four days.

The mining methods employed are of the crudest nature, and the work is all done by native miners. The diamonds and carbons are found in a gravel, known as *cascalho*, which occurs on the sides and slopes of the Sincora mountains and in the Paraguassu river bed and tributaries. Owners and lessees of diamond lots generally allow well-known miners to work their properties, who receive a return of one-fifth to one-fourth of the value of

[1] From *Daily Consular and Trade Reports*, Jan. 7, 1908. The author of this article, which was written about a year ago, was formerly American vice-consul at Bahia.

their finds. The former generally buy in all the precious stones themselves and sell them either on the spot to the diamond buyers who represent the Bahia firms, or the stones are shipped to Paris or London for their account, or on joint account with the firms established in Bahia, with their head offices at home. They receive from these firms from 80 to 90 per cent. of the value of the invoices, and the balance is paid after the goods are disposed of.

The buyers are always well supplied with cash from the firms in Bahia, and purchases effected by one firm alone often amount to over 300,000 milreis ($100,000) in one month. The following were the limits or prices given by the firms in Bahia to their buyers in the interior in the spring of 1906:

Carbon.—First quality, weighing from 6 graos to 120 graos, price 31,000 reis per grao; weighing from 120 graos and upward, price 30,000 reis per grao; second quality (porosos), 15,000 reis per grao; (crystallino), 10,000 reis per grao.

Ballas Broken Pieces.—Of 6 graos upward, 10,000 reis per grao; of 4 graos, 20,000 reis; of 3 graos, 12,000 reis; ballas fine for "fundos" (small and imperfect stones), price 2000 reis per grao.

Ballas, or Borts.—First quality, white, from 6 graos and upward, 30,000 reis per grao; colored green or blue, from 6 graos upward, 25,000 reis per grao.

Diamonds.—Bons, good, sound, well-formed stones, approximating 2 graos, viz.: 35 stones to the oitava, 17½ carats, 860,000 reis per oitava; stones of 6 graos, each 20,000 reis per grao to 8 graos; 4 graos, each 16,000 reis per grao; 10 graos, each 25,000 reis per grao to 16 graos; fazenda fina (small, fine stones of different colors), 5500 reis per grao, or 396,000 reis per oitava; vitrie or vidrilhos (small, bright, good-shaped stones), 7000 reis per grao.

Fundos.—(Small, badly colored and imperfect stones), 1500 reis per grao.

Native Weights.—One oitava is equal to 17½ carats. One oitava is equal to 72 graos.

The foregoing prices were those in the mining districts. The milreis is worth 33⅓ cents, while 1000 reis equal one milreis.

If any foreign firm or strangers to this district wish to secure a part of the diamond trade, they should send out an expert who can work in conjunction with a house which knows the diamond district and can introduce him to good connections in the interior. The present Government is anxious to bring new capital into the country, and provided those entering are willing to comply with the State laws, they will receive every assistance that may lie within the power of the authorities. If first-class machinery were erected and mining carried out in a proper and scientific manner there is a good return awaiting the proper development.

Good lots can still be acquired at a reasonable lease either from the Government or from private individuals who hold large tracts of diamond-bearing lands, and who are not interested in the combination. These parties often dispose of their goods at low figures to the buyers, as there is such a restricted outlet for the finds in Brazil.

Labor in the diamond district is rather expensive, a common laborer generally receiving from 2000 to 3000 reis per day, but provided mining is carried out in a proper manner good native laborers can be secured from the northern parts of the State at 1000 to 1500 reis per day, provided they receive constant work.

Diamond and carbon lots measuring from 484,000 to 29,040 sq.m. (1 sq. m.=10.764 sq.ft.) can be leased from the State government, by making a written petition to the director of the district at Lencoes, stating the lot desired. This petition is published in the district for 30 days, and it is then put up for auction and sold to the highest bidder. These leases generally run from 500,000 reis to about 10 contos per annum. The properties can be leased for a period of either one year or 10 years, an increase of 50 per cent. on the price being paid at every renewal.

All the most easily worked and the richest lots are already secured, but these can always be obtained by renting them from the present owners, who prefer to receive a fixed sum annually than to allow miners to work their claim at 20 to 25 per cent. of their finds. The district is generally not healthful on account of the constant overflow of the Paraguassu river, causing malarial fever of a very bad character in many districts.

All buyers in the interior are obliged to purchase entire parcels from the miners or dealers, as the latter will not allow the parcel to be divided by the selection of only the good stones and the rejection of the inferior qualities. Some months ago there returned from the diamond fields an expert sent from New York to secure first-class diamonds and carbons, and although he was successful in securing some beautiful stones he had to pay a high price for the selection, namely, 35 to 40 milreis per grao. A large carbon weighing 52 carats was found in Lencoes, and was held by the owner for about two years waiting a good price. This stone was sold early in 1906 for 90 contos of reis, exchange at 17 pence to the milreis, and was shipped to Paris.

[The carbon region begins about 267 miles from Bahia and can be reached in four days, viz.: Bahia to Cachoera, 45 miles by water, one day; Cachoera to Bandeira de Mello, 158 miles by rail, one day; thence to Andarahy, 64 miles, on mule back, two days.—EDITOR.]

CARBORUNDUM.

By F. J. Tone.

The Carborundum Company, of Niagara Falls, N. Y., which is the sole producer of carborundum in America, manufactured in 1907, 7,532,670 lb. This production comprises only crystalline carborundum and does not include amorphous carborundum which is used in large quantities for refractory purposes. This is the greatest annual production in the history of the company and the relative progress of the industry may be seen from the accompanying table.

PRODUCTION OF CARBORUNDUM IN THE UNITED STATES.

Year.	Pounds.	Metric Tons.	Value.	Year.	Pounds.	Metric Tons.	Value.
1891	50	......	....	1900	2,401,000	1,089	$168,070
1892	2,145	1	....	1901	3,838,175	1,742	268,672
1893	15,200	7	....	1902	3,741,500	1,698	261,905
1894	52,190	24	....	1903	4,760,000	2,160	333,200
1895	225,930	102	....	1904	7,060,380	3,203	494,227
1896	1,190,600	540	$365,612	1905	5,596,280	2,539	391,740
1897	1,242,929	564	153,812	1906	6,225,280	2,824	435,770
1898	1,594,152	724	151,444	1907	7,532,670	3,418	451,960
1899	1,741,245	791	156,712				

During 1907 the furnace plant of the company was remodelled and enlarged, its present capacity being 7000 electrical horse-power. his comprises both one-thousand and two-thousand horse-power units. Many improvements have been made in the methods of handling materials and in the discharging and recharging of furnaces but the main principles of the Acheson process remain unchanged. A new crushing and grading plant was completed and will soon be in operation. In 1906 the company erected at Reisholz, near Düsseldorf, Germany, a plant for the manufacture of grinding wheels and sharpening stones and these works are now in full operation, supplying carborundum products to Germany, France, Spain, Holland, Denmark, Scandinavia and Russia. The carborundum is shipped in graded form from the American works.

The manufacture of wheels and articles for abrasive purposes continues to constitute the main portion of the industry, and the infinite variety of grinding operations into which carborundum is entering is worthy of note. Besides all classes of metal grinding, it is used in the grinding of brick, terra-cotta, glass, slate, soapstone, porcelain, mosaic floors, pearl, rubber, paper, wood, shoes, hats, leather, hides, carbon, paper pulp,

granite, marble, petrified wood, precious stones and dental work. Some of these recent developments are worthy of special mention.

Marble Working.[1]—The application of carborundum to the marble industry, while of recent origin, is fast resulting in the entire displacement of old methods. The marble block as it comes from the quarry is sawed into slabs by means of gang saws. These saws which according to former practice were fed with a supply of sand and water, are now supplied with coarse carborundum grains, and the time of cutting a block has been greatly reduced. With carborundum grains it is possible to saw 18 in. of Italian marble per day and 1 ℔. will saw 2 sq.ft. of marble. The slab goes from the saw to the rubbing bed or rubbing machine. A recently patented form of drum rubber[2] equipped with a carborundum drum 6½ ft. long and 16 in. in diameter, will rub in eight hours 1000 sq.ft. of the most delicate and fragile marble, leaving a beautiful opalescent surface known as the spline finish, having the appearance of woven silk. The coloration and veining of the marble are fully exposed, making it an easy task to match the slabs perfectly.

The slabs are then placed on coping machines equipped with thin carborundum wheels by which they are cut into the desired sizes. The coping machine has been developed into several special types. The turning-head coper has a carborundum wheel mounted on a swivelling head, enabling the operator to cut the marble in two directions at right angles to each other without shifting the marble. By means of the gang coper comprising 13 carborundum coping wheels mounted on a single shaft, one slab may be cut into strips of various widths in a single operation. The cut-off coper is auxiliary to the gang coper. Special forms of carborundum wheels[3] have been designed for the work of coping.

For the cutting of molds a two-wheel molding machine has recently been developed in the operation of which a coarse carborundum wheel does the heavy cutting and is immediately followed by a fine wheel, which brings the mold to the exact shape and gives the required finish. The molding of marble by means of carborundum wheels is perhaps the greatest saver of labor which the industry has known. Molds are cut at a speed of from 5 to 20 lineal feet per minute, and it is possible to cut hundreds of feet of intricate molds with perfect uniformity without redressing the wheel. This work was previously done by a planer and laborious hand labor.

The counter-sinking or variety-molding machine uses a corrugated carborundum plug as the cutting element. Floor polishing machines for polishing mosaic or terrazzo floors have a capacity of 50 sq.ft. per hour. Rubbing and polishing machines for working marble are now

---

[1] *Eng. News*, Feb. 6, 1908, p. 143.
[2] J. R. Peirce, U. S. pat. 798,587, Aug. 29, 1905.
[3] J. R. Peirce, U. S. pat. 876,087 Jan. 7, 1908.

almost entirely equipped with revolving heads mounted with carborundum blocks. The form of head has been the subject of many recent patents.[1]

Paper Making.—The use of carborundum in the operations of beating and fibering paper stock has increased greatly during the past year and is viewed by many as the most important discovery that the trade has known in recent years. The preferred form of bed plate[2] for the beating engine consists of carborundum plates $1\frac{1}{4}$ in. thick, alternating with $\frac{1}{4}$ in. steel plates. These plates are assembled parallel with the axis of the beater, and the upper edges of the plates conform to the arc of a circle. As the stock is passed between the revolving beater and the carborundum bed plate, it is drawn out into long slender fibers, the action being similar to carding, and all bruising, breaking and crushing is obviated. Among the many advantages it may be mentioned that all color and dirt is quickly liberated, water penetrates the stock more readily and there is a saving of 25 per cent. in time and 50 per cent. in power. Bleaching is whiter and less bleach is required. The Jordan engine is now also equipped with carborundum bars coöperating with cone triplex, or quadruplex, steel bars, producing paper that felts better, closes up quicker and watermarks more clearly than in the regular Jordan. The pulp peels off the first press-roll more easily and a higher speed is possible. These results are especially noticeable on rag or manila stock.

The back fall and curb lining[3] of beating engines and rag washers are now constructed entirely of carborundum slabs, thus facilitating the work of the beater.

Refractory Uses.—Carborundum firesand has become the standard refractory of the brass foundry. The material is an amorphous form of carborundum, having great resistance to heat and flame action. It is mixed with silicate of soda and fire clay, and molded into the furnace shell to form a solid lining.

Carborundum enters largely into the concrete floors of fire-proof railway coaches, the cars of the new Hudson River tunnel equipment following this construction.

Metallurgical Uses.—Silicon carbide has long been known to be a powerful reducing agent. F. M. Becket[4] has devised a process for the reduction of the metals vanadium, molybdenum, tungsten, etc., by the use of silicon carbide as reducing agent. The process involves reduction in two stages, the first by carbon and the second by silicon carbide. A charge of vanadium oxide, for example, and carbon is first smelted in the electric furnace and the ore reduced to a lower oxide. Silicon carbide is

[1] N. C. Harrison, U. S. pat. 863,172, Aug. 13, 1907; U. S. pat. 730,527.
[2] S. R. Wagg, U. S. pat. 763,817.
[3] S. R. Wagg, U. S. pat. 871,540, Nov. 19, 1907.
[4] F. M. Becket, U. S. pat. 858,329, June 25, 1907.

added and complete reduction is then effected. Another process[1] refers to the production of ferro-alloys of the rare metals such as vanadium, and involves the direct reduction of the sulphide of the metal by silicon carbide, the resulting products being a ferro-alloy of the metal and sulphides of silicon and carbon. By this process vanadium alloys containing a very low percentage of carbon and merely a trace of sulphur may be produced from sulphide ores in a single operation by use of a relatively cheap reducing agent.

Carbide of silicon[2] is still used in large quantities as a silicon addition in the manufacture of steel, notwithstanding the growing use of 50 and 75 per cent. ferro-silicon.

Wireless Telegraphy.—As a responder in wireless telegraphic apparatus carborundum is the discovery of Gen. H. H. C. Dunwoody, formerly Chief Signal Officer, and it is now largely used by the United States Navy and the Signal Corps. The United Wireless Telegraph Company, controlling the DeForest system, uses the carborundum detector in more than 120 stations, and as a result of exhaustive experimentation it has found that with the exception of the electrolytic detector it exceeds in sensitiveness every known form of receiver. Carborundum detectors are used in approximately 130 out of 145 wireless telegraphy stations erected in the United States, or installed on vessels plying its waters.

The green variety of crystals is preferred and is found more sensitive than the blue or black crystals. Well defined crystals are required and fragments of close, dense masses are not sensitive. Silicon is also a very sensitive detector but unlike carborundum it is difficult to hold in place after the party sought is found, because when pressure is applied its sensitiveness is lost.

Hardness.—According to Dr. George F. Kunz,[3] no precise value in the scale of hardness has yet been determined for carborundum. All that can be said is that it is certainly near, but below, 10. Although exceedingly hard and close to the diamond, it is not equal to it as at first claimed. Beyond the fact that it will scratch strongly the surface of a ruby or sapphire and not that of a diamond, its hardness is not easy to determine for lack of other bodies with which to compare it. The differences between the units of the Mohs scale of hardness are found to increase greatly in ascending, so that the interval between 9 and 10 is regarded as probably equal to the entire difference from 1 to 9. This general fact was alluded to several years ago by Prof. T. A. Jaggar, of the Mass. Inst. of Technology, and it has been confirmed more in detail by later experiments conducted at Johns Hopkins University. The hardness of carborundum may, therefore, be far greater than that of sapphire or corundum, while much below

[1] F. M. Becket, U. S. pat. 876,313, Jan. 14, 1908.
[2] *Stahl und Eisen*, 1908, p. 255.
[3] *Trans. Am. Electrochem. Soc.*, 1907, XII, 46.

that of diamond. These observations agree with the determinations on the hardness of carborundum and various samples of corundum, made by Dr. Joseph Hyde Pratt, described in THE MINERAL INDUSTRY, Vol. XV, p. 98.

Index of Refraction.—Lewis E. Jewell has determined the curve of the index of refraction for carborundum and has compared the same with that of the diamond and also with that of flint and crown glass. The results given in Fig. 1 are of great scientific interest and show that the refraction of carborundum is the greatest of any known substance which is transparent to the whole visible spectrum. The values of the double refraction are more remarkable. A comparison between lines in the spectra

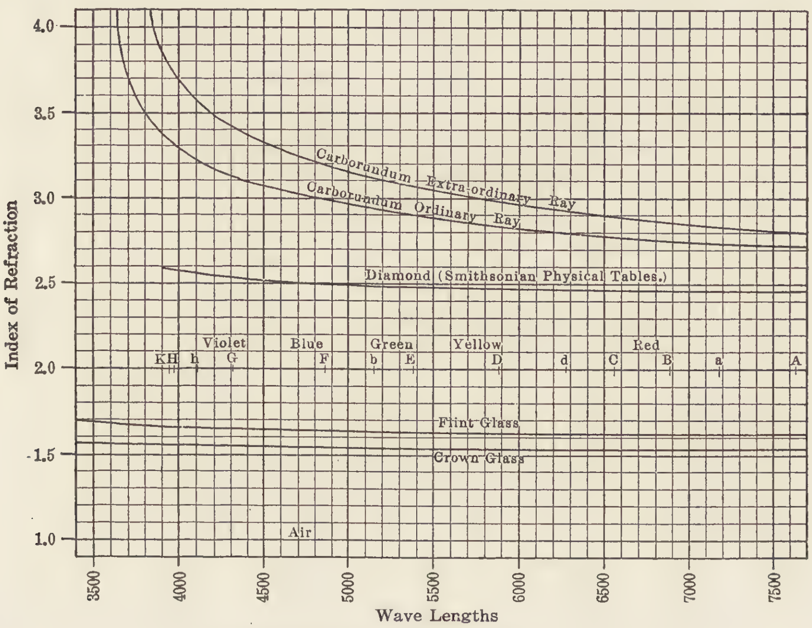


FIG. 1. INDEX OF REFRACTION OF CARBORUNDUM. LEWIS E. JEWELL.

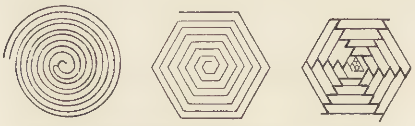
of the ordinary and extraordinary rays shows that the latter is truly an extraordinary ray having refractive indices of 2.8 at the end of the red, and 3.80 for the K line at the end of the violet. The transparency of carborundum is considerably greater perpendicular to the optic axis than parallel with it.

The production of clear transparent crystals of carborundum of sufficient size for optical purposes has been the subject of considerable research, and the efforts in this direction have been more or less successful. The material will prove useful in combination with cleavages of diamond for microscopic objectives of high power and good color values. Small

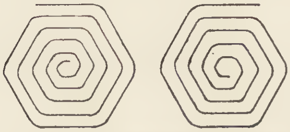
prisms of carborundum combined with prisms of diamond may be used in small direct-vision prism ocular spectroscopes. The combined prism which would be thin and have nearly parallel sides would possess great dispersive power.

Among the beautiful and interesting surface workings on carborundum crystals, remarkable spiral workings appear as shown in Fig. 2. Some spirals have been observed having 30,000 to 50,000 lines to the inch. Some of the spirals and some of the more or less straight lines form small gratings which give spectra of the fifth order or higher.

The remarkable refractive properties of carborundum and also its peculiar property of double refraction suggest its use as a gem. If crystals of sufficient thickness can be produced and cut in a manner to use the extraordinary ray, the fire or separation of light into color should be five times that of the diamond. Some crystals will produce gems of a color



Characteristic Spirals on Carborundum Crystals.



Twin Spiral on Carborundum Crystals.

FIG. 2.

more or less resembling sapphire but more brilliant, and others are of a remarkable beautiful sea green or bluish green color. These gems would exceed all natural gems in beauty and brilliancy.

Heat of Combustion.—The heat of combustion of silicon and silicon carbide have been recently determined by two investigators, who have published results on thermal changes involved in the formation of silicon dioxide. W. G. Mixter[1] obtained by careful calorimetric bomb determinations a heat value of 191,000 calories per 28.4 grams of crystalline silicon burned to silicon dioxide, which corresponds to 6720 calories per gram of silicon. Dr. H. N. Potter[2] obtained a value of 7595 calories per gram of crystalline silicon also by means of a bomb calorimeter. The value by Mixter was obtained by means of a sodium peroxide oxidation method, while that of Dr. Potter was obtained by means of compressed oxygen in the calorimeter.

The heat of combustion of silicon carbide when burned to silicon dioxide

[1] *Am. Journ. of Science*, XXIV, Aug., 1907.
[2] *Trans. Am Electrochem. Soc.*, 1907, XI, p. 263; ibid., 1907. XII, p. 202.

and carbon dioxide was found by W. G. Mixter to be 283,800 calories per 40.4 grams of carbide. This corresponds to 7025 calories per gram of silicon carbide. From the above values for silicon carbide and silicon dioxide and from known values of carbon dioxide, Mr. Mixter calculates an almost negligible value for the heat of combination of carbon and silicon from the elements. The value obtained was 2000 calories per 40.4 grams or about 50 calories per gram of silicon carbide.

Specific Heat.—The specific heat of crystalline carborundum has been determined by Dr. H. N. Potter as 0.1857, and that of siloxicon as 0.1911.

Carborundum when very pure is colorless and semi-transparent. W. R. Mott[1] states that the bright colors observed on silicon carbide are not due to iron or other impurities, but rather to a very thin (0.00001 to 0.0001 cm. thick) colorless layer of silica which produces the same play of colors as is observed on an oil film on water. This conclusion is verified by treating silicon carbide crystals with hydrofluoric acid, which dissolves the iridescent layer of silica, leaving the silicon carbide crystals a dull uniform gray.

Moissanite.[2]—The iron meteorites from the Canon Diablo, Arizona, have long been known to contain microscopic diamonds. These meteorites are found strewn over an area of several miles in hundreds of pieces weighing from a fraction of an ounce to a mass of 1680 ℔. In cutting a section from one of these masses certain very hard portions were encountered which interfered with the saw and broke it, leaving lines upon the steel. These portion proved to be cavities containing extremely hard particles which, upon examination by Dr. George A. Koenig, of the University of Pennsylvania, were identified as diamond carbon. A comparatively recent investigation by the late Prof. Henri Moissan has disclosed the fact that there are also present, associated with the diamonds, minute crystals of silicon carbide. He found that carbon was present in and around the nodules of troilite, and in certain minute fissures that seemed to connect them. In close association with them he found silicon carbide and apparently minute amounts of iron phosphide. The nodules, consisting mainly of troilite (iron protosulphide), were treated with hydrochloric acid, releasing much hydrogen sulphide, and dissolving a considerable amount of iron, some nickel, and traces of cobalt, phosphorus, sulphur, carbon and magnesium. The iron and phosphorus probably existed in combination as a phosphide of iron, apparently Fe_3P_2. The insoluble residue contained silica, amorphous carbon, graphite and diamonds, both black and transparent, the latter as minute octahedrons with rounded edges. The black diamonds were

---

[1] *Trans. Am. Electrochem. Soc.*, XII, p. 241.
[2] G. F. Kunz, *Trans. Am. Electrochem. Soc.* XII, p. 39.

abundant but very small. The most novel result of this examination however was the discovery of carbon silicide (carborundum) existing as a natural mineral in small green hexagonal crystals of characteristic type. These were found with the carbon, etc., in the insoluble residue of the nodules. These observations led him to the view that the presence of sulphur, phosphorus or silicon in a molten iron may tend to favor the separation of carbon therefrom; while the nickel present in meteoric iron lessens the solubility of carbon in the alloy thus formed. These two phenomena have combined to separate carbon from what must have been a liquid condition when silicon carbide was formed. Dr. George F. Kunz has given to silicon carbide, when thus occurring as a true natural mineral, the name of moissanite.

Recent Patents.—F. G. Smith (U. S. pat. No. 860,701) proposes a new form of zinc retort in which the main body of the retort is made of refractory fire clay and the lining is composed of carborundum mixed with chromic oxide. A refractory body is thus formed which resists the action of corrosive slags and is impermeable to zinc vapor. Carborundum is very extensively used on the Continent for the lining of zinc retorts.

George Egly (U. S. pat. No. 866,444) discloses a process for making rods, tubes, discs, etc., of high electrical conductivity and great mechanical, thermal and chemical resistance by mixing silicon with silicon carbide, moulding the same to the desired form, and heating in an atmosphere of nitrogen. The resulting body is stated to be silicon carbide cemented together by silicon nitride or silico-carbon nitride, and to be especially adapted for electric heating bodies.

There is a considerable demand in electrical work for a resistance element which is capable of dissipating large amounts of energy in a small space without oxidation in the air, and this is the object sought in the many resistance elements containing carborundum, which have been patented in recent years.

H. N. Potter (U. S. pat. No. 875,272) uses carborundum as a reducing agent mixed with silica for the production of the new product, silicon monoxide. This product is used as a pigment in printers' ink and in paint. It has high heat insulating properties, is adapted for the filtration of gases and has certain applications in ceramics.

R. Bouvier (French pat. No. 375,338) uses carborundum in place of diamonds in the manufacture of drill heads. The carborundum is incorporated in a suitable matrix, either ceramic or metallic.

G. Michaud (French pat. No. 378,665, Sept. 25, 1907) describes furnace apparatus composed of carborundum for electrically heating lamp filaments.

H. N. Potter (U. S. pat. No. 875,673) covers the method of producing silicon carbide consisting in heating together silicon monoxide and carbon.

CEMENT.

By Robert W. Lesley.

A review of the cement industry for 1907 shows that for the first time in a decade it has reached the normal condition characteristic of the other great industries of the United States. That is to say, there has been arrested the recent tendency toward speculative development which naturally followed the tremendous demand for cement. The statistics show that the portland cement industry has now reached that stage of development where its progress and capitalization must be attended by the same care and intelligence as are applied to kindred industries.

The manufacture of portland cement in the United States has developed by leaps and bounds. In the 10 years ending with 1907, the statistics show that the production has increased from 2,430,903 bbl. in 1897 to 48,785,390 bbl. in 1907. This growth, as a reference to the accompanying tables will show, has been at a rate of percentage per annum—even in latter years when the production has attained enormous figures—almost staggering, even to those who have grown up with the industry. From 1897 to 1899 the production almost doubled. The production of 1901 was more than twice the production of 1899. The production of 1903 was almost double the production of 1901; the production of 1905 was half as much again as that of 1903, and the production of 1906 was double that of 1903.

PRODUCTION OF CEMENT IN THE UNITED STATES. (a)
(In barrels.)

Year	Portland.			Natural Hydraulic.			Slag Cement.			Total.	
	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.
1897	2,430,903	$3,724,905	$1.54	7,890,573	$3,976,050	$0.50	40,000	$60,000	$1.50	10,361,476	$7,760,955
1898	3,584,586	6,168,106	1.72	8,168,106	3,819,995	0.47	157,662	235,721	1.50	11,903,326	10.223,822
1899	5,805,620	10,441,431	1.80	9,686,447	5,058,500	0.52	244,757	360,800	1.47	15,736,824	15,786,789
1900	8,482,020	9,280,525	1.09	8,383,519	3,728,848	0.45	446,609	567,193	1.27	17,312,148	13,576,566
1901	12,711,225	12,532,360	0.98	7,084,823	3,056,278	0.43	272,689	198,151	0.73	20,068,737	15,860,731
1902	17,230,644	20,864,078	1.21	8,044,305	4,076,630	0.50	478,555	425,672	0.81	25,753,504	25,366,380
1903	22,342,973	27,713,319	1.19	7,030,271	3,675,520	0.50	525,896	542,502	1.03	29,899,140	31,931,341
1904	26,505,881	23,355,119	0.90	4,866,331	2,450,150	0.50	303,045	226,651	0.75	31,675,257	26,031,920
1905	35,246,812	33,245,867	0.94	4,473,049	2,413,052	0.54	382,447	272,614	0.71	40,102,308	35,931,533
1906	46,610,822	51,240,652	1.10	3,935,151	2,362,140	0.60	481,224	412,921	0.86	51,027,321	54,015,713
1907	48,785,390	53,992,551	1.10	2,887,700	1,467,302	0.51	557,252	443,998	0.79	52,230,342	55,903,851

(a) Statistics of production for 1900 and subsequent years are as reported by the U. S. Geological Survey. The barrel of portland cement contains 380 lb. of the material; of natural cement, 265 lb.; of slag cement, 330 lb.

STATISTICS OF CEMENT IN THE UNITED STATES.
(In barrels.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.	Barrels.	Value.
1897..	10,361,476	$7,760,955	2,200,871	$2,688,122	62,761	$103,389	12,499,586	$10,345,688
1898..	11,903,326	10,223,822	2,119,880	2,624,228	55,969	98,121	13,967,237	12,749,929
1899..	15,736,824	15,860,731	2,219,246	2,858,286	116,079	213,457	17,839,991	18,505,560
1900..	17,312,148	13,576,566	2,512,300	3,330,445	147,305	289,186	19,677,143	16,617,825
1901..	20,068,737	15,786,789	994,624	1,305,692	303,380	752,057	20,759,981	16,340,424
1902..	25,735,504	25,366,380	2,100,513	2,582,281	367,521	575,268	27,486,496	27,373,393
1903..	29,899,140	31,931,341	2,439,948	3,027,111	312,163	466,140	32,026,925	34,492,312
1904..	31,675,257	26,031,020	1,101,361	1,383,044	816,640	1,158,572	31,959,978	26,256,392
1905..	40,102,308	35,931,533	891,134	1,102,041	1,060,054	1,428,489	39,933,308	35,605,085
1906..	51,027,321	54,015,713	2,321,803	2,950,268	600,386	964,373	52,748,738	56,001,608
1907..	52,230,342	55,903,851	2,006,228	2,637,424	900,550	1,450,841	53,336,020	57,090,434

Applying these figures to the growth of population in the decennial period of 1890 to 1900, the growth of the per capita consumption of the inhabitants can be readily seen to have been far greater than the growth of any other kindred industry. It is these remarkable figures, almost representing a doubling of the industry every year, that has led to the opinion that the production of portland cement was to be treated upon the basis of a mining promotion. This period of mining promotions in the portland cement business is shown by the figures of 1907 to have ceased. While in 1906 there were produced 46,610,822 bbl., in 1907 there were produced only 48,785,390 bbl. Referring to the figures for 1905, it will be found that while between 1905 and 1906 the growth was approximately 11,500,000 bbl., the growth between 1906 and 1907 was but 2,000,000 bbl., a percentage so small in this rapidly growing field as to be almost imperceptible and indicating, as above stated, that the business has, in a measure, reached a period of ordinary industrial development where the promoter has no further opportunity to advance his predictions upon the abnormal growth of the business and where the future development is to be based upon a normal increase of the plants now in business and upon enterprises conservatively projected, properly capitalized and carefully managed. For the first time the effect of the great multiplication of plants has been made apparent, so far as their relation to the limited demand is concerned, and for the first time the industry can see over-production. The mining promotion method as applied to the cement industry has a very different effect than when applied in its own proper field. A mine promotion injures no one but the investors should the mine prove unproductive. A modern cement mill involves the expenditure of a million or two millions of dollars, and if properly located with relation to its raw materials or to its distributing area, has an opportunity to exist, but if improperly financed and improperly managed and situated, remains after it has passed through a receivership a threat to the industry.

In 1905, for the first time, exports exceeded imports, and in 1906 the imports were nearly four times the American exports. The year 1907 shows the exports to be about one-half of the imports, and the figures for the first few months of 1908 show a tendency for the exports again to overtake the imports. These conditions are largely produced by two sets of facts. Imports to Southern, Gulf and Pacific Coast ports are largely brought over as ballast from Europe, and in years of large grain and cotton exports from this country, cement imports as a rule increase. Exports from this country fall off in prosperous times and increase when business in the United States is decreasing. The panic of 1907 had its effect upon the export business, and it is a source of regret to those in the portland cement industry that with the demand that can be built up abroad for cement of the quality which is produced in this country, the curve of exports shows such a variation (dependent upon good or bad times in this country), when this curve could be made a constant one of continuing growth by attention to the export business and the supplying of the established trade abroad, whether times are good or bad in this country. The development of the foreign trade in portland cement is likely to be one of the features of the future, especially in view of the over-production caused by the over-building of cement plants and the necessity of continuing markets to provide an outlet for the extra supply.

While the early months of 1907 showed prices fairly comparable with those of 1906, all over the country, the prices in the East fell materially early in the fall. This condition also occurred some months later in the West. The reduction was about 20 per cent. and even with this concession in prices the demand fell off largely as compared with the demand in the corresponding periods of 1905 and 1906. Natural cement fell off about 25 per cent., the production showing a constantly decreasing curve from 1902, when the production was about 8,000,000 bbl., down to 1907, when the production was 2,887,000 bbl. Slag cement, with an output of 557,252 bbl. in 1907, more than regained its standing of 1903, when the production was 525,896 bbl., the low output of 1904 with its 305,000 bbl., being almost doubled by the production of 1907.

Industrial Conditions.

For many years litigation has been going on in the cement industry over patents issued to Messrs. Hurry and Seaman, engineers of the Atlas Portland Cement Company, for the successful use of apparatus for burning pulverized coal in rotary kilns. Testimony was taken in all parts of the United States and Europe, and the most noted experts and manufacturers were called as witnesses. Six years were spent in preparation of the suit and when the case was argued officers and attorneys of the leading companies crowded the court room at Scranton, Penn., where, in 1906,

with the Alpha Portland Cement Company as defendant, and the Atlas Portland Cement Company as plaintiff, the case was brought to a final hearing. Before judgment was entered, however, five of the leading companies of the Lehigh region, including the defendant, came to terms with the Atlas Portland Cement Company and agreed to take out licenses recognizing the Hurry and Seaman patents, and to pay a substantial royalty. The North American Portland Cement Company was organized in the latter part of 1906, with a capital stock of $10,000,000, having among its purposes the acquiring from the Atlas Portland Cement Company of the Hurry and Seaman and other patents, and the licensing thereunder of portland cement manufacturers. During 1907 the North American company was extremely active in prosecuting infringers of its patents and in acquiring other patents for the protection of its licensees.

Another marked advance in the cement industry was due to Thomas A. Edison, who devised new kilns, together with several unique methods of fuel consumption. In particular, he designed and patented a rotary kiln 150 ft. long, and 7 to 8 ft. in diameter, having a daily capacity of from 700 to 1000 bbl. of cement. Until that time the largest kilns in use were 60 to 80 ft. long, and 5 and 6 ft. in diameter, with a capacity of but 200 bbl. per day. The adoption of the long kiln by the portland cement industry in general, and the consequent infringement of the exclusive patents held by Edison covering it, proved to be a possible fertile source of litigation which was only recently terminated by the acquisition by the North American company of the patents for long kilns, burners, and similar apparatus owned by Mr. Edison.

As a result of the threatened litigation in the trade by reason of the patents above referred to, and others under similar control, the Association of Licensed Cement Manufacturers was organized in New York on Jan. 9, 1908, by the North American Portland Cement Company, the Atlas, the Alpha, American, Lehigh, Lawrence and Vulcanite portland cement companies, and various other important companies in the East and West, including among others, the Dexter, Edison, Nazareth, Pennsylvania, Penn-Allen, and Catskill, all of which have secured licenses under the Hurry and Seaman, Edison, Carpenter and other patents controlled by the North American company. The purposes of the association include the general betterment of the mechanical and chemical processes used in making cement, the improvement of the quality of cement, dealing with matters of traffic and shipment and the establishment of an association laboratory for technical tests and experiments. It is understood that all existing and properly equipped cement plants will be granted licenses and admitted to membership, and that infringers of the patents above referred to will be rigorously prosecuted. Nearly 70 per cent. of the output of the portland cement industry in this country is already represented by the

association, this being double the annual production in Great Britain, the pioneer portland cement manufacturing country, equal to the combined output of England and France, and in excess of that of Germany.

New Plants.—Outside of Kansas, where plants have multiplied very rapidly in the gas belt, and California, where the earthquake and the rebuilding of San Francisco seem to have opened up new markets, 1907 shows but little addition to the portland cement capacity of the country. In Iowa one new works was completed and the new works started at Dixon, Illinois, by the Sandusky Cement Company, of Ohio, was finished. The Devilslide works in Utah was brought into operation, and the Dixie works, constructed by the Iola Portland Cement Company, near Chattanooga, also began to produce cement, as did the new works at Dallas, Texas.

Materials.—No new materials seem to have been introduced in the manufacture of portland cement during 1907, though in some quarters a tendency to use rocks high in magnesia has been indicated. A change in the specification of the American Society of Civil Engineers at its January, 1908, meeting, to correspond with that of the American Society for Testing Materials, permits of the use of a larger percentage of magnesia in the finished product.

Machinery.—As to machinery, the development of the Fuller mill from coal grinding alone, to both raw materials, and in some cases clinker, has been marked. The three-ball Griffin mill has also made considerable progress. All new mills practically are being constructed with the long kilns, and a tendency to larger diameters is also noted.

Engineering Investigations.—The standard form of specification has continued in general use and has passed the scrutiny of the committees on cement of the American Society of Civil Engineers and the American Society for Testing Materials, without material alteration. This specification, of which thousands of copies have been distributed all over the United States and in many foreign countries, has had much to do with the standardization of portland cement in the United States and much to do with the uniform quality of cement wherever produced. General recognition of the specification has been productive of the highest results both in the engineering profession and on the manufacturing side of the cement industry.

The work of the National Advisory Board on Fuels and Structural Materials has been carried on at the Government laboratory at St. Louis, under the direction of the expert in charge, Richard L. Humphrey, and a number of bulletins have been issued of the greatest value, all having to do with cement or concrete. Tables of the results of the various tests of cements, of the qualities of sands and gravels, and of the strength of beams, girders, etc., have been made during the past year, and a record of some 25,000 experiments has been published under Government

auspices. It is a source of congratulation that this work, which is done at the Government's expense, under the general supervision of the National Advisory Board on Fuels and Structural Materials, upon lines suggested by the Committee on Tests of the Joint Committee on Concrete and Reinforced Concrete, has been so successfully maintained, and that Congress has recognized the value of the work by again appropriating the sum of $100,000 for its continuance in 1908.

The Committee on Concrete and Reinforced Concrete, which is composed of members representing the American Society of Civil Engineers, the American Society for Testing Materials, the Association of American Portland Cement Manufacturers, the American Railway Engineering and Maintenance of Way Association and the American Institute of Architects, held numerous meetings during 1907, and it is expected that as a result of the work so far done at the Government Laboratory and in the college affiliated with the Committee, a preliminary report as to the preparation of standard specifications for concrete and reinforced concrete will be presented during the course of 1908.

Market Conditions.

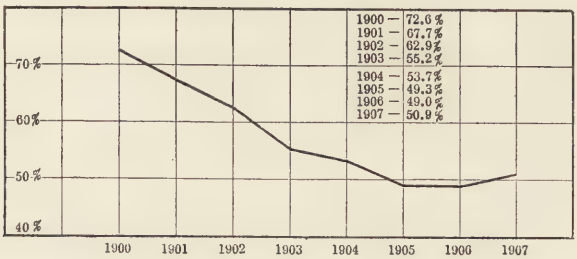
At the end of 1906, manufacturers looked forward to a prosperous year in 1907. The early months opened with a constantly increasing demand and contracts for future shipments were larger than at probably any other time in the industry. The winter of 1906-07 was a severe one and the enormous shipments of grain and agricultural products of the West caused a general shortage of cars, the result being that the early Spring months of 1907 showed, in many parts of the country, decreased shipments of cement as compared with 1906. Later on, with better weather conditions and more cars, shipments became more active and for a time the indications were for a large domestic trade, while the anticipated shortage of cement led to considerable importations of foreign cement on a speculative basis. As the season wore on there came the usual July lull and business fell off. With the usual revival of business in September and October (the big months in the industry), the money shortage came on and subsequently the panic, and mills all over the country found themselves confronted with lack of demand and a cancellation of orders owing to the cessation of new railroad construction and the abandonment of manufacturing enterprises projected all over the country. As a result, 1907 closed with large stocks of cement in the hands of the manufacturers, reduced prices, and a recognized over-production.

According to Edwin C. Eckel, of the U. S. Geological Survey, the existing American plants have now a total capacity of about 60,000,000 bbl. a year, and it seems unlikely that much more than two-thirds of this total capacity can be operated profitably in 1908. The only advantage

of this condition is that it will act as a check upon fraudulent and foolish promotion of cement projects.

An interesting development in the cement market was the request for bids for the total amount of cement required for the Panama Canal. This mythical amount was the source of a great deal of literature relating to the promotion of new cement plants, but an official statement that the cement required for the canal would be only 4,500,000 bbl., served to put at rest exaggerated ideas concerning the importance of this work to the cement trade.

In THE MINERAL INDUSTRY, Vol. XV, it was stated that 1905 marked for the first time a transference of the center of the industry. The figures for 1907 show that the pendulum has again swung in the old direction. In 1905, 49.3 was the percentage of production in what is known as the Lehigh district, embracing the mills in the Lehigh valley and those in



New Jersey around Phillipsburg. In 1906 this same region produced 49 per cent. of the total output of the United States. The year 1907 shows a percentage of 50.9 produced in the Lehigh-Jersey district, and the accompanying diagram gives an interesting illustration of the rise and fall of the production of cement in this originally developed and central field, and in a measure verifies the thought that for some years to come this district is likely to remain to the cement industry what Pittsburg is to the iron and steel trade.

REVIEW BY STATES.

The statistics of the cement production of the United States in 1907, as collected by E. C. Eckels, of the U. S. Geological Survey, are as follows: Alabama, Georgia, West Virginia, Virginia, 1,274,470 bbl.; Colorado, Utah, Texas, South Dakota, Arizona, 1,399,472; Kentucky and Missouri, 3,186,925; California and Washington, 1,893,004; Illinois, 2,036,093; Indiana, 3,782,841; Kansas, 3,353,925; Michigan, 3,572,668; New Jersey, 4,449,896; New York, 2,290,955; Ohio, 1,151,176; Pennsylvania, 20,393,965; total, 48,785,390 barrels.

Alabama (By Eugene A. Smith).—The raw materials suitable for the manufacture of portland cement occur in Alabama both in the Coastal

Plain region and in the so-called Mineral district (Paleozoic). In the Coastal Plain the Selma chalk of the Cretaceous and the St. Stephens limestone of the Tertiary are the two limestone formations available for this use. The St. Stephens limestone is in part soft and friable and is free from insoluble impurities and magnesia. It is well known in Mississippi, Alabama, Georgia and Florida as the "chimney rock." It is often quarried by means of a crosscut saw, and afterward cut and dressed (with saw and plane) into blocks of proper size for building chimneys and pillars for houses. It is an excellent material for portland cement manufacture, containing from 90 to 96 per cent. of carbonate of lime. It is well exposed along the Tombigbee river, at St. Stephens, and at Oven Bluff; and on the Alabama, at Marshall's Landing and the neighboring bluffs.

This limestone belt is crossed by the main line of the Southern Railway at Glendon, Clarke county, and by the Louisville & Nashville Railroad below Evergreen, on Murder creek, and at other points in Conecuh county. In the southeastern part of the State much of this rock has become silicified, and is thus not available. In all the region of its occurrence suitable clay for mixing with the limestone is found in the residual clays overlying the limestone, and in the clays of the Grand Gulf formation. The Mobile Portland Cement Company, recently organized, projects a large plant at St. Stephens, a most convenient locality for the raw materials, but of course distant from the coal fields. With the opening of the river by locks and dams, many of which are already completed and others in active construction, this will not be a serious matter as the coal can be transported at a cheap rate by barges.

The middle third of the Selma chalk has some 300 ft. thickness of rock which is of suitable grade for portland cement, some of it having naturally very nearly the proportion of clayey matter required for the cement mixture. Overlying the limestone in most places are residual clays which have been found by experience to be of the right composition for the mixture; also in the flatwoods nearby there is a great abundance of suitable clay.

The first cement plant in Alabama was erected at Demopolis, and utilized these limestones and clays. This same limestone outcrops across the State from the vicinity of Montgomery to the Mississippi line and beyond. It crosses the Alabama river at White Bluff and the Tombigbee at Demopolis, following that river up as far as Epes. Above Epes are also bluffs of it to Gainesville. The locality at Epes seems to be exceptionally favorable for the location of a plant for this manufacture.

The Paleozoic limestones which are suitable for cement making belong to the sub-Carboniferous and Trenton formations; the clays or shales needed for the manufacture occur in close proximity to the limestones in

the Coal Measures, the sub-Carboniferous, and in the lower Silurian and Cambrian formations. The coal is also always close at hand. The only plant in this district at present is that of the Standard Portland Cement Company, at Leeds, in Jefferson county, using the Trenton limestone and Carboniferous shales for raw materials. Another plant will probably soon be erected as Village Springs, in Blount county.

In addition to supplying the market afforded by such cities as Birmingham, Chattanooga, Atlanta, Montgomery and Mobile, cement plants situated on the navigable rivers of Alabama will be enabled to place their product at any point on the Gulf or South Atlantic seaboard at very low freight rates, owing to the cheapness of transportation by water as compared with railroad freights which most other plants will be compelled to meet.

Illinois (By H. Foster Bain).—Three sorts of cement are now made in this State, four plants manufacturing portland, two making natural, and one puzzolan cement. As the last does not reach the general market, only the natural and portland cement plants need be considered.

For nearly half a century natural cement has been manufactured at Utica, on the Illinois river. Previous to the great development of the portland cement industry the mills of this locality furnished a large output. There is still available an abundance of raw material and two plants, those of the Illinois Hydraulic Cement Company, and the Utica Hydraulic Cement Company, are operated. Each plant includes two kilns of about 400 bbl. daily capacity and the combined daily output is accordingly approximately 1600 bbl. The material used is derived from the lower magnesian formation here brought to the surface by the Lasalle anticline. It is an impure dolomite as is shown by the analyses in the accompanying table.

ANALYSES OF UTICA NATURAL CEMENT ROCK.

	1.	2.	3.	4.	5.	6.	7.	8.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Silica	29.84	27.70	26.46	11.89	27.60	15.02	14.42	4.58
Iron	1.52	1.41	1.36	1.35	0.80			
Alumina	3.36	2.33	3.39	11.61	10.60	8.20	11.34	3.72
Lime	30.17	29.94	30.30	29.51	33.04	25.40	26.12	28.36
Magnesia	20.69	20.01	20.81	20.38	17.26	12.50	9.82	18.30
Ignition	10.24	16.03	13.38	....	....	38.54	38.70	44.92
Water at 105 deg.	....	....	....	....	....	....	....	0.11

Nos. 1, 2 and 3 by E. B. Lihme; Nos. 4 and 5 quoted from Eckel's "Cements, Limes and Plasters;" Nos. 6, 7 and 8 from State Geological Survey.

The lower magnesian outcrops extensively and in favorable situations in the vicinity of these plants. It is found also in Ogle and in Calhoun counties, and probably equally good rock could be found if there were any reason for searching in other localities. It does not, however, seem probable now that the natural cement industry will experience any important revival until conditions change materially.

Portland cement is manufactured in two districts, the Chicago and the Lasalle, and a new plant is about ready for operation at Dixon. At Chicago the cement is made by combining blast furnace slag and limestone. The slag is derived from the furnaces of the Illinois Steel Company, and the limestone is furnished by the Caparis Stone Company, from quarries near Fairmont, in Vermillion county. The stone is a non-magnesian limestone found in the Coal Measures and is now worked by stripping, followed by blasting and steam-shovel work. The deposits are flat-lying and are locally extensive. They occur in the midst of a level drift plain and except for a small outcrop were located entirely by drilling from the surface. The Universal Portland Cement Company, a branch of the Steel Corporation, which controls this industry, is building a large additional cement plant in the Chicago district, though across the line in Indiana.

The most important cement manufacturing district in Illinois at present is at Lasalle. Here the presence of suitable material, the abundance of coal, and favorable transportation, both by rail and water, have led to the rapid building up of the industry. The proposed Lakes-to-the-Gulf deep waterway runs through the heart of the district and while at present water shipments are insignificant, the presence of a navigable river and a small canal has not been without influence in the adjustment of freight rates. Both eastern and western railways traverse the district and the old main line of the Illinois Central runs through it from north to south.

The material used at Lasalle is a Coal Measure limestone with which are associated suitable shales. The rock outcrops through a limited district, changing in character and diminishing in thickness away from the productive center. Its distribution has been recently traced for the State Geological Survey by Gilbert H. Cady and a report is in preparation. It usually forms a ledge from 15 to 30 ft. in thickness and is commonly divided into two well defined divisions of which the upper is harder and in natural outcrops overhangs the lower. The upper member is 5 to 15 ft. in thickness, depending on the depth of the pre-glacial erosion. Below it is a calcareous shale varying from 8 in. to 3½ ft. in thickness. Still lower is the bottom ledge ranging from 6 to 16 ft. in thickness. These various members, with the shale below, permit a flexible adjustment of mixtures. Characteristic analyses are given in the accompanying tables.

ANALYSES OF PORTLAND CEMENT ROCK AT LASALLE, ILL.[1]

	Upper Bed.					Lower Bed.				
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2	4.92	4.43	2.66	2.88	1.98	17.76	9.62	15.24	10.34	8.24
Fe_2O_3, Al_2O_3	3.08	2.86	1.96	2.24	1.56	9.56	5.56	7.58	4.40	3.40
CaO	50.52	57.32	52.32	51.78	53.32	36.64	46.08	41.54	45.58	47.72
MgO	.89	.59	.58	.69	.75	2.42	.74	1.14	1.38	1.31
Volatile	41.06	41.92	38.54	42.06	42.66	34.36	39.16	38.80	37.88	38.90

[1] Analyses by State Geological Survey.

ANALYSES OF CLAYS USED AT LASALLE CEMENT PLANTS.

	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2..........	53.12	54.30	52.70	49.02	(a)53.48
Al_2O_3........	20.60	19.33 }	21.73	20.14	(a)22.36
Fe_2O_5........	4.09	5.57 }			
CaO..........	4.02	3.29	12.37	13.82	(b)7.64
MgO..........	2.24	2.57	2.01	2.10	(b)1.78
Volatile.......	13.70		11.27		(b)

(*a*) Analyses quoted from Eckel's: "Cements, Limes and Plasters."
(*b*) Analyses by State Geological Survey.

The rock is worked both in open pits and by underground methods. The companies operating in the district are the Chicago Portland Cement Company, the Marquette Portland Cement Company, and the German American Portland Cement Company. The first-named has a daily capacity of 1500 bbl. which it is expected to increase shortly to 4500 bbl. The second has a capacity of 2500 and last of 1800 bbl.

At Dixon, the Trenton limestone, the western equivalent of the famous Lehigh cement rock, outcrops and is about to be utilized by the Sandusky Portland Cement Company. A characteristic section in this vicinity is as follows: (5) yellow clay, 4 to 7 ft.; (4) gravel, 2 to 3 ft.; (3) gray, fossiliferous limestone in 1-in. layers, 4 to 6 ft.; (2) light gray limestone in beds 3 to 9 in. thick and including a 4-in. shale band at top, 10 ft.; (1) bluish-gray, fine-grained, hard, fossiliferous limestone, in beds 8 to 16 in. thick, 9 ft. Beds 1, 2 and 3 of this section, as sampled by Mr. Cady and analayzed in the Survey laboratory, gave the following, the results being given for beds 1, 2 and 3 respectively in the order named: SiO_2, 7.56, 5.10, 4.78 per cent.; Fe_2O_3 and Al_2O_3, 3.54, 2.58, 4.44; CaO, 48.48, 45.84, 47.04; MgO, 0.60, 4.58, 2.04; Ignition, 40.54, 41.94, 41.92; H_2O at 105 deg., 0.09, 0.52, 0.14.

Aside from the localities mentioned there are a large number of points in the State at which materials suitable for manufacture into cement occur. These are now being examined by the State Geological Survey, samples having been collected and analyses made from rock at 92 localities. In general it may be stated that the limestones of the Carboniferous afford the best material. In the Lower Carboniferous, or Mississippian, the maximum of favoring conditions are found in the Chester group. Below that the limestones of this series, while generally low in magnesia, are usually so filled with flint nodules as to necessitate great care in the selection of a site. The Chester includes three heavy limestone beds usually associated with shale. It outcrops in the area between the edge of the Coal Measures and the Mississippi and Ohio rivers, so it is very favorably situated as regards transportation, fuel and water. Between East St. Louis and Thebes there are a number of favorable sites.

Within the area underlaid by the Coal Measures there are numerous

outcropping limestones similar to those now utilized at La Salle and Fairmont. A number of them have been sampled and it is very probable that among them some will prove suitable as to thickness and composition. It is clear that the situation of such limestone, in the heart of the coal-fields, is exceptionally good, and it may be confidently predicted that Illinois will ultimately take high rank among cement producing States.

Michigan (By A. C. Lane).—The portland cement plants of Michigan did not have too good a year in 1907, and some of the weaker and more heavily capitalized were pushed to the wall, notably the Great Northern, at Marlborough. It is noticeable, however, that all the plants which originally planned to use limestone are doing well, and a number like the Elk, Hecla, and Newaygo have changed over from using boglime (marl) and the wet process, and are looking for limestone. On the other hand there is a group of plants near the south line of the State which are marl plants and are doing well.

The portland cement plants of Michigan are of two different types, those using limestone and the dry process and those using boglime and the wet process. In the first group are several plants near Alpena, viz., the Alpena, El Cajon, and a new one, the Huron, built on the strength of the success of the others. On the same belt of Trasverse (Hamilton) limestones on the other side of the Peninsula, at Petoskey and Elk Rapids, is the Elk. This reports about 200,000 bbl. of cement made in 1907, with about 31,000 bbl. carried over. Of the plants in the central part of the State, the Great Northern has shut down; the Newaygo has a valuable waterpower; the Hecla is near the Saginaw coal-fields, but has also been in difficulties. The moral is that something more than a good marl bed is necessary at present prices to make a safe cement undertaking, and it is cheaper to grind limestone than to dry the water out of the marl. The plants near the southern part of the State, such as the Wolverine, Omega and Peninsula, have certain freight advantages in their position.

Portland cement is also made incident to the soda ash business by the Fords, at Wyandotte. The plant at Bellevue uses sub-Carboniferous, Bayport limestone, which was at one time shipped to Wyandotte, but was not pure enough for chemical uses. Mr. Beal, the chemist of the Wyandotte plant, took advantage of the stopping of the rotary to make a complete and interesting series of samples and analyses showing the exact stages in the process. Professors Campbell and White, at Ann Arbor, have published a very important series of tests, showing the effect of free and combined magnesia in a cement.

New Jersey (By Henry B. Kümmel).—The portland cement industry of New Jersey is an important one; in fact the State is second only

to Pennsylvania in the amount annually produced, manufacturing in 1907 about 9 per cent. of the total for the United States. There are at present three active plants, all of them in Warren county, near Phillipsburg. During 1907 three new kilns were added at one of these plants, making a total of 55 kilns in operation. These plants used upward of 971,000 tons of cement rock and 195,000 tons of pure limestone, and produced 4,517,453 bbl. of cement, valued at the mill in bulk at $4,344,090. Comparing these figures with those of 1906, as reported by the U. S. Geological Survey, we find an increase in production of 93,805 bbl. and a decrease in value of $101,274. The small increase in production and the decrease in value finds explanation, of course, in the general stagnation in all lines of business during the last two months of the year. This resulted in at least the partial shut-down of all the mills for varying periods during November and December. During 1906 prices averaged $1.005 per bbl. at the mill exclusive of the package, while during 1907 the average price, as compiled from the above returns, was 96.2c., and during December prices were quoted as low as 75@85c. per bbl. net.

Inasmuch as the total cost per barrel, when all fixed charges such as interest on the plant, depreciation, etc., have been allowed for, seems to be not less than 80c. and maybe 85c. or more, it is apparent that there was not an excessively large margin of profit for the manufacturers during 1907, although conditions were much better than in 1904 and 1905, when the average price seems to have been between 75 and 80c. per bbl. The bearing of these facts upon the many efforts recently made to float new cement companies both in New Jersey and Pennsylvania, by the means of highly colored and alluring prospectuses and promises of "net profits of 60c. per bbl." which shall be "as sure in this company as a 4 per cent. in a bank or trust company or a 7 per cent. Pennsylvania railroad stock," needs no emphasis. The cement business has unquestionably been profitable, else its wonderful development during the last decade would have been impossible, but it is far from being a get-rich-quick industry, except perhaps for the promoters who sell stock on the basis of these grossly exaggerated claims.

In New Jersey, as in nearly all the Lehigh region of Pennsylvania, the cement rock used is somewhat deficient in lime, necessitating the addition of a certain percentage of high-grade limestone. Naturally for this purpose the attempt is made to secure as pure a limestone as possible. The amount added to the cement rock at the New Jersey mills varies from 18 to 22 per cent. of the total mixture and the aggregate amounted in 1907 to about 195,000 tons. The greater part of this is obtained from quarries at Annville and Palmyra, Penn., but about one-third or less is supplied from New Jersey, chiefly from the white limestone quarries near Franklin Furnace.

New York (By D. H. Newland).—The companies manufacturing hydraulic cement reported for 1907 an output of 3,245,729 bbl., with a value of $2,971,820. The totals consist of 2,105,450 bbl. of portland cement, valued at $2,214,090, and 1,137,279 bbl. of natural rock cement, valued at $757,730. In 1906 there were 4,114,939 bbl. produced, valued at $3,950,699, so that there was a loss for 1907 of 869,210 bbl. in quantity and $978,879 in value. The poor showing has been due largely to the unfavorable conditions that obtained in the natural-cement trade, which has shown a steady decline for several years.

North Dakota (By A. G. Leonard).—The cement industry of North Dakota is still in its infancy and there is only one mill engaged in the manufacture. The cement rock is confined to the northeastern corner of the State, where Niobrara formation outcrops at many points in eastern Cavalier county. The rock is a highly calcareous clay shale, or marl, of a gray color, mottled with white spots. The composition, as shown by a large series of analyses, lies within the following range: Silica, 9 to 15 per cent.; alumina, 4 to 8; iron oxide, 2 to 3; carbonate of lime, 63 to 75; carbonate of magnesia, 1 to 2.5.

Beds whose composition falls within the above limits appear to be confined to certain horizons of the formation and are seldom over 8 or 9 ft. thick. The rock is mined and run up an incline from the mouth of the slope to a platform over the upright kilns. The cement marl is dumped into the top of the kilns and after burning, the clinker is drawn out of the bottom, ground up, and sacked. The plant has from the start been at a disadvantage because of its distance from the railroad—it is nine miles to the nearest station—but it is expected that a railroad will be built to it in the spring.

The plant runs only a few months of the year and turns out from 7000 to 9000 bbl. annually. It has at present a capacity of over 250 bbl. a day, but does not run up to its full capacity.

CHROMIUM AND CHROME ORE.

By Charles G. Yale.

The chrome ore produced in the United States comes wholly from California. Some years ago several thousand tons a year of chrome iron ore were shipped from California to the Eastern States, but of late the shipments have virtually ceased because there are no sailing vessels, giving low freights, between San Francisco and New York. Work has entirely ceased in the chrome deposits of Glenn, Tehema, Placer, Alameda, and San Luis Obispo counties, because the mining did not pay, the higher grade of ores having been exhausted. The production in California is now virtually confined to Shasta county, where a few hundred tons are annually mined and utilized for lining furnaces at the copper smelters within the State.

The Low Divide chrome mines in Humboldt county, near the coast line, were recently sold, as were some other deposits in Shasta county near Sims, and it is understood that attempts will be made to make these deposits profitable by suitable development, etc. Most of the chrome ores in the coast range of mountains of California assay less than 50 per cent. Cr_2O_3; many of them carry only 40 per cent. This is too low a grade for shipment. In San Luis Obispo some years ago concentrating works were established for chrome ores, but these were long since given up as the enterprise did not pay. The Livermore deposits in Alameda county, formerly productive, are no longer so, and little or nothing is being done there. The relative importance of the domestic production in comparison with the consumption is shown by the accompanying table.

STATISTICS OF CHROME ORE IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production (a)			Imports.			Consumption.	
	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.	Value per Ton.	Long Tons.	Value
1897	*Nil.*	......	......	11,566	$186,313	$16.11	11,566	$186,313
1898	*Nil*	......	......	16,304	272,234	16.70	16,304	272,234
1899	*Nil.*	......	......	15,793	284,825	18.03	15,793	284,825
1900	140	$1,400	$10.00	17,542	305,001	17.39	17,682	306,401
1901	130	1,950	15.00	20,112	363,108	18.05	20,242	365,058
1902	315	4,725	15.00	39,570	582,597	14.73	39,885	587,322
1903	150	2,250	15.00	22,931	302,025	13.13	23,081	304,275
1904	123	1,845	15.00	24,227	348,527	14.38	24,350	350,372
1905	40	600	15.00	54,434	725,301	13.32	54,874	725,901
1906	317	2,859	9.00	43,441	557,594	12.84	43,758	560 453
1907	335	5,620	20.00	41,999	491,925	11.71	42,333	498,605

(a) Reported by the California State Mining Bureau except for 1907.

The chief consumers of chrome ore for chemical purposes are the Kalion Chemical Company, of Philadelphia, and the Baltimore Chrome Works, of Baltimore; the Harbison-Walker Refractories Company, of Pittsburg, is the chief manufacturer of chrome brick. These and other Eastern consumers obtain their supply of chrome ore, duty free, from New Caledonia, Greece, Canada and Turkey.

Market Conditions.—Quotations on chrome ore were steady throughout 1907. The price during January and February was $17 @ $19.75 per long ton for 50 per cent. ore at New York; during the remainder of the year $17.50 @ $20 was quoted. Chrome bricks, f.o.b. Pittsburg, were quoted at $175 per M. during the entire year. Market conditions presented nothing of special interest.

Chrome Ore in Foreign Countries.

By Edwin Higgins.

Canada.—The production of chrome ore in this country is treated in a special article following this one.

Greece.—A large part of the chrome ore mined in this country comes from one deposit in Thessaly, mined by Messrs. Apostolides.

THE PRINCIPAL SUPPLIES OF CHROME ORE. (*a*)
(In metric tons.)

	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907
Bosnia........	396	458	200	100	505	270	147	279	186	320	(*c*)
Canada.......	2,393	1,834	1,824	2,119	1,156	817	3,184	5,512	7,781	7,936	6,528
Greece........	563	1,367	4,386	5,600	4,580	11,680	8,478	15,430	8,900	11,530	(*c*)
India.........	.....	.....	.....	.....	.....	.....	260	3,654	2,751	4,445	(*c*)
New Caledonia (*b*).........	9,054	14,300	12,480	10,474	17,451	10,281	21,437	42,197	51,374	57,367	31,552
Newfoundland	3,084	657	717	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	(*c*)	(*c*)
New South Wales.......	3,433	2,145	5,327	3,338	2,523	454	1,982	403	53	15	30
Norway.......	*Nil.*	*Nil.*	41	165	85	22	*Nil.*	154	*Nil.*	*Nil.*	(*c*)
Russia........	13,433	15,467	19,146	18,233	22,169	19,655	16,421	26,575	27,051	(*c*)	(*c*)
United States.	*Nil*	*Nil.*	*Nil.*	142	132	320	152	125	40	322	339

(*a*) From the official reports of the respective countries. No statistics are available from Turkey. (*b*) Exports. (*c*) Statistics not yet available.

New Caledonia.—There were no new developments of consequence in the chrome ore industry during 1907. The Lucky Hit and Tiebaghi mines continued to be the principal sources of production, although the latter is at present inactive pending alterations in the mode of transport. The Tiebaghi, the largest and richest chrome mine known, is situated in the northern part of the island, on the top of Mount Tiebaghi. The ore is exceptionally rich, as high as 57 per cent. chromium sesquioxide being often encountered. This mine has facilities for producing 5000 tons of ore per month.

The Lucky Hit is one of a group lying 20 km. south of Noumea, where a vein with a maximum thickness of 15 m. is mined. The ore carries between 30 and 40 per cent. chromium sesquioxide. This mine has a capacity of 12,000 tons per year.

Ore freights from New Caledonia to London via Sydney range from £1 5s. to £1 6s. per ton plus transhipment charges of 4s. per ton. at Sydney. To the Continent by steamer via Sydney the rates are some 6s. higher. Freights on ore from Noumea to Sydney vary from 4s. to 6s. per ton. Coastal freights per sailer to Europe range from £1 3s. 6d. per ton for nickel and cobalt to £1 8s. per ton for chrome.

Rhodesia.—Chrome ore is found in the neighborhood of Selukwe, where it is becoming of increased importance. A sample of this ore sent to the Imperial Institute, London, by the British South African Company, gave the following analysis: Chromium oxide, 46.36 per cent.; alumina, 13.18; ferrous and ferric oxides (calculated as FeO), 18.66; cobalt and nickel oxides, 0.17; magnesia, 13.64; silica, 4.58; water, 2.72. A fire assay of the sample showed a trace of platinum.

Chrome Iron Mining and Milling in Canada.[1]

By H. F. Strangways.

The deposits of chrome ore in the eastern townships of Quebec lie in the region known as the "serpentine belt." The center of the deposits is situated approximately at the little village of Black Lake, in the township of Coleraine, but the productive region may be said to extend for a radius of about five miles from this village. All the districts from which chrome is now shipped lie within 10 miles of Black Lake. The Quebec Central Railway passes through this region and none of the chrome pits is more than seven miles from the track.

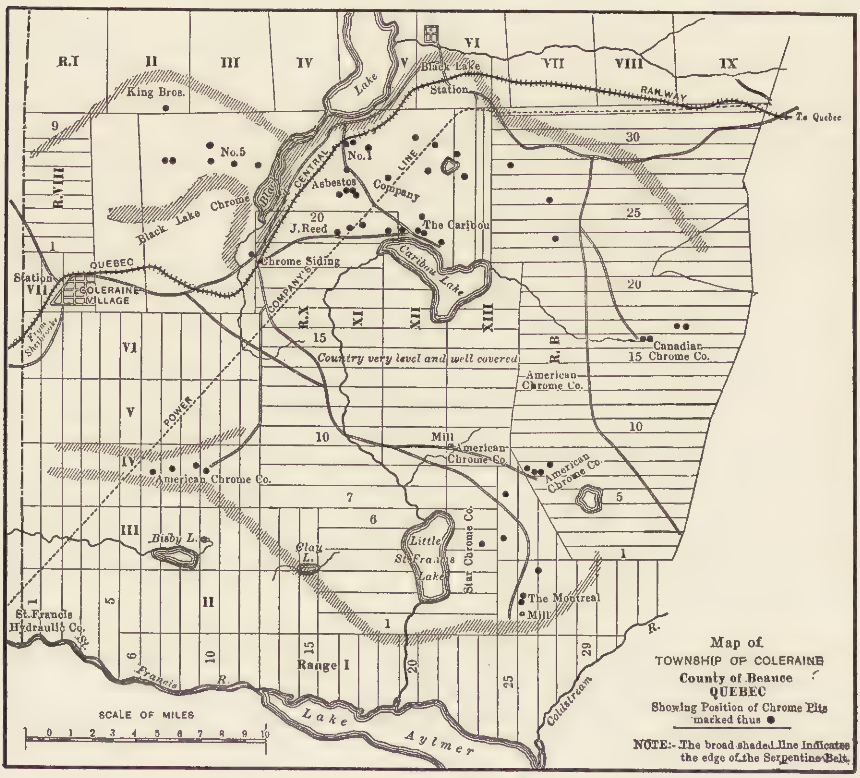
Chrome mining was begun in this region in 1894 and until recently was carried on in a very spasmodic manner. The production up to 1898 amounted to only 10,000 tons, while in 1905 alone it amounted to 7600 tons, and in 1906 it had increased to nearly 9000 tons. The greater portion of the ore is shipped to the United States. Ore which is sufficiently rich to be shipped without concentration is termed "crude," and contains at least 45 per cent. chromic sesquioxide. Concentrates also are shipped and paid for according to grade. The prices paid for both "crude" and concentrate vary between $9 and $12.50 per ton, depending on the amount of Cr_2O_3 present.

Geology.—The geology of this region is complex. The chrome deposits all lie in what is known as the "serpentine belt." This serpentine represents the alteration of a peridotite, which was intruded through sedi-

[1]Abstract of a paper read at a meeting of the mining section of the Canadian Society of Civil Engineers, held at Montreal, Nov. 21, 1907.

mentary rocks, probably of Cambrian age. Chromite has been found only in peridotites and allied magnesian rocks, or in serpentines which have resulted from the alteration of these rocks. There are alluvial deposits of chrome, but these can be traced back to the original peridotite rocks. The New Caledonia deposits are of this type.

Up to the present time comparatively little investigation has been made into the origin of chromite deposits. The theory set forward by Pratt, Vogt, Adams, and others is, that deposits of chrome ore, wherever found,



are segregations which have crystallized out from igneous magmas on cooling. Pratt, in 1899, in speaking of the North Carolina deposits, says: "My observations have shown that large deposits of chromite occur in the peridotite rocks, near the contact of these rocks with the inclosing gneiss. Also, that where there is but a small amount of chromite, either in pockets or crystals, these are more abundant near the contact and diminish in number toward the center of the mass. The constant occurrence of the chromite in rounded masses of varying size near the contact of the peri-

dotite with the gneiss, and its occurrence in the fresh, as well as in the altered peridotite, would indicate that the chromite was held in solution in the molten mass of the peridotite when it was intruded into the country rock, and that it separated out among the first minerals as this mass began to cool."

This theory of segregation can very well be applied to the eastern townships' deposits as regards the proximity of the orebodies to the contact. Nearly all the pits which have been worked lie close to the contact between the serpentine and the country rock, which here is probably Cambrian. Even if the orebodies were not near the contact the irregular and pockety nature of these deposits would lead one to accept this theory of segregation. The irregularity is further accentuated by an extensive fracturing and faulting which has taken place. The ore sometimes occurs as large masses of "crude," which gradually die away or disseminate into concentrating ore and then pass to barren rock. Sometimes, however, the "crude" will separate cleanly from the barren rock without any gradual transition, suggesting a possible re-arrangement by underground waters, although in most cases in which this happens the rock face will be found to be slickensided so that this clean separation may more probably be due to the extensive fracturing and movement which has taken place, especially as such orebodies represent a line of weakness along which movement would occur.

Mining.—Canadian chrome ore is produced principally by three companies, the Black Lake Chrome Asbestos Company, the Canadian Chrome Company, and the American Chrome Company. Of these the first is by far the largest. In its extensive holdings the company has several chrome pits, only four of which have been worked to any great extent. These are No. 1, the Caribou, No. 5, and the Montreal. The company employs between 50 and 75 men when four pits are all in full working order. No. 1 pit is the largest single chrome pit in Canada. It is admirably situated as regards transportation, power and water facilities.

Originally there were three distinct orebodies at the surface; one of these was very small and soon pinched out; the other two were worked by an open cut to a depth of 70 ft. At this point both bodies showed signs of pinching out and the company expected to lose them altogether. The pit at the surface measures roughly 100x50 ft., which can be taken as the dimensions of the orebody at this point; at the 70-ft. level the width of the orebody is only 5 ft. However, on following this narrow strip of ore, it was found that both bodies took a sudden pitch and then widened out considerably, the distance between the hanging and foot wall being as much as 40 ft. in some places. After several modifications of the mining methods, made necessary by the occurrence of several accidents, the pit was considered too dangerous to be worked and about a year ago an incline

shaft was sunk, starting about 40 ft. to the left of the present pit. At present the shaft is 270 ft. deep and sinking is being continued.

A certain portion of the ore as it is mined is sufficiently rich to be shipped; this includes all above 43 per cent. chromic oxide, and is known as "crude." This proportion varies, but the average for this pit is about 15 per cent. of the total rock hoisted. The mill waste, or concentrating ore, varies in contents of chromic oxide from 7 per cent. to 40 per cent., and amounts to about 45 per cent. of the total rock hoisted; the rest, amounting to about 40 per cent., is dump material. Nothing, roughly speaking, under 7 per cent. is put through the mill.

Milling.—The mill is situated at the siding some 1200 ft. from the pit, and a tramway connects the two. The mill scheme is a simple one, being as follows:

From the cars the ore is dumped into a 100-ton bin, falling thence to a 12x15 Blake crusher. From the crusher it is elevated to a shaking trough, which distributes the ore to the battery or feed bins. From the bins the ore is fed by Challenge feeders to the stamps, which, in this mill, number 30. The duty of the stamps is in the neighborhood of 2½ tons per day; their weight is 1100 lb. and the drop 8 in., which, as the dies wear, increases to as much as 15 in. The mortar has a double discharge and the screens are 20 mesh. Each battery of five stamps discharges its pulp directly through a pipe to a single Wilfley table, and each table makes three products—heads, middles and tails. The heads are shoveled from receiving boxes into barrels, wheeled to the concentrate shed and dumped there to drain. The bulk of these concentrates is large, amounting to as much as 15 tons a day, and two men are kept fully occupied in handling them. A drag conveyer was once installed for this purpose, but the wear and tear due to the hardness of the ore was so great that this method was abandoned. The middles from all six tables fall into a launder and are passed thence to a separate Wilfley table which makes two products, heads and tails. These heads are of lower grade than the heads from the first six tables and are put through the mill again.

Other Pits.—The Caribou pit lies about 2½ miles to the south of No. 1, and a wagon road connects the two. A large quantity of "crude" of good grade has been taken from this pit, but at the present time no work is being done in it.

No. 5 pit is situated on the west side of Black Lake, and is not being worked at the present time, although a considerable quantity of ore has been extracted from it in the past. The "crude" from this pit is of the highest grade of any found in the district, running as high as 56 per cent. chromic oxide.

The Montreal pit was originally owned and worked by the Eastern

Townships Chrome Iron and Milling Company. It lies about 7½ miles east of the railway, and a wagon road connects it with Chrome Siding, whence all its ore is shipped. Chrome Siding is about 1½ miles south of No. 1 pit. At present the Montreal pit and No. 1 are the only two out of the four which are being worked. The Montreal has its own mill.

Up to 1898 there had been taken from this property 3000 tons of ore, and there was every indication of its being able to furnish many times that amount. Numerous outcrops of ore from 12 to 15 ft. in thickness occurred along a length of some 300 ft., but when worked were found to pinch out.

The present company took the property over two years ago and since that time a considerable quantity of ore has been taken out. Some diamond-drill exploration work has been done and has proved very satisfactory. The size of the body as at present developed by the borings is 400 ft. in length, with an average of 50 ft., while the thickness varies from 16 to 50 ft. There is a covering of barren rock some 50 ft. in thickness over this body. The deposit will be mined probably by a room and pillar system, although no very regular system can be employed on account of the variation in the thickness. The ore is roughly sorted in the pit and the large pieces of "crude" are hoisted separately and dumped at the surface in heaps to be cobbed.

The mill is situated on the hillside about 400 ft. from the pit and about the same distance from the lake. The scheme is the same as that at No. 1 mill. The ore is fed to a Blake crusher, falling thence to the battery bins. Fifteen stamps, in batteries of five, discharge the pulp through 20-mesh screens to three Wilfley tables. The middlings from these three tables all go to a fourth, which makes lower-grade heads and tails. These heads are re-fed to the crusher and again passed through the mill.

Canadian Chrome Company.—This company's property is situated about four miles from Thetford, from which point all its ore is shipped. The company has opened up four small pits, all within a distance of 400 ft.; only one of these pits is being worked at present, but it is by far the largest of the four, being some 100x125 ft. at the surface and 60 ft. in depth. There is a small amount of "crude" and a considerable quantity of concentrating ore showing in the present workings. A single-cable hoist, similar to those used in the asbestos pits, elevates the ore to a tramway carried on a high trestle. The ore is conveyed on the tramway to the mill, dumped into a chute which feeds into a large Blake crusher, and is elevated from the crusher to the battery bins. The mill consists of 20 stamps, delivering pulp to four Wilfleys; the middles all go to a fifth Wilfley. The mill scheme is similar to that of all other mills in the district.

COAL AND COKE.

BY FLOYD W. PARSONS.

The coal industry in the United States during 1907 was affected less by the scarcity of money and the later recession of industrial activity, than any other line of business. The year also was uneventful so far as

PRODUCTION OF COAL IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1906 Short Tons.	1906 Value at Mines. Total.	1906 Value at Mines. Per Ton.	1907 Short Tons	1907 Value at Mines. Total.	1907 Value at Mines. Per Ton.
Bituminous.						
Alabama	12,851,775	$17,349,896	$1.35	14,417,863	$19,608,294	$1.36
Arkansas	1,875,569	2,438,240	1.30	1,930,400	2,606,040	1.35
California	80,000	232,000	2.90	29,800	50,660	1.70
Colorado	10,308,421	13,916,368	1.35	10,920,527	15,179,533	1.39
Georgia and N. Carolina	363,463	407,247	1.12	365,300	423,748	1.16
Illinois	38,318,848	39,467,108	1.03	(c)51,317,146	54,396,175	1.06
Indiana	11,422,000	11,878,880	1.04	11,692,072	12,276,676	1.05
Indian Territory (f)	2,980,600	5,663,140	1.90	3,450,000	6,486,000	1.88
Iowa	7,017,485	11,619,455	1.60	(a) 7,568,424	11,882,426	1.57
Kansas	6,010,858	8,935,195	1.49	6,137,040	9,328,301	1.52
Kentucky	9,740,420	10,714,462	1.10	10,207,060	11,431,907	1.12
Maryland	5,014,995	6,772,243	1.35	5,529,663	6,617,354	1.20
Michigan	1,370,860	2,193,376	1.60	(b) 1,898,446	3,758,923	1.98
Missouri	3,860,000	6,176,000	1.60	4,350,000	6,916,500	1.59
Montana	(a) 1,787,934	3,186,620	1.78	1,810,000	3,348,500	1.85
New Mexico	1,973,658	2,960,487	1.50	(a) 2,302,062	3,729,340	1.62
North Dakota	(a) 300,998	937,894	1.45	268,300	563,430	2.10
Ohio	27,213,495	29,934,845	1.10	32,465,949	35,712,244	1.10
Oregon	(a) 79,731	212,338	2.66	51,600	141,900	2.75
Pennsylvania	129,532,991	145,076,950	1.12	149,759,089	172,222,952	1.15
Tennessee	6,272,457	7,565,286	1.20	6,760,017	8,179,621	1.21
Texas	1,290,600	2,064,960	1.60	1,300,000	2,080,000	1.60
Utah	1,839,219	2,942,750	1.60	1,967,621	3,364,632	1.71
Virginia	4,546,040	8,501,095	1.87	4,570,341	8,244,614	1.80
Washington	3,293,098	6,421,541	1.95	3,713,824	7,427,648	2.00
West Virginia	46,452,000	44,129,400	0.95	47,205,965	46,733,905	0.99
Wyoming	5,805,322	10,159,314	1.75	6,218,859	10,883,003	1.75
Alaska and Nevada	10,000	40,000	4.00	15,500	60,450	3.90
Total Bituminous	341,612,837	$401,717,090	$1.18	388,222,868	$463,654,776	$1.20
Anthracite.						
Colorado	50,000	155,000	3.10	45,113	145,264	3.22
New Mexico	20,000	70,000	3.50	17,000	64,600	3.80
Pennsylvania	72,139,566	166,082,002	2.30	86,279,719	198,443,354	2.30
Total Anthracite	72,209,566	$166,307,002	$2.30	86,341,832	$198,653,218	$2.30
Total Coal { Sh. Tons	413,822,403	568,024,092	1.37	474,564,700	662,307,994	1.40
Total Coal { Met. Tons	375,336,920	....	1.51	430,430,183	....	1.54

(a) For the fiscal year ending June 30. (b) For the 12 months ending Nov. 30, 1908. (c) As reported by the U. S. Geological Survey. (f) Includes Oklahoma.

serious labor troubles were concerned, the only difficulties being of a local nature. The Miners' Federation was, however, active throughout this period of quiet, and many new districts heretofore employing non-union

labor were organized and brought into the union fold. This activity confined itself principally to western States, and in many instances the mines are not only employing union labor, but also have submitted to the adoption of the check-off system which requires that the operating company shall itself deduct from each miner's pay the regular union dues, as well as all fines and similar charges. This check-off system is taken advantage of at many mines in such an undesirable manner that the plan is highly objectionable, and is sure eventually to cause friction and work further harm. The scarcity of miners, which was so apparent during the early months of 1907, gradually disappeared, and at the end of the year all coal plants throughout the country had a sufficient labor supply.

COAL PRODUCTION IN THE CHIEF COUNTRIES OF THE WORLD.
(In metric tons.)

Countries.	1902	1903	1904	1905	1906	1907
Asia:						
China	....	....	....	....	....	10,450,000
India	7,543,272	7,557,400	7,682,319	7,921,000	9,783,250	11,200,000
Japan	9,701,682	10,088,845	11,600,000	11,895,000	12,500,000	12,890,000
Australasia:						
New South Wales	6,037,083	6,456,524	6,116,126	6,035,250	7,748,384	7,850,000
New Zealand	1,386,881	1,442,916	1,562,443	1,415,000	1,600,000	1,784,000
Other Australia	931,148	771,536	769,723	805,000	870,000	900,000
Europe:						
Austria Hungary (c)	39,479,560	40,160,823	40,334,681	40,725,000	37,612,000	39,876,511
Belgium	22,877,470	23,913,240	23,380,025	21,844,200	23,610,740	23,824,499
France	29,997,470	34,906,418	34,502,289	36,048,264	34,313,645	37,022,556
Germany (c)	150,600,214	162,457,253	169,448,272	173,663,774	193,533,259	205,542,688
Italy	413,810	346,887	359,456	307,500	(e) 300,000	(e) 225,000
Russia	16,465,836	(f)17,500,000	19,318,000	17,120,000	16,990,000	17,800,000
Spain (c)	2,807,550	2,800,843	3,123,540	3,199,911	3,284,576	(e) 3,250,000
Sweden	304,733	320,390	320,984	331,500	(e) 265,000	305,000
United Kingdom	230,728,563	233,419,821	236,147,125	239,888,928	251,050,809	267,828,276
North America:						
Canada—						
Western	1,826,221	1,791,798	2,619,816	3,183,909	3,717,816	4,780,301
Eastern	4,699,396	4,700,645	4,194,939	4,775,802	6,196,360	5,730,660
United States	273,600,961	317,272,110	318,275,920	351,120,625	375,397,204	430,430,183
South Africa (a)	2,213,275	2,957,736	3,015,000	3,218,500	(e) 3,900,000	3,945,043
Other countries (e)	3,500,000	4,000,000	4,250,000	4,550,000	5,500,000	3,475,780
Totals	804,115,125	882,865,185	867,020,658	928,049,163	988,173,043	1,089,110,496

(a) Transvaal, Natal and Cape of Good Hope. (c) Includes lignite. (e) Estimated. (f) Estimated by Minister of Finance.

The demand for the smaller sizes of anthracite was unusually large while the general hard-coal trade continued almost normal. The average price of coal at the mines in 1907 shows but little change from 1906. Anthracite mining is becoming a business of general uniformity and is hardly affected by the severe fluctuations in other lines. Anthracite is being used almost entirely for domestic purposes, and the demand for it is growing much faster than the production.

The immensity of the bituminous industry is amazing. The production of soft coal has tripled during the last 15 years and within the next two years will probably attain a total of one-half billion tons annually. Pennsylvania, West Virginia and Illinois continue to furnish the greater

part of the production, while Pennsylvania alone makes an output nearly equal to that of all the other States combined. Fair and profitable prices for bituminous coal and coke were maintained throughout 1907.

The business depression affected the manufacture of coke more than any other branch of the coal industry. Several new plants were started during 1907, but the close of the year witnessed the complete suspension of coke manufacture at many mines in all parts of the country. In the western States, the coal that would ordinarily have been manufactured into coke for use at the smelters, was thrown upon the market and greatly helped to relieve the scarcity of fuel which would have been acute if business and mining operations had been continued at the pace established during 1906. At present there are about 548 coke manufacturing establishments in the United States, which is an increase of more than 250 since 1900. This shows that there is approximately one coking plant for about every 10 mines. Reports also show that for each active coke operation during 1907 there was an average of 198 ovens in use.

PRODUCTION OF COKE IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1906			1907		
	Short Tons.	Value.		Short Tons.	Value.	
		Total.	Per Ton.		Total.	Per Ton.
Alabama	3,217,068	$7,494,680	$2.55	3,096,722	$7,463,100	$2.41
Colorado	1,133,643	2,766,089	2.44	1,097,051	2,885,244	2.63
Georgia and North Carolina	(e) 10,000	29,500	2.95	71,460	278,694	3.90
Illinois	268,693	1,205,462	....	281,400	970,830	3.45
Indian Territory	49,782	204,205	4.10	57,600	239,616	4.16
Kansas	12,000	37,200	3.10	10,500	38,850	3.70
Kentucky	74,064	169,846	2.29	77,055	177,227	2.30
Missouri	3,000	87,000	2.90	7,800	24,258	3.11
Montana	38,182	266,024	6.97	31,400	204,100	6.50
New Mexico	105,000	315,000	3.00	203,437	711,925	3.50
Ohio	293,994	1,013,248	2.90	310,640	913,282	2.94
Pennsylvania	20,681,702	56,874,681	2.75	23,516,309	63,961,640	2.72
Tennessee	484,672	1,350,629	2.79	495,200	1,386,560	2.80
Utah	282,195	234,000	2.60	324,692	883,162	2.72
Virginia	1,408,300	3,450,335	2.45	1,622,734	3,850,620	2.37
Washington	55,000	247,500	4.50	61,400	276,300	4.50
West Virginia	3,395,744	7,436,680	2.19	4,078,222	8,849,742	2.17
Other States (c)	1,820,000	5,400,000	3.60	1,650,000	5,940,000	3.60
	33,333,039	88,582,079	2.657	36,993,622	99,055,150	2.687

(e) Estimated. (c) Includes output of by-product coke for Massachusetts, Maryland, Minnesota, New York, Michigan, Wisconsin.

The periodical exodus of foreign miners from our shores was greatly augmented by the recent financial panic. Perhaps the greatest percentage of foreign labor is employed about the coke works in the Pittsburg region, and for this reason the operators in and about Connellsville are attempting to take the initiative in solving this labor problem. One of the plans suggested and largely favored is to do away with alien labor, even though the employment of native or naturalized American labor necessitates an

increase in the wage scale. The records of the larger companies seem to show that the employment of this more intelligent labor at a higher cost leads to a larger profit and does away with the annoyances and losses attendant upon the employment of ignorant foreigners. The coke region of western Pennsylvania contains 40,667 ovens, of which the Frick company controls 50 per cent. With two men per oven, it is evident that the coke industry alone in this district requires 80,000 employees. The coke industry in the Connellsville region is attended with large profits. To open about 1500 acres of Connellsville coking coal, the cost for each oven of a modern 800-oven plant is about $1100 per oven, which includes sinking the shaft, erecting the houses, etc. Some companies with modern equipment are producing coke for $1.12 per ton which insures a profit of not less than $1.20 per ton, or if we consider that each acre will have a minimum production of 8000 tons of coke, the profit per acre will amount to more than $9000. Is it any wonder that good Connellsville coking coal land is selling for $1500 to $2500 per acre?

During the first nine months of 1907 the cry of inadequate transportation facilities was heard in all parts of America not only from coal-mining operators, but from shippers in other lines of business. In some aggravated cases the railroads were to blame, but taking the situation as a whole the transportation companies did all in their power to handle the traffic originating on their lines, and where they failed, the manufacturer and the industrial shipper were as "hard hit" as the coal operator. The car situation in the Northwest was principally responsible during the winter of 1906 for the fuel famine which occurred in that territory. General conditions throughout Montana, Wyoming and Utah at the close of 1907 were not so acute, although there was a slight fuel shortage in some localities. The falling off in business was in part responsible for the relief to the fuel trade, but it is also true that most roads, especially those in the Northwest, were better prepared to handle a large tonnage.

The year 1907 brought few new coalfields to our attention, although the area embraced in the old territories was largely extended. Great development took place in practically all of the coal States. West Virginia and Illinois led in activity. The Western States furnish opportunities for new discoveries, and it is here that the future will show the greatest developments. What we desire most is a new supply of anthracite, but little hope seems to exist that we shall find a field to succeed the present district in Pennsylvania. With the completion of the Denver, Northwestern & Pacific Railroad through the Rockies, it is possible that some fairly good anthracite will be marketed, while the reports from Alaska and Washington are somewhat reassuring. In the latter State, near the city of Spokane, a coal is said to have been discovered equal to the best Eastern anthracite. The extent of the field is at present unknown.

Those interested in coal mining are much concerned as to what steps the different railway companies will take to comply with the "Hepburn law," that was scheduled to go into effect May 1, 1908, but which, according to present advices, will either be deferred in execution until Jan. 1, 1910, or will lack enforcement until the Supreme Court renders a decision. This law requires that no railroad shall transport between States any coal in the mining or sale of which it is "directly or indirectly interested." The railroad company can ship coal to other States for its own use and can sell coal to local trade within the State borders. The Union Pacific Railroad Company will probably use all the coal produced at its mines in Wyoming, to supply the fuel demands of its own lines, as well as those of the Southern Pacific and Oregon Short Line railroads. The "Gould roads" (Western Maryland, Wabash, Missouri Pacific and Denver & Rio Grande) may try to incorporate separate companies and turn over to them the coal lands now held by the various lines. It is possible that even such action as this will be considered as an evasion of the law, which will place the companies in an embarrassing position. One other difficulty now confronting the Gould lines in selling their coal holdings, or disposing of them to another company is the fact that most of the coal properties, valued at about $55,000,000, are already pledged as security for bonds.

EXPORTS FROM THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907
Anthracite	2,228,392	2,229,983	2,216,969	2,698,072
Bituminous	6,345,126	6,959,265	7,704,850	10,448,676
Total coal	8,573,518	9,189,248	9,921,819	13,146,748
Coke	523,090	599,054	765,190	874,689
Total	9,096,608	9,788,302	10,687,009	14,021,437

(a) These figures do not include coal bunkered, or sold to steamships engaged in foreign trade.

Some of the coal roads such as the Delaware, Lackawanna & Western, and the Delaware & Hudson Railroad Company, have old Pennsylvania charters, which specifically state that the railroad companies concerned may mine and sell anthracite coal. These roads may fight the present law, basing their defense on the "Darthmouth College Decision," which held that a company's charter was a contract with the State and could not be abrogated by any government, unless provision was made in the charter itself. President Baer has stated that the coal properties of the Philadelphia & Reading are completely divorced from the railroad company, the ownership being vested in the Reading Company, which is simply a holding corporation. Several railroad companies have already anticipated the enforcement of the Hepburn law, and have prepared to meet

its provisions in different ways. The Baltimore & Ohio Company has sold its stock in the Consolidation Coal Company of Baltimore to outside parties, while the Buffalo, Rochester & Pittsburg Company has transferred its coal shares to a new company, the stock of which was sold pro rata to the stockholders of the railroad company.

DESTINATION OF EXPORTS. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907
Canada	6,577,954	6,964,630	7,533,346	9,843,315
Mexico	880,747	927,170	1,084,319	1,066,502
Cuba	519,227	564,385	689,833	804,310
Other West Indies	253,585	300,776	319,839	474,382
Europe	144,354	101,277	81,734	220,479
Other countries	197,651	331,010	212,748	737,760
Total	8,573,518	9,189,248	9,921,819	13,146,748

(a) The European exports in 1907 were chiefly to Italy, that country receiving 138,648 tons. Other countries are chiefly the South American republics and Canada. The Canadian shipments were 75.9 per cent. of the total in 1906, and 74.9 in 1907.

IMPORTS OF COAL AND COKE INTO THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907
Canada	1,211,304	1,331,292	1,427,731	1,398,194
Great Britain	135,292	94,600	106,771	42,830
Australia	235,069	184,426	191,758	552,918
Japan	45,429	41,956	11,996	123,720
Other countries	759	569	6,251	8,356
Total coal	1,628,675	1,652,843	1,744,507	2,126,018
Coke	....	181,376	128,461	132,355
Total	1,628,675	1,834,219	1.872,968	2,258,373

(a) Of the coal imported in 1907, there were 9896 tons classed as anthracite. Nearly all the imports were for the Pacific coast. The features of 1907 are found in the unusually large receipts of Australian and Japanese coal. These were engaged in expectation of a repetition of the 1906 fuel famine in the West, and proved to be in excess of the demand. As a consequence, large surplus stocks of Australian coal were reported in San Francisco at the close of the year. The coke received is from British Columbia with the exception of a few thousand tons from Germany.

Review of Coal Mining by States.

Alabama (By L. W. Friedman).—During the entire 12 months, there was a steady demand on the railroads to furnish cars. There was but little cessation of development in the coalfields. The consumption fell off toward the end of the year, but apparently companies were behind in their deliveries of coal and the movement from the coalfields during the last two months of the year was active. The Southern Steel Company closed two of its mines in November and one or two others were practically closed at the same time. The production in 1907 amounted to 14,417,863 short tons. The new mines put in operation more than made up for the

mines which shut down. The price throughout the year was satisfactory. The State mine inspector's report shows the output in 1906 to have been 12,851,775 tons. The coke industry suffered considerably during 1907, and the production was less than in 1906.

Development was pushed in the western part of Jefferson county and in Walker county by the Pratt Consolidated Coal Company; in the southeastern portion of Jefferson, in the Acton basin, by H. F. de Bardeleben and associates; in the southwestern part of Jefferson by the Birmingham Iron and Coal Company, or the Atlanta, Birmingham & Atlantic railroad; in the northeastern part of Jefferson and in St. Clair county by H. F. de Bardeleben and associates.

COAL AND COKE PRODUCTION IN ALABAMA.
(In short tons.)

Year.	Coal.	Coke.	Year.	Coal.	Coke.
1900	8,273,362	2,110,837	1904	11,273,151	2,340,219
1901	8,970,617	2,148,911	1905	11,900,153	2,576,786
1902	10,329,479	2,552,246	1906	12,851,775	3,217,068
1903	11,700,753	2,693,497	1907	14,417,863	3,096,722

The year 1907 was not unusual so far as accidents were concerned. The greatest loss of life occurred at the Yolande mine on Dec. 16, where an explosion of gas and dust killed 57 persons. The best available records show that 93,623 tons of coal were produced for each life lost. Jefferson county produced more than one-half of the coal mined.

(By Eugene A. Smith.) The termination toward the southwest of the great Appalachian coalfields embraces an area in Alabama of about 8800 square miles. This area is divided into three distinct fields by narrow anticlines in which the limestones, iron ores, etc., of formations older than the coal measures, make the surface. From the main streams which drain them, these fields have been named the Warrior, the Cahaba and the Coosa. In all three fields the measures have a general dip or pitch toward the southwest, and each is a trough with its axis near the southeastern border; thus the greatest thickness of the measures in each field will be near the eastern border and at or toward the southwestern end. The maximum thickness of the measures will not fall far short of 4000 ft. The coal seams vary in thickness from a few inches up to 16 ft., but the thick seams are always more or less shaly. About 25 of these beds have a thickness of 18 in. and upward and have been worked.

Previous to 1874, it has been estimated that the total coal production of Alabama did not exceed 480,000 tons, the earliest mining operations having been carried on in the '40s, in the Trout creek and Broken Arrow regions of the Coosa field, and in the Montevallo district of the Cahaba field. Since 1874, the production has increased rapidly and in 1907,

according to the report of the State mine inspector, it was 14,417,863 tons, Alabama ranking fifth among the coal producing states of the Union.

This coal was furnished by 447 mines on about 25 different seams. Of these mines, 324 are drifts with natural drainage, 114 are slopes, and nine are shafts, and their annual production varies from 2000 to nearly 500,000 tons. The larger mines are provided with the most modern equipment for mining and raising the coal. The pillar-and-stall system is the mining method most in use, but the long wall system has been adopted in some cases. During 1907 there were employed in and around the mines 20,255 men. Fatal accidents numbered 154; of these fatalities, 57 occurred in the explosion at the Yolande mine on Dec. 16. While fire-damp exists in perhaps a fourth of the mines, they are as a whole comparatively free from such gases.

The Alabama coals are all bituminous and of a quality that compares favorably with the coals of other States. By the use of improved shaking screens, the product as mined is separated into lump, nut and slack. The first two go to the general market for steam and domestic purposes, while the slack after washing is used mostly for making coke for the iron furnaces, though some of it is used for blacksmithing. Within the last few years a good deal of run-of-mine coal from several different operations has been shipped to Demopolis and Leeds, and there used in the rotary kilns of the portland cement plants, for which purpose it has been found to be well adapted.

The principal markets for the coal are within the State, but much of it goes to South Carolina, Georgia, Tennessee, Mississippi, Louisiana and Texas; to the steamships at Mobile, Pensacola, New Orleans and Savannah; and to the export trade, chiefly to Mexico. The home supply is used mainly for the manufacture of coke for the iron furnaces, while the commercial shipments are mostly of steam coal which is supplied to almost every railroad in the South.

The growth of the coke industry in Alabama has been even more rapid than that of coal mining. It was not until 1876 that it was known that the Alabama coal would make coke suitable for iron smelting, and the State ranks now second in the Union as a coke producer. The coke production in 1907, as given in the Alabama State mine inspector's report, was 3,096,722 tons. The greater part of this coke was used in the iron furnaces of the State, though a portion of it was shipped to other States and to Mexico for smelting and foundry purposes. The present product does not supply the demand, and many new ovens are in course of construction. The coke in 1907 was made in 10,090 ovens, all of the beehive pattern except 240 Semet-Solvay ovens at Insley, and 90 at Holt in Tuscaloosa county. Nearly all the product of the beehive ovens is 48-hour coke.

Most of the product is made from slack coal, but the entire output of some of the mines, after crushing and washing, is converted into coke. With one or two exceptions, the heat and gases from the beehive ovens are allowed to go to waste; but the Semet-Solvay ovens of course, utilize these products.

Alaska.—The coal mining industry of Alaska is still practically undeveloped, the total production for 1905 being 6,660 short tons, while the output in 1907 was 10,500 tons, with a value of nearly $40,000. The most active mining operations have been off Cook Inlet, in southwestern Alaska. On the Yukon, in Seward peninsula, and at Cape Lisburne, the mines were worked to provide fuel for use, on small coastwise or river steamers, and at mining camps and canneries.

Alaskan coal lands have in recent years been the subject of considerable investigation by the U. S. Geological Survey. There are good prospects of early development of the coalfields in the coastal provinces west of Mount St. Elias, and it is probable that Alaskan coal will soon be shipped regularly to many ports on the Pacific coast.

Arkansas.—There was but little change in the coal mining industry of Arkansas during 1907. The most favorable factor was the general improvement in the car supply of the railroads. The better demand for coal occurred during January, February, October, and November. During the other months of the year the trade was exceedingly dull and the only important demand came from the railroads. The most important outside occurrence bearing upon the coal situation in the State was the complaint made by the operators before the Interstate Commerce Commission, asking for a reduction in freight rates to Texas and Louisiana. A hearing was held at South McAlester on Nov. 19; the matter is still before the Commission.

The Arkansas coalfield comprises the eastern end of a large fuel area which has its greatest extent in Oklahoma. Recent examination at the U. S. Geological Survey's fuel testing plant at St. Louis has shown that the better grades of Arkansas coal are of a quality superior to any other coal west of West Virginia. The workable coal areas, however, are more or less irregularly distributed, and many of them are in basins not known to be connected with one another. The productive areas of the eastern end of the field are north of the Arkansas river; those of the western end are south of it.

The development of the coal industry in Arkansas on a commercial scale began about 1870, and the production increased to a maximum in 1903, when the output amounted to 2,229,172 short tons. The slight decline in production during the last four years has been caused chiefly by the competition of fuel oil from Texas; present conditions seem to indicate that the demand for Arkansas coal is again increasing.

Colorado.—In addition to the opening of many new mines, the coal industry in Colorado during 1907 was marked by numerous improvements at the principal mines operated by the large companies. The chief labor difficulty occurred at the mines in the vicinity of Colorado Springs, where about 600 miners went on strike early in October, for an advance in wages and an eight-hour day, which resulted in the men returning to work two weeks later at the old scale. The agitation throughout the State for an

COAL DEPOSITS OF THE CENTRAL AND WESTERN STATES.

eight-hour day and an increase in wages ceased when the banks stopped payment of currency in October.

No orders for domestic coal were cancelled, although there was, after Nov. 1, a falling off in the demand from the metal mines. Previous to November the car supply was sufficient to handle only 70 per cent. of the production, but the close of the year showed ample transportation

facilities. The year was the most satisfactory, both in regard to tonnage and profit per ton, that the coal mining interests in Colorado have yet experienced.

(By John D. Jones.) During 1907 unusual activity was manifested in Colorado in opening and developing new mines, of which there are altogether 18; these are equipped for capacities ranging from 150 to 1500 tons daily. In addition to the opening of these new mines, a great many improvements were made in the old mines tending toward greater safety to life and property; new air-shafts were sunk and new ventilating fans of larger capacities were installed.

The relations between employer and employees were friendly, and few local disputes occurred, all of which were of short duration. The railroad car shortage was noticeable as usual in some parts of the State, especially in the last three months. The scarcity of men in the coal camps retarded the production considerably, but during the closing months of the year the money stringency relieved the labor shortage.

A general summary of Colorado's coal industry during 1907 is as follows: Number of mines in operation, 183; number of new mines opened, 18; lignite coal produced, 2,062,154 tons; semi-bituminous coal produced, 985,226; bituminous coal, 7,803,147; anthracite, 45,113; unclassified, 70,000; total tonnage, 10,965,640, or an increase of 657,219 tons over 1906. The production of coke amounted to 1,097,051 tons, while the number of ovens was 2566. The employees at the coke ovens numbered 960, and the men and boys in and about the mines numbered 12,900.

Georgia.—The coal industry in Georgia showed no unusual activity during 1907. The mines in Vade and Walker counties produced a small amount of coal which was used to supply the local demand. The slack coal from these mines was made into a good grade of coke for use in the furnaces near Chattanooga, Tenn. This coke is made by two establishments, both of which have been in operation since 1900. The general practice is to wash the coal before charging it into the oven. Although the quantity of coke produced in 1907 showed no increase over 1906, the value showed a substantial increase, as most of the product was sold at an average price of $3.90 per ton.

Illinois.—During 1907 the coal mining industry in Illinois showed great advances in all lines. Coal was produced in 45 counties, bringing the total output well up toward the 50,000,000 tons. The Saline-Gallatin coalfield, near the southwest end of the eastern interior district, has been the seat of recent investigations by the U. S. Geological Survey. This field extends into Indiana and Kentucky, and has not been generally developed or investigated in late years. The area includes parts of four Illinois counties—Saline, Gallatin, White and Hamilton—and measures 30½ miles from east to west, and 18½ miles north to south, including approximately 550 square miles. The district is at present being rapidly

developed and contains several coal seams of excellent quality. The production so far has been entirely in the Illinois portion of the area, which yielded 314,927 tons in 1906, showing a gain of 115 per cent. over 1905. Only two beds are now mined, but as several lower seams have been found to be workable in areas to the south and east and in other parts of Illinois, it seems probable that drilling may show similar conditions in various districts of this region. Illinois showed a much larger increase in production during 1907, than did West Virginia; the difference in the total production of the two States, however, is slight. The usual complaint of car shortage was often heard during the early portion of the year, but at the close of 1907, the car supply was better than at any time in recent years.

The southern districts in Illinois contain the thicker and better coal seams, and for this reason the supply of miners was better in the southern lower districts than in the northern field, where the scarcity of labor was severely felt. It is stated that the cost of mining during 1907 was nearly 5c. per ton greater than in 1906; the selling price, however, showed but little advance over 1906. Although there are 933 mines and openings of all kinds in Illinois, the number of shipping or commercial mines is estimated at 411. In round numbers, the total production for Illinois was made up as follows: Mine-run coal, 11,000,000 tons; lump coal, 21,000,000; egg coal, 2,500,000; nut coal, 2,500,000; pea coal, 9,000,000; slack coal, 1,500,000. The average days of active operation for all the mines was 184. The average value per ton for all grades of coal at the mine was $1.002.

Further figures from the report of David Ross, secretary of the Bureau of Labor Statistics, show that 1105 mining machines were distributed and used in 101 mines; the number of tons undercut by machines was 14,490,454, while 33,308,168 tons were mined by hand. The total number of employees in and about the mines numbered 56,714. The average price paid per gross ton for hand mining from shipping mines was 59.2c.; the average price paid per gross ton for machine mining was 47.9c. The number of men killed inside the mines was 158, while seven men were accidentally killed while working outside the mines. The wives made widows numbered 95, while 273 children were left fatherless. The coke industry in Illinois during 1907 showed considerable activity. The coke production in 1905 amounted to 10,307 short tons, valued at $27,681, while the 1906 production of coke was 268,693 short tons, valued at $1,205,462. The production of coke in 1907 was nearly 300,000 tons and the State now ranks about twelfth among the coke producers.

(By David Ross.) The total output of coal in Illinois for the fiscal year ended June 30, 1907, was 47,798,621 short tons. This is an increase over the preceding year of 9,479,773 tons, or 24.2 per cent. In the report

for the fiscal year ended June 30, 1907, is collected the production of Illinois mines by months. From this it appears that the difference in the production during the first and second periods of the year is only 1.6 per cent. in favor of the last six months. Estimated on this basis the production for the calendar year ending Dec. 31, 1907, will be about 48,562,112 tons. The total number of mines is 933, of which 424 were shipping and 509 were local; the men employed in and about the mines numbered 65,423.

The most important feature in connection with the industry in 1907 is the remarkable increase in the output, 98 per cent. of which was the product of shipping mines operating but 195 days during the year. From this it appears that the mines of Illinois, with present equipment, running approximately full time, could easily produce 75,000,000 tons per annum.

Indiana.—The coalfields of Indiana are included in the eastern interior field which underlies the greater part of western Kentucky, eastern Illinois, and 20 counties in the southwestern part of Indiana, containing about 7000 sq. miles of workable coal land. This area extends from Warren county southward for 150 miles to the Ohio river. Seven or more distinct workable seams have been discovered. These vary from 3 to 11 ft. in thickness and aggregate in a few places from 25 to 28 ft.

The coals of Indiana are of three varieties, which in certain localities merge into one another. From a manufacturing point of view, the "block coal" mined principally in Clay and Vigo counties is regarded the most valuable. It possesses a laminated structure and can be mined in blocks as large as convenient to handle. The expansion of this coal while burning is scarcely perceptible, and it does not cake or run together. It is regarded as a very valuable fuel for the blast furnace and for power plants, and has an excellent reputation for steam and household purposes. The bituminous or coking coals found in Indiana are variable in character, but their average will compare favorably with the coals found in adjacent States.

There is a large area of bituminous coal land in Indiana undeveloped. However, during the last two years thousands of acres have either been purchased, leased or taken under option. With the construction of additional railroads the future development of Indiana coal lands will be extensive. A review of the coal-mining industry for 1907 shows that Indiana coal is not only holding its own in the home market, but is reaching out into other States, in spite of strong competition. In 1906 there were 11,422 tons produced by the mines exclusive of the mines working 10 or less men, an increase of 426,055 tons over 1905. The same mines during 1907 produced 11,692,702 tons, an increase of 250,702 tons. Of this amount about half was consumed in Indiana and half shipped to other States. Prices were better and steadier than for a number of years

past, ranging from 90c. to $3.25 for mine-run. Of the total production about 724,000 tons were block coal, a decrease of about 22,670 tons as compared with 1906. The average price at the mine was about $2.40 in 1907. Steam coal (slack), the lowest-priced product of the mines, could not be produced as rapidly as the demands arose and this scarcity compelled the use of a higher grade by many concerns; this latter condition swelled the profits of the operators to a greater extent than during former years.

Total production, wages paid, number of miners and workmen employed, etc., show only a slight increase over 1906, owing to the following reasons: Continuous car shortage for at least two-thirds of the year; a shortage of miners in some localities and the refusal of miners to work during certain periods since Oct. 20, when currency could not be obtained to meet the wages. The more stringent mining laws, the product of the last Legislature, were rigidly enforced, and, as a consequence, there were fewer accidents resulting in fewer deaths and a less number injured. Of the 10 strikes during the year all were local and none lasted over 15 days. In one instance the miners occasioned a shutdown, which under the agreement, licensed the operators to deduct $1 a day from their wages. for the time they were out. The miners reluctantly returned to work and have sued for the wages deducted by the operators. A test case in this matter will likely reach the Supreme Court.

The number of new mining companies organized and new mines opened during 1907 exceeded the number of mines closed by 16. There was a slight increase in machine mining and the mining properties generally improved. Fewer fatal accidents occured, due perhaps to a more vigilant inspection and a rigid enforcement of the mining laws.

The Indiana operators are inclined to be optimistic relative to the future of the coal mining industry. They feel assured that the new steel city of Gary will soon be consuming hundreds of thousands of tons of Indiana coal, and that the iron ore of Indiana will in time become sufficiently developed to make a large demand upon the mines. They are also appreciative of the work of the Indiana Railroad Commission in the enforcement of a rule conducive to an equitable distribution of coal-carrying cars and equalization of rates.

According to State Geologist Blatchley, if Indiana coal is mined at the same rate as it has been in recent years, by 1910 about 200,000,000 tons will have been taken out. The product of the Indiana mines for the next 20 years will be enormous. Mr. Blatchley further states that only one-fortieth of the total available coal supply in the State has been mined. Working out the coal at this rate year by year, it will be about 800 years before the most available Indiana coal will be worked out and exhausted. According to his forthcoming report, the so-called available coal is about one-fifth of the total coal deposits or supply of the State—the remaining

four-fifths of Indiana coal being unworkable under present conditions, owing to depth of seams or other drawbacks. The State geologist estimates that the volume of coal within the State is 40,000,000,000 tons, about 8,000,000,000 tons of which are available under present methods and conditions. Up to 1899 about 100,000,000 tons had been worked out, and the next 11 years will see 100,000,000 tons more mined.

Iowa (By Jas. H. Lee).—The production of coal in Iowa in 1907 was the maximum on record, in spite of the mild weather prevailing during November and December, which naturally tended to diminish the output. The reports of the mine inspectors show an output for the fiscal year ending June 30, 1907, of 7,568,424 tons, an increase of 550,939 tons over the previous year. A part of this increase may be attributed to the fact that all the mines in Iowa were idle during April, 1906, which had a tendency to decrease the production for that year. Counteracting this condition, however, is the fact that during 1906 most of the important mines were operated a greater number of days than during the preceding year. In view of these conditions, then, it would appear that the increment for 1907 is due largely to the normal, healthy growth of business and domestic demands.

Data gathered by the Iowa Geological Survey show an output for 1906 of 7,266,224 tons and insofar as these figures are comparable with those of the State mine inspectors they show a well defined increase for the calendar year of 1907. The value of the 1906 yield was $11,609,455, an average of $1.60 per ton, and the indications are that this price was maintained during 1907.

The greatest producing county during 1907, and for a number of years preceding, was Monroe, with an output of about 2,500,000 tons. Polk county, located in the center of the lower coal measures belt, was second, producing 1,425,000 tons.

There was not a large amount of development work carried on in 1907. A new field was opened in Boone county, in the northwestern part of the productive coal measures. In addition to this there were a number of new mines developed in the fields already operated, enough to offset the mines which have been worked out and closed down. There are about 325 companies operating in Iowa, and these companies worked about 400 mines. The use of mining machinery is not looked upon with great favor by the miners, and as the agreement between the operators and miners is not specific upon this point the latter are abandoning the use of machinery in many cases. Many seams also do not permit the economical use of machinery; consequently it is not used to a large extent.

The present agreement between operators and miners expired on Apr.1, 1908, and the mines will doubtless be closed for sometime pending the acceptance of a new arrangement for the succeeding biennial period.

This together with the mildness of the winter of 1907-8 may cause a diminution in the output for the current year, although recent experience points toward the opposite being the more likely outcome.

During 1907, the Iowa Geological Survey carried on coöperative topographic work with the United States Geological Survey. Under this plan the work of mapping the coalfields, begun in previous years, and centering about Des Moines, was prosecuted by the mapping of a quadrangle to the south of the area previously surveyed. This work should prove of especial value in a State such as Iowa, where the coal basins are local and limited in extent and so far have not been accurately correlated with one another. By far the greater portion of Iowa's coal output comes from the lower coal measures. A series of maps covering this area will doubtless be of great assistance in the development of the various fields and the correlation of the different seams.

Kentucky.—The coal mining industry in Kentucky during 1907 showed greater activity than in any preceding year. With increased and better railroad facilities, the opening of new fields was greatly facilitated; this general development was also accompanied by extensive improvements at many of the older mines operated by the large companies. One of the most important occurrences was the construction of a branch of the Chesapeake & Ohio Railway, which opened extensive coal beds in Pike and Letcher counties, in eastern Kentucky. In view of the prospective importance of these beds, reconnaissance of the region was made in 1906 by the U. S. Geological Survey. Until 1906, Pike county was accessible only by wagon except when floods on the Big Sandy made it possible for steamers to ascend the river to Pikeville, but the completion of the railroad brought about the opening of several mines. Many coal banks were opened in this field during the Civil War to get fuel for blacksmithing and for engines used in logging, but the abundance of timber at hand has made wood so cheap that there has not been much incentive to mine or use the coal. The active mines, therefore, have been developed by outside capital. The general method of operation is the room-and-pillar system, the coal being shot from the solid. At the beginning of 1907, the various mines in this district had a combined output of about 1000 tons daily. The field contained six coal beds of workable thickness; the fuel is of excellent quality . Recent investigations show that the quantity of coal available for mining aggregates nearly 800,000,000 tons.

Among the companies that were active in new development work in 1907 were the River and Rail Coal and Coke Company, of Uniontown, where the Davidson Mining property was recently purchased and operations looking to the opening of new mines immediately begun. The Big Sandy Company, of Boston, took up additional coal leases and is preparing to open a large coking operation. The Freeburn Coal and Coke Company,

in Pike county, also enlarged its coal mining operations and has a second mine ready to produce. The combined output of both mines of this latter company will be 1000 tons daily.

During the first eight months of 1907, there was a scarcity of labor, which condition, however, improved during the last months of the year, when the business depression became pronounced. The production in the western coalfield of Kentucky, which in 1906 showed an increase of nearly 1,000,000 tons over 1905, showed but little increase in 1907. The extraordinarily large tonnage of 1906 was due to the cessation of mining in Indiana and Illinois in the spring of that year. The activity in coal mining in Illinois and Indiana during 1907 was uninterrupted and this reacted materially upon the production in Western Kentucky.

One interesting fact pertaining to coal mining in Kentucky is that this is the only one of the States whose coal supplies are drawn from any two of the great fields. The coal measures of the Appalachian system underlie the eastern counties of the State, while the southern extension of the eastern interior or Illinois-Indiana field is worked extensively in the western part of the State. Some of the coal in both of these fields is suitable for making coke, although the product of the eastern counties may be more safely classed as a coking coal. There was about 550 coke ovens active during 1907; the total output from the seven plants which furnish the principal supply of coke from this district, amounted to about 19,000 tons.

Maryland.—The coal industry in Maryland during 1907 showed greater activity and a larger output than in any previous year, 5,529,663 short tons being mined and shipped. Since mining first began in this region 160,038,392 tons of coal have been produced. The great increase in the George's Creek region during 1907 was entirely in the West Virginia basin. Of the total amount of coal mined, 3,148,495 tons were shipped over the Baltimore & Ohio railroad; 632,752 tons over the Pennsylvania; 203,527 tons over the Chesapeake & Ohio canal, and 1,899,458 tons over the Western Maryland railroad. Of the entire output, 1,476,104 tons were converted into coke and used locally or included in the surplus.

Of the coal shipped by the Chesapeake & Ohio canal, 202,054 tons were delivered by the Consolidation Coal Company, the heaviest producer in the Georges Creek field; of the remainder of the coal transported by the canal, 1486 tons were shipped to the Georges Creek Coal and Iron Company. The canal shipments were 4022 tons in excess of those of 1906. The Consolidation Coal Company shipped 2,092,016 tons, a decrease of 36,863 tons compared with the previous year; the Black-Sheridan-Wilson Company shipped 737,062 tons, a decrease of 60,640 tons; the Piedmont & Georges Creek Coal Company shipped 405,890 tons, an increase of 60,380

tons; the Georges Creek Coal and Iron Company produced 297,626 tons an increase of more than 33,000 tons. In the West Virginia and Upper Potomac basin 1,884,795 tons were mined by the Davis Coal and Coke Company at its nine mines, which are the largest shippers in that region.

(By William Bullock Clark.[1]) The coal trade was remarkably good during 1907, the tonnage and value of the product mined slightly exceeding that of the previous year. The production for 1907 was the largest in the history of the Maryland field, amounting to 4,937,199 long tons, representing an increase of 84,116 long tons, as compared with that of 1906. The total value of this product was \$6,617,354, which represents a gain of \$142,561. The average price per ton was \$1.34 during 1907, as against \$1.33½ in 1906; the "big vein" coal of the Georges creek valley ranged from \$1.20 to \$1.70 per ton, the average being about \$1.45. There was much development in recent years of the "small seams" of western Maryland, both in the Georges creek basin and at points in Garrett county from which coal had not hitherto been shipped. The seams chiefly developed were the Sewickley, Upper Freeport and Lower Kittanning, all of which contain coal of high grade, although the beds are at times thin and faulty. With the gradual exhaustion of the "big vein" of the Georges creek basin it has come to be recognized that the "small veins" actually contain a larger amount of coal than was ever found in the "big vein." Their development will increase the life of the Maryland coalfields by many years.

Michigan (By A. C. Lane).—The coal production in 1907 was about 2,000,000 tons. The purchase of the Bay county properties by the Consolidated Company, the largest producer, released capital, which is now being used in opening a mine in Cuscoal county. About the only independent operator left in Saginaw and Bay counties is the Consumers Coal Company, a relatively small concern. Drills have been actively at work in Bay county, and in Midland and Gladwin counties, finding some coal. The explorations farther west, near the highest part of the lower peninsula, southeast of Cadillac, have failed. Bed rock was never reached, and the indications are that several hundred feet of glacial deposits cover that part of the coal basin and make development impracticable. There is a little hand-to-mouth mining still continued around Grand Ledge and Jackson, where small areas of coal are opened from time to time. Some difficulties and injunctions have been encountered in mining under the city of Saginaw, and some of the mines there are not far from exhaustion.

The Geological Survey of Michigan has issued an elaborate report on the peat resources of the State, by C. A. Davis. So far, however, the

[1] The statistics in this article were collected by the Maryland Geological Survey with the coöperation of the U. S. Geological Survey.

value of the peat as a fuel seems less demonstrated than its value in paper manufacture, and as an absorbent. The Pilgrim Paper Company at Capac, Mich., seems to be a thoroughly successful plant; peat is also used in Chicago in the preparation of fertilizer from slaughterhouse tankage.

(By Lee Fraser.) The coal production in Michigan for the 12 months ending Nov. 30, 1906, amounted to 1,372,854 short tons, at an average cost per ton for mining of $1.50. An average of 2119 persons were employed daily in this mining at an average daily wage of $2.40; this low scale resulted from the mines lying idle for several months. In the 12 months ending Nov. 30, 1907, the greatest tonnage yet produced in this State was recorded; the total output was 1,898,446 short tons, at an average cost of $1.63 per ton. An average of 2812 persons were employed at an average daily wage of $3.24.

Two new shafts were sunk in Saginaw county, one by the Bliss Coal Company and one by the Consumers Coal Company. The total capacity of both mines is between 1200 and 1500 tons per day. Both these shafts were sunk on pockets of coal lying along the same general northeast line as that established by the Pokagon, Riverside, Barnard, and other fields. There was considerable activity to the east of Saginaw county, in the Thumb, in prospecting for new coal lands. Coal has been mined for some time from a 5-ft. seam in the vicinity of Akron. It was long known to exist in this district, but such of it as was mined could find no market, due to an excess of sulphur which caused the coal to cake in a solid mass upon the grate, from which it could be removed only by a hammer and chisel. Extensive development work was carried on in Bay and Midland counties, and a new shaft is now being sunk upon a reported thick seam in Bay county. New coal beds were discovered in Midland and adjacent counties, and it is safe to assume that there is now located and ready for development in the Michigan field fully 40,000,000 tons of coal.

The standard No. 2, Père Marquette No. 2, Chappell and Forduly mines of the Consolidated Coal Company, were abandoned Apr. 1, all the workable coal having been removed. The Standard mine was a comparatively new operation with a life of only four years. It was abandoned on account of the numerous difficulties encountered in mining. The Père Marquette, Chappell and Forduly mines were pioneers in the coal development of the district. The Link Belt Company, of Chicago, under contract to the Consolidated Coal Company, erected a 100-ton-per-hour washery at Saginaw. The ash and refuse contained in the slack and nut coal in this district averages at least 15 per cent. This is the first washery erected in the State.

Missouri.—The coal production in Missouri during 1907 was 4,350,000 tons, an increase of about 600,000 tons over 1906. The depression in business and the consequent money stringency did not particularly

affect the coal industry, as the mines were worked steadily and prices remained firm. Owing to the rulings made by the Railroad and Warehouse Commission of the State, freight rates were somewhat unsettled. The year passed without any serious labor trouble. One of the most favorable features of the coal industry in Missouri is the fact that there were no explosions in any of the mines during 1907 and but eight fatal accidents.

Montana.—Notwithstanding the lack of cars and scarcity of labor the coal-mining industry in Montana was prosperous in 1907. The greatest activity in new development work occurred in the Bear Creek field. The great difficulties attending coal operations in this district were due to inadequate transportation facilities. An independent line, known as the Yellowstone Park Railroad, has a branch running through the field and connecting with the Northern Pacific at Bridger. The Northern Pacific refused to transfer cars to the Yellowstone line, because of the latter's inability to make exchanges, having no equipment of its own. The matter has been taken up by the Interstate Commerce Commission, and it is hoped that some satisfactory solution of the problem will be found.

The vast coalfields of Montana have hardly been touched. The largest operators in the State are the Northern Pacific Railroad Company, with its mines at Red Lodge, the Amalgamated Copper Company, with its mines at Bear Creek and Belt, and the Great Northern Railroad Company, with its operations at Sandcoulee and Stockett. Several new lines of railroad are being extended to various parts of the State and it is certain that the next few years will show great activity in the Montana coal-mining industry. The most important of the new developments will probably be along the line of the Burlington Railway, which in building its line to the Pacific coast is opening several new and important fields.

Nebraska (By Erwin H. Barbour).—It is unfortunate that a region of such extent as Nebraska should be so destitute of natural fuel. The State is situated just between the Carboniferous coal area of Iowa, and the lignitic, or Cretaceous coalfields of Colorado and Wyoming, so that it simply touches their edges where they begin to rapidly pinch out. In eastern Nebraska, along the Missouri river as far north as Omaha, coal, varying from a few inches to a foot and one-half in thickness is to be found in certain places. However, by the time Beatrice or Lincoln, to the westward, has been reached, this layer is reduced to 6 in. Prior to 1906, intermittent prospecting and coal mining were carried on, chiefly by farmers, in the carboniferous counties in the southeastern corner of the State. The layers worked, though reported as 18 in. in thickness, really averaged about 12 to 15 in., where authentically reported, and are heavily underlaid and overlaid by compact shale. Simultaneously, so called

coal mining, was attempted in northwestern Nebraska, in the Cretaceous coal which is of no greater thickness and of less quality. At various times, Carboniferous coal has been mined at Nebraska City, Rulo, South Fork, and elsewhere in the southeastern counties of the State. At one time a mine at South Fork supplied coal for the neighboring towns, Table Rock, Humboldt, Salem, Dawson, and Seneca.

The most sustained and determined effort to mine coal in this region, was made by C. G. Bullock and a company of Lincoln men; who failed, however, to make their mine at Rulo profitable, and it was abandoned at the end of two years. A somewhat similar attempt to mine coal in the Cretaceous measures of Dixon county failed. Little more than laborers' wages were realized in these undertakings. In February, 1906, a coal bed was exposed and a new mine of considerable local interest was developed about four miles southeast of Peru, on Honey creek, the operation being known as the Honey Creek Coal mine. A good bed of bituminous coal was discovered, the seam varying in thickness from 31 to 35 in., the average of 11 measurements being 31 in. This bed was traced along the bluffs for a mile or so, and the hope is entertained that "the State without a mine" may yet boast of a few hundred acres of workable coal. Starting without means and with but little experience, the promoters of this enterprise have developed an industry which, though small and local, seems of consequence in a State where there are so few mineral resources. During 1906 and 1907 eight miners were employed, and the output was as follows: from February to August, 1906, 75 tons, valued at $262; from September to February, 1907, 325 tons, valued at $1138; total, 400 tons, valued at $1400. The output for the following year cannot be given by months but the total production was 700 tons, worth about $2400.

As a reward for the discovery of a workable bed of coal, the State has for a number of years offered a bounty amounting to $4000 for a bed 26 in. in thickness, and $5000 for a bed 36 in. thick. The discoverers and owners of the Honey Creek coal mine made formal application to the last Nebraska legislature for this award but the claim was not allowed. However, it seems sure to be awarded at some future session, for all of the requirements have been fairly met, and now that the seam has thickened to a full 3 ft., the $5000 bounty may be legitimately claimed.

New Mexico (By J. E. Sheridan).—During the fiscal year ending June 30, 1907, the coal-mining industry of New Mexico made greater progress than in any preceding year, both as to percentage of increase of production, and by the installation of machinery and equipment of the latest and most improved type. With the present equipment the coal mines of New Mexico can easily double the output of last year, if a full quota of miners and ample transportation facilities can be secured. The shortage

of these two factors of production restricted the output of the mines fully 50 per cent. during the last fiscal year, as the demand far exceeded the production of the mines, causing a rise of prices which gave large profit to the operators, but they were hampered by the dearth of miners and lack of transportation facilities, causing the shutting down of some of the principal mines from one to two days per week, which restricted the output fully 15 per cent. In view of the shortage of cars it is doubtful if the production could have been increased even if a full force of miners had been available.

The gross production of the coal mines of New Mexico for the fiscal year ended June 30, 1907, was 2,302,062 tons of 2000 ℔.; amount used in operating the mines, 80,678 tons; leaving the net product shipped from the mines, 2,221,384 tons. The estimated production of coke was 203,437 tons, valued at $3.50 per ton at the ovens. It is safe to assume that the percentage of increase of coal production, shown above, was maintained to the end of the calendar year 1907. The domestic market of California would absorb fully double the production of the Gallup coalfield, but there is always an unfilled demand from that quarter for New Mexico coals, especially the coal of the Gallup field, which maintains a preference for domestic uses both at home and in distant markets.

The outlook for demand for coke is not so favorable for 1908, as the shutting down of many large copper furnaces in the Southwest has lessened the need among those principal consumers. These copper-mining companies had very large stocks of coke on hand when the decrease in demand for copper came, which stocks of coke are now being utilized. But it is doubtful if the decreased demand will be materially felt by the coke producers of New Mexico as the entire production of New Mexico has not been a tithe of the demand upon them for coke, a great deal of the coke for the Southwestern markets having been shipped from Colorado, West Virginia, Pennsylvania, and even from Germany and Australia, via San Francisco. It is probable that the present stagnation in the metal-mining industry may redound to the benefit of the New Mexico coal mines. Many of the men thus thrown out of employment are seeking work in the coal mines of this Territory, and the companies come nearer to having a full complement of miners than at any time in the last four years.

During the last fiscal year there were 2966 men and 93 boys employed in the coal mines, an increase of 676 men and 29 boys over the number employed during the preceding fiscal year. The year was an exceedingly unfortunate one on account of fatalities in the mines, 31 men being killed, a percentage of 1.04 of the men employed, as against a percentage of 0.382 in the preceding year. Heretofore New Mexico has compared favorably with the large coal-mining districts of the world, as to the lives lost in the mines; but during the last fiscal year two extraordinary accidents increased

the ratio of fatalities beyond the usual degree, one, an explosion at the Dutchman mine, whereby 10 men were killed, and the other a mine fire in which three men lost their lives.

The promise of a prosperous year for the coal-mining industry of New Mexico, in 1908, is not keeping pace with the anticipation of those interested. The great depression in all lines of business, and especially in the production of copper has seriously impaired the demand for both coal and coke. While the amount of coke produced in New Mexico would be quickly used at the copper smelting plants of Arizona and Mexico, and would not supply one-fourth of the demand even under present conditions, yet many coke ovens are idle in the Territory. The reason for this is that large contracts were placed by the smelters for coke in Connellsville, Penn, and in Colorado, at a time when the smelting works needed far greater quantities than the New Mexico coke ovens would supply. These contracts have still a long time to run, it is said until July, 1908; and coke is now being shipped from those markets past idle coke ovens in New Mexico. The trans-continental railroads are not using within 25 per cent. of the quantity of coal used in 1907, thus decreasing the demand for coal. Another cause of decreased demand was the exceeding mild winter in the Middle West, Western States and Territories and on the Pacific coast; this lessened the demand for coal for domestic purposes. It is hardly probable that the production of coal and coke in New Mexico during 1908 will equal the production for 1907, unless there is a sudden revival of business.

North Dakota (By A. G. Leonard).—The North Dakota coalfields are situated in the western part of the State and contain approximately 32,000 square miles. The coal is a brown, woody lignite of early Tertiary (Fort Union) age. The tests made by the U. S. Geological Survey showed that the North Dakota lignite has special value as a source of producer-gas for gas engines, as it yielded more and better gas than the other coals tested.

The coal varies in thickness from a few inches to 35 ft. Lignite beds 6, 8, and 10 ft. thick are common; those from 10 to 20 ft. thick are not rare, but seams over 20 ft. thick are seldom found. The individual beds are persistent over considerable areas and several have been traced by frequent outcrops for distances of 15 to 30 miles. One coal bed is known to have had an extent of at least 640 square miles.

Though coal is mined to a greater or less extent throughout the area, practically all of the 20 shipping mines are found in the counties of Burleigh, Morton, Stark, Ward and Williams. Burleigh county leads in point of production, and is followed by Ward, with Stark county third. The Consolidated Coal Company, of Dickinson, has recently acquired

valuable coal land in southern Billings county and will open mines on the new line of the Chicago, Milwaukee & St. Paul Railroad.

Coal has been mined in North Dakota for nearly a quarter of a century, the earliest mining so far as known having been carried on by the Northern Pacific Railroad, which operated mines along its line as early as 1884. The mine at Sims was worked for many years and in 1888 there were two other shipping mines, one at New Salem and another at Dickinson. By 1894 the number had increased to eight, and in 1900 the large Washburn mine at Wilton was opened. Several years prior to this the mines in the vicinity of Kenmare began operations. There is an increasing number of large, well equipped properties in North Dakota. In some of these, electric undercutting machines and electric drills are used to advantage and in several cases compressed air is employed to run the machinery.

Mining is carried on either by drifting in along the seams, or by slopes leading down to them, or less commonly (when the coal is at a considerable depth) by shaft. Since many of the lignite beds outcrop at the surface and lie at no great depth, they are readily worked by running a drift in along the seam, and the majority of the larger mines are of this kind. Another common method is that of stripping off the overlying clay. Where the cover is not over 8 or 10 ft. thick this is the simplest and least expensive way of mining the lignite. At many points throughout the area there are openings of this character where the ranchmen of the vicinity secure their coal. Many of the seams outcrop in cut banks along streams where they can be readily reached only in winter when the water is frozen over.

The roof of the coal beds is commonly clay which affords an insecure cover, requiring careful timbering to prevent it from falling. It is customary, therefore, when the seam is of sufficient thickness, to leave from 6 in. to 1 or 2 ft. of lignite to form the roof of the mine. This makes an excellent top and one which frequently requires but little timbering except along the main entries. The room-and-pillar system is used in the mines throughout the region. Dangerous gases are seldom present and owing, doubtless, in some measure to the general freedom from accident from this cause, the ventilation of the mines has not in some instances received the attention it deserves. An air shaft with a fire kept burning at the bottom of it is the usual method employed to secure air circulation.

By far the greater portion of the North Dakota coalfield is yet practically untouched and some parts of it are but little known. Until the last few years much of the region was thinly settled and there are still extensive districts without railroad facilities. Lignite is in general the only fuel used and the ranchmen and settlers occasionally haul their coal 10 or 15 miles. Usually, however, there is a seam near by from which a supply may be obtained.

The coalfield is traversed by four railroads, viz.: the Great Northern, Northern Pacific, Minneapolis, St. Paul & Sault Ste. Marie (Soo), and the new extension of the Chicago, Milwaukee & St. Paul. The coal from the Ward and Williams county mines is shipped east as far as the Red River valley over the Great Northern and Soo roads. The Washburn mine, in Burleigh county, ships its product to points along the latter road and down into South Dakota, while the mines of Stark and Morton counties use the Northern Pacific for the transportation of their coal to many North Dakota towns.

The retail price paid for the lignite varies from year to year and in different parts of the State. At Dickinson it sold during 1907 for $2, at Bismarck for $2.40 and in the Red River valley towns for $3 per ton.

Ohio.—The progress in coal mining in this State in 1907 was not affected seriously by labor disputes, and the demand for coal was sufficient to keep the mines working more than half the time. The disastrous floods which took place during March interfered for a time with the operation of some of the mines in the Hocking Valley district. The scarcity of cars during 1907 was not pronounced, the railroads being able to meet all demands upon them. The lake trade was the largest ever known and indications point to still larger shipments in the future. The coal output of Ohio was disposed of at satisfactory prices, and the majority of the mining companies had a successful year financially. The increase in tonnage and in the number of men employed, however, was accompanied by a large increase in fatalities. There were 153 fatal accidents in 1907, as compared with 127 in 1906. The number of deaths per thousand employed increased from 2.73 in 1906 to 3.2 in 1907.

Coal was mined in 29 counties, the total output amounting to 32,465,949 tons, a gain of 5,252,454 tons over 1906. The most remarkable gains occurred in Belmont and Jefferson counties; the former produced 6,355,583 tons, while Jefferson county showed an output of 4,648,263 tons. The mines in Meigs county were idle a great portion of the year on account of a strike, caused by a disagreement on the weight scale, which accounts for the large loss in that county. Stockton county, which in former years was the leading coal producing district in the State, showed a loss in tonnage; many of the large operations in this county have been nearly exhausted, and the larger portion of the celebrated Jackson county No. 2 coal is practically worked out.

There were 47,876 persons employed in and around the mines of the State during 1907. The average time worked by the pick miners was 191 days; by the machine miners 201, and for the machine runners 206 days. The total pick tonnage of the State was 6,511,773, or 20.1 per cent. of the entire production. The pick tonnage increased 11,510. The machine tonnage for the year was 25,954,176, a gain of 5,240,944, and was 79.9 per

cent. of the entire production of the State. There were 1404 mining machines in use, an increase of 138. The average tons of run-of-mine by pick miners for the year was 676, and per day 3.5; for the machine runners it was 8759 for the year, and 42.5 tons per day.

Oklahoma (By William Cameron).—The production of coal in 1907, in that part of Oklahoma which was formerly Indian Territory, was approximately 3,450,000 tons. Owing to the increased value of slack, the 232 coke ovens in the district named worked but little and irregularly. The total production for the year was about 20,000 tons.

There are now operating in the Indian Territory some 64 companies and individuals, there being 118 mines in operation, of which 31 are shafts, 85 are slopes and two are drifts. The seams of coal operated in this district are the McAlester, the upper Hartshorne, the lower Hartshorne, the Secor, the upper Witteville, the lower Witteville, and the Panama, besides some seams on which there were scattered small operations which have no particular designation. It is possible that the upper and lower Witteville seams are separate portions of some other beds, probably the upper and lower Hartshorne, but the coal having characteristics of its own, I have given it a separate name.

In addition to these principal beds there are operations on what are known as the Cavanal seam, the Arkansas and the Lehigh seams (which latter is no doubt a continuation of the McAlester bed, but having a character so changed as to show distinct individual characteristics). There has been a considerable increase in the development of what is known as the Henryetta seam (which is supposed to be a continuation of the Kansas bed) in the Creek Nation, and there have also been some smaller operations. This Henryetta coal is coming rapidly into prominence as a desirable fuel for steam purposes especially. There are also some operations on an isolated seam which (not being sure of its connection with any principal bed) I have named the Blocker seam.

The total number of men and boys employed was as follows: over 16, underground, 6093; under 16, underground, 137; over 16, above ground, 1454; under 16, above ground, 26; total 7710.

The production of coal in 1907 was larger than for any previous year in the history of coal mining in the Indian Territory and the prospect is that it will still further increase. The actual capacity of the mines in operation is something like 4½ to 5 million tons, if run full time. The chief obstacles to full development are the scarcity of railroad cars and a scarcity of labor. If there were abundant transportation facilities and if labor were plentiful I have no doubt the maximum figures here given would be reached.

The differences between the operators and the men during 1907 were not of much magnitude and when they arose they were settled by repre-

sentatives of each body in a satisfactory manner, without any serious delay in the operations of the mines.

(By Charles N. Gould.) The Oklahoma coalfield, which is situated in the eastern part of the State in what was originally the Choctaw, Creek and Cherokee Nations, is a part of the Southwestern coal field which includes Oklahoma, Arkansas and Texas. All the coal in Oklahoma is of Pennsylvanian age; of the different deposits, those in what was the Choctaw Nation are the most extensive. Before the Choctaw and Chickasaw Indians took their allotments of land, a treaty was made with the Government whereby the greater part of the land underlaid with coal was segregated. Joseph A. Taff, of the U. S. Geological Survey, was detailed to do this work. Having already spent five years in studying and mapping the coal deposits of the Indian Territory, he was the best informed man in the country on the subject. A year's additional time was spent checking up and verifying his previous work.

In accordance with his report, the total amount of land segregated was 437,743 acres. This land lies in what is now LeFlore, Haskell, Latimer Pittsburg, Atoka, Coal and Carter counties. The main body extends uninterruptedly from the Arkansas line near Fort Smith for a distance of 125 miles southwest as far as Lehigh and Atoka. Mr. Taff estimated the combined thickness of the seams of coal underlying the area at 7 ft. and the average output for the entire region at 7000 tons per acre. This would give a total of approximately 3,000,000,000 tons of coal on the segregated land. However, not all the coal land in the State has been separately apportioned. There is a large amount of coal which has never been set apart, but which has been allotted to individual Indians. The greater part of this unsegregated coal is found in Sequoyah, Hughes, Okfuskee, Okmulgee, McIntosh, Haskell, Pittsburg, Coal, Muskogee, Tulsa, Wagoner, Mayes, Craig, Nowata and Washington counties.

There are four workable beds which outcrop on the surface in the old Cherokee and Creek Nations; the thickest of these seams enters Oklahoma from Kansas just west of the main line of the "Katy" railroad, and passes south a few miles west of this road as far as Vinita. It outcrops near Chelsea, Claremore, and Catoosa, on the "Frisco" railroad, and continues south past Broken Arrow, crosses eastern Okmulgee county, is mined at Morris, Schulter and Henryetta, and continues south past Dustin and Lamar as far as the South Canadian river. The thickness of this seam varies from 2 to 3½ ft. It is the same bed that is mined at Pittsburg and Weir City, Kansas.

Another seam which enters Oklahoma from Kansas is in Nowata county; this bed passes south through Washington and Tulsa counties, and is mined at Collinsville, Dawson and Tulsa. There are half a dozen more seams of varying thickness in different parts of the State. One in

Haskell and Pittsburg counties covers more than 20,000 acres. Another in Coal county is nearly as extensive. Judging from all data at hand, it is safe to assume that the amount of unsegregated coal is at least equal to that which has been separated. On this basis, the amount of coal available in Oklahoma is not far from 6,000,000,000 tons.

More than 100 coal mines, large and small, are now being operated, and the amount of coal mined is over 3,000,000 tons a year. The principal coal mines are situated at or near McAlester, Krebs, Hartshorne, Haileyville, Alderson, Wilburton, Sutter, Panama, McCurtain, Heavener, Lehigh, Coalgate, Edwards, Savanna, Blocker, Henryetta, Schulter, Broken Arrow, Dawson, and Collinsville. The coal is chiefly a high-grade bituminous, and the output of the Oklahoma mines supplies a large part of the southern great plains with fuel.

Pennsylvania.—If the production of coal in Pennsylvania could have been maintained throughout 1907 at the rate recorded during the first half, the increase in production over all other years would have been enormous. As it was, the slackening in industrial activity and the mild winter caused a considerable curtailment in output. However, the year's production reaching the enormous total of 236,038,808 net tons greatly exceeded that of any previous 12 months. Comparing this output with that for 1906, there is shown an increase of 34,366,251 net tons. The production of the State will be more than 50 per cent. of the total production of the United States, and is now greater than the total output of our country as recently as 1897.

The decrease in tonnage of other kinds of freight gave the railroads a better chance to carry their coal to market before severe weather set in. The anthracite production was affected by the general business depression less than the bituminous industry. The anthracite tonnage for October was the largest in history for any one month. The production of anthracite coal for the entire year was 86,279,719 net tons, an increase of about 14,000,000 over 1906. The enormous shipments did not satisfy the demands, and at the end of the year the companies were engaged in a rush to fill orders for the lake and Western inland markets.

Great activity prevailed also among the bituminous interests. Several new mines began production during 1907. One of these, at California, Penn., owned by the Jones & Laughlin Steel Company, and known as Vesta No. 4 mine is said to be the largest bituminous operation in the world. During the first month of the mine's existence, the 1400 miners produced 174,338 tons of coal. The largest output for a single day was 7225 tons. Twenty-eight electric locomotives are used to haul the coal to the surface.

Anthracite operators were busy fighting several mine fires, but as a whole, the year 1907 did not show any considerable loss of life or property

in the anthracite districts. The bituminous mines were not so fortunate; the last month of the year brought two serious disasters, the first occurring Dec. 1 in the Naomi mines of the United Coal Company at Fayette City, Penn. This caused the death of 34 miners and was pronounced by the coroner's jury to be due to an explosion of gas and dust ignited by sparks from the electric wires, or by an open light at some point not definitely discovered. The jury which investigated and returned the verdict condemned the use of electric wires on return air currents, and also advised against the use of open lights in gaseous mines. The second explosion occurred at the Darr mine of the Pittsburg Coal Company near Jacob's Creek, Penn., and resulted in the loss of about 150 miners.

The anthracite companies are leading all other operators in inaugurating new reforms that will tend to place coal mining on a higher and safer basis. Nearly all companies have efficient first-aid corps, and several of the larger companies are carrying on lecture courses and institute meetings which are sure to prove beneficial to the miners by increasing their understanding of the problems and dangers that accompany their work.

Several companies are using steel props to advantage and concrete barns and overcasts are common in many districts. Chemical fire engines designed to go underground are kept constantly ready for emergency use in case of mine fires. The larger companies also have a specially designed railroad car which is equipped with all modern appliances and such apparatus as is necessary for rescue or first-aid parties when entering a mine after an explosion or other serious accident. Emergency cars of this sort are kept constantly ready at some convenient point, and immediately on the occurrence of an accident at any mine, the car is rushed to the property affected.

No great changes in the mine laws were made during 1907 and practically no efforts were made to enforce radical reforms. The employment of children at some of the mines caused considerable discussion. The Supreme Court of Pennsylvania in a recent case against the Pittsburg Coal Mining Company held that the act of June 2, 1891, prohibiting the employment of any person under 15 years of age to oil machinery in a coal mine is a valid exercise of the police power, and that under this act the employer cannot set up as a defense contributory negligence. A boy employed in violation of the statute is not chargeable with having assumed the risk of employment in such occupation, and any employer who violates the act by engaging a boy under the statutory age does so at his own risk, and, if the boy be injured while engaged in the performance of the prohibited duties for which he is employed, his employer will be liable for injuries thus sustained.

Tennessee.—The production of coal in Tennessee in 1907 was slightly greater than in 1906. It would have been considerably increased had it

not been for the mild winter, which made the local demand less than usual. Labor was exceedingly scarce during the first nine months of the year, but as in other States, the trouble from this source disappeared when the business depression became pronounced. Nearly one-half of the output of Tennessee is used for railroad fuel, while 15 or 20 per cent. is consumed locally. At the close of 1907 the iron trade was so slack that coke could not be sold at a profit; this caused many large concerns in Tennessee to market a portion of their output as steam coal, thus causing a considerable drop in prices. Mining machines were used by 20 companies, the total number of machines that worked being 154. The total number of persons employed in and about the coal mines was 10,900. There were 19 coke plants in operation during 1907.

Texas.—The production of coal in Texas in 1907 showed but little increase over 1906. The principal drawback was the scarcity of labor. It was claimed also by the operators that the freight rates were excessive and consequently prevented the shipment of lignite to distant and competing points. The price of oil in Texas was sufficiently high to enable coal to compete as a fuel, except in the most southerly districts.

It is reported that the Beargrass Coal Company, of Hillsboro, which has 3000 acres of lignite lands in Leon county, six miles west of Jewett, will erect a mining plant costing $50,000. It is the purpose of this company to develop the property, and to produce 1000 tons per day at first. The district in which this property lies is said to be underlaid with several seams of lignite varying from 4 to 14 ft. in thickness.

Utah.—During the early part of 1907, the great problem confronting the operators of this State was the scarcity of miners; this trouble was corrected when the smelters and metal-mining companies shut down. Then for the first time in many years, the coal operators had a satisfactory supply of labor. The other serious problem with which Utah operators have always had to contend is the transportation question. Cars have been more plentiful this winter than heretofore and notwithstanding the reduced demand for coal and coke for smelting and metal-mining interests, the season has been successful.

The Utah Fuel Company, controlled by the Denver & Rio Grande Railroad Company, produces nearly all the coal mined in the State and is extending and opening several new properties. At some of the mines of this company all shotfiring underground is done electrically, and many Eastern companies would find it to their advantage to adopt this system so sucessfully used in Utah. The system of sprinkling the mine workings is also admirable and is far in advance of the methods generally used in Pennsylvania and West Virginia.

Utah probably employs a larger percentage of Americans in its coal mines than any other State; at least 50 per cent. of the miners are Ameri-

can born. Of the foreigners, the Austrians are in the majority, while Italians, Finns and Greeks are largely employed. During the last year, a considerable number of Japanese were brought in; results proved that they make successful miners.

Much of the coal mined by the Utah Fuel Company is made into coke for smelting use. A contract for coke was with the Amalgamated Copper Company for its Montana smelting works. When the curtailment in the output of copper was ordered, the demand for coke from this source ceased and a large number of coke ovens became idle. This curtailment in the coke production permitted the coal companies to supply domestic trade with large quantities of coal that otherwise would not have been available, and removed the danger of another coal famine in Utah.

(By Lewis H. Beason.) The year 1907 was the most active in the history of coal mining in Utah. A total of 1,967,621 tons were mined, a gain of 128,402 tons over 1906. Carbon county led in production by an output of 1,815,133 tons; Summit county came next with 73,918 tons; then Sanpete with 4500 tons; Morgan with 425 tons and unclassified mines in Iron and other counties with 70,117 tons. In the annual report of the State inspector of coal mines, J. E. Petit, filed on Jan. 5, the total production of hydrocarbons is given as follows: Gilsonite, 21,462 tons; elaterite, 125 tons; tabbyite, 100 tons; asphalt, 4000 tons; ozokerite, 1562 tons; coal, 1,967,651 tons; coke, 324,692 tons.

Virginia.—The coal production of Virginia in 1907 was about normal. The greatest effort in the way of new development work occurred in Wise county, on the Clinch Valley branch of the Norfolk & Western Railroad. The coke made in this district is the only coke made in Virginia from coal mined in the State. The Lowmoor and Covington coke plants draw their supply of coal from mines in the New River district of West Virginia. The coal for the ovens at Pocahontas in Tazewell county is produced at the Pocahontas mine, whose workings extend into West Virginia, although the openings to the mine and the coke ovens are in Virginia. It is a question whether the coal from these mines should be credited to West Virginia or to Virginia, although the latter practice is followed. Considerable new development work was carried on in the Black Mountain region of Lee county, and it is possible that this district will soon become an important producer.

Activity in the coke industry continued in 1907, the total output exceeding that of the previous year, while the average price per ton was $2.37. There were 21 coke plants with a total of about 5500 ovens in operation in 1907. Recent tests made on some of the coals found near Dante, show that the seams in that district contain a fuel that could be made into a high-grade coke.

Washington.—According to the State Mine Inspector, D. C. Botting, the coal production of Washington in 1907, with the exception of the returns from a few small mines which under the State mining laws are not required to give information as to their output, was 3,713,824 tons, against 3,293,098 tons in 1906. By counties the production in 1907 was as follows: Kittitas, 1,524,363; King, 1,446,602; Pierce, 616,120; Lewis, 100,985; Thurston, 25,752. With the exception of 5000 tons, exported to Mexico from Tacoma, the coal was consumed in the State.

(By R. P. Tarr.) Washington contains the only coal areas of importance in northwestern United States. There are undeveloped and almost unknown deposits in Whatcom, Skagit, Snohomish, Chehalis, Pacific, Lewis, Cowlitz, Clarke, and Skamania counties, and other parts of Kittitas besides those to which operations are now confined. At the present time 30 mines are in operation, about half of which may take ranking importance. The remainder have small outputs and are undergoing every sort of struggle to stagger on. Several of them are reopenings of abandoned properties and owing to former mistakes in mining methods cannot be operated with advantage. Of the 30 producing mines in the State, five are operated by the Northern Pacific Railway Company and produce 44 per cent. of the entire output; eight are operated by the Pacific Coast Steamship Company and produce 37 per cent. of the coal mined. The remaining 17 mines produce but 19 per cent. of the entire tonnage. Of the entire output, 66 per cent. is consumed by railroad and steamship companies. The remaining 34 per cent. is used commercially in Washington and tributary territory having a population of 1,250,000 inhabitants.

West Virginia.—The coal production of West Virginia in 1907 was greater than in any previous year. The railroads extended branch lines into new fields, and a large number of new companies started production. The all-absorbing problem was the cause and prevention of coal-mine explosions. Much discredit has fallen upon the operators and the mining methods employed in West Virginia, because of the apparent inability to reduce the number of accidents. The most notable mine accidents in West Virginia in 1907 occurred at the following places and on the dates given: Jan. 24, Florentz, 11 killed; Jan. 29, Stuart, 91 fatalities; Feb. 4, Elkins, 38 fatalities; May 2, Charleston, 11 fatalities; Dec. 6, Monongah, 345 killed.

One of the most important cases affecting the coal industry in West Virginia during 1907 was the conclusion of the fight between the Loup Creek Colliery Company and the Virginia Railway Company (formerly the Deepwater Railway Company) and the Chesapeake & Ohio Railway Company. The case was submitted to the Interstate Commerce Commission June 18, and was decided November 6. The complainant situated at Page, West Virginia, on the Virginia Railway, nine miles from its

junction with the Chesapeake & Ohio, applied for the establishment of through routes and joint rates, with divisions thereof, for the transportation, in carloads, of coal and coke over the two roads, from Page to the destination on the Cheasapeake & Ohio outside of West Virginia, such rates in no case to exceed those applied by the Chesapeake & Ohio from the junction point of the new road and from other points on the line of the last mentioned carrier in the same rate group. The commission, after a thorough investigation, concluded its report by saying: "We do not believe that the facts of this case justify the exercise of the authority invoked. The complaint will therefore be dismissed."

In the northern part of West Virginia, practically all of the mines are controlled by the Fairmont Coal Company; in the central and southern parts of the State the independent companies still continue to hold their own. In the New River field, the New River Company continued to expand and is by far the most important operator. The New River Smokeless Coal Company passed from the control of Wittenberg interests to the Guggenheims.

A mine which is advertised as the largest bituminous operation in the world, began production in November, and shipped coal from the new town called Dorothy. The company is known as The Big Coal Company of West Virginia, and controls about 65,000 acres of land in Raleigh county. Seven seams of coal underlie the greater part of this territory, and are said to contain a workable thickness of probably 40 ft. Geologists say that the entire tract contains more than 4,000,000,000 tons. The Chesapeake & Ohio Railroad Company extended its Cabin Creek branch for a distance of 15 miles to this property and has completed its extension to the new mining town established by the company. The mine is equipped to produce 500,000 tons a year, but the company expects to open several other properties at an early date and thus increase its output so that the annual production will be about 7,000,000 tons.

Wyoming.—Although 1907 was successful so far as the Wyoming coal industry was concerned, the operators were greatly handicapped by the scarcity of labor and inability of the railroads to furnish cars. The business depression helped the labor problem and somewhat relieved the car situation. Practically all of the coal mines in Wyoming are unionized and the companies were forced to adopt the check-off system, whereby the company is obliged to deduct all union dues, fines, etc., from each miner's pay before any store bills or other settlements are made. The miners at the Union Pacific mine, near Rock Springs were organized for the first time in 20 years. The utter impossibility of securing any kind of efficient labor prevented the company from opposing the union organizers.

In the northern part of the State, the Sheridan Coal Company experienced great trouble in securing a sufficient number of miners, while in

southern Wyoming, the Union Pacific mines had the greatest difficulty in obtaining men. The mines of the Amalgamated Copper Company at Diamondville, on the main line of the Union Pacific Railroad, shipped a large part of their output to the mines and smelters at Butte, Mont.; after the closing down of the mines at Butte, the Diamondville coal was shipped to other points for supplying the domestic fuel demand. The Kemmerer Coal Company and other operators stopped taking large winter orders as early as last August, having sufficient orders booked to keep them busy during the greater part of the winter.

(By H. C. Beeler.) The coal production of Wyoming in 1907, was 6,218,859 tons, valued at $1.75 per ton. During the last two years, the production has increased 20 per cent., and the new ruling as to coal production by transportation companies will undoubtedly result in the opening of new mines throughout the State and a constant increase in the quanity of coal produced. The recent extension of some of the existing lines of railroad into new fields and the progress of new lines tributary to the existing fields, as well as several new lines of railroad across the State projected from new sources, all indicate that the undeveloped coalfields of Wyoming will be actively exploited within the next few years.

Wyoming is divided into two coal districts. District No. 1 covers the southern tier of counties, namely Uinta, Sweetwater, Carbon, Albany and Laramie. Coal is produced in all except Laramie county. The greater production of the field comes from the western portion. The total production of District No. 1 was 4,564,858 tons, and the total number of employees in and about the mines was 4935.

COAL ANALYSES. DISTRICT NO. 1.

	Kemmerer, Uinta Co.	Rock Springs, Sweetwater Co.	Hanna, Carbon Co.	Carbon, Carbon Co.
Water	3.53%	4.11%	8.09%	7.42%
Volatile matter	43.58	40.10	44.52	35.43
Fixed carbon	51.36	53.41	43.84	48.30
Ash	1.53	2.38	3.55	8.85

The coal of the Rock Springs and Uinta county fields is a semi-bituminous, non-coking, steam coal and is in general demand throughout the West for steam and domestic purposes. The eastern portion of this district produces a lignite, running higher in ash and moisture than the coals of the western fields, but is in active demand for its heating and general steaming qualities.

District No. 2 comprises the counties of Converse, Natrona, Fremont, Big Horn, Sheridan, Johnson, Crook and Weston. The greatest produc-

tion was from the mines of Glenrock and vicinity on the Chicago & Northwestern railroad, and at Cambria and Sheridan on the Burlington route. The production of this district in 1907 was 1,653,920 tons.

COAL ANALYSES. DISTRICT No. 2.

	Crosby, Big Horn Co.	Sheridan, Sheridan Co.	Cambria, Weston Co.	Big Muddy, Converse Co.	Lander, Fremont Co.
Water	13.54%	13.05%	5.72%	11.50%	11.40%
Volatile matter	34.36	37.55	40.13	38.40	36.60
Fixed carbon	44.89	44.70	43.65	43.70	47.60
Ash	6.54	4.70	10.50	6.40	4.40

This district embraces nearly all of the unexploited coal seams of the State, which have been recently made available by the extension of the Chicago & Northwestern line from Casper to Lander, Fremont county, and the extension of the Burlington route into the Big Horn Basin at the Crosby mines, near Thermopolis. It is generally considered that these two lines are but a portion of trans-State lines, and their reaching these fields at this time is an event of the greatest importance to the coal industry.

The opening of the Crosby and Kirby mines on the Burlington is the first step in the active development of the extensive coalfields of the Big Horn basin on a commercial scale and a great tonnage may be looked for in this section for many years. The coal from the new mines at Hudson and other mines in the vicinity of Lander has not yet been put on the general market, as these properties are mostly new and in the preliminary stage of development, but as the railroad facilities are extended, these will enter into the active production of commerical coal, as there is a constantly increasing demand both locally and in the tributary sections in Nebraska and the Black Hills.

Coal in Foreign Countries.

Australia.—Great activity characterized the coal industry in Australia in 1907, and the production in all of the fields will show an increase, both in tonnage and value. The accompanying table shows the gross outputs from the West Coast collieries in 1906 and 1907. The output in 1907, as compared with that of 1906, shows an increase of 94,800 tons. In addition to the above output, the production of the small mines in 1907 may be safely estimated at 20,000 tons, making an approximate total of 1,053,233 tons. The average coal mined per employee was about 472 tons. The chief cause of injury to the miners was from falls of roof.

PRODUCTION OF WEST COAST COLLIERIES.
(In tons of 2240 lb.)

Colliery.	1906.	1907.	Colliery.	1906.	1907.
	Tons	Tons		Tons	Tons
Denniston	307,557	313,006	Tyneside	61,547	74,862
Millerton	264,002	297,755	Point Elizabeth	156,872	205,337
Seddonville	36,507	35,117	Puponga	18,230	10,900
Blackball	72,039	90,268			
Brunner	21,676	5,986	Totals	938,432	1,033,233

(By F. S. Mance.) The coal raised in New South Wales in 1907 was 8,657,924 tons valued at £2,922,419, exceeding the output of 1906 by 1,031,562 tons and £585,192 in value. The development of the export trade seems to be proceeding on satisfactory lines, the quantity of coal shipped during 1907 amounting to 5,743,057 tons valued at £2,662,218 or 781,967 tons and £581,618 in value in excess of the 1906 shipments. The expansion is evidenced especially in the shipments to over-sea ports which totaled 3,364,483 tons, valued at £1,676,262; in the last two years this trade has exhibited an increase of 1,713,006 tons and £992,762 in value. The exports to Australasian ports also show a gratifying advance, the quantity of coal despatched during 1907 amounting to 2,379,024 tons, valued at £985,956, which exceeds that of 1906 by 118,934 tons and £107,045 in value. The output from the respective districts is shown in the accompanying table.

The quantity of coal raised in New South Wales up to the end of 1907 is estimated at 138,678,149 tons, valued at £53,279,162; the number of

COAL OUTPUT IN NEW SOUTH WALES.

District.	Output 1907.		Increase over 1906.	
	Tons	Value	Tons	Value
Northern	6,058,580	£2,231,901	722,392	£513,723
Southern	1,835,425	515,786	52,030	20,915
Western	763,919	174,732	257,140	50,554
Total	8,657,924	£2,922,419	1,031,562	£585,192

persons employed in mining for coal during 1907 was 17,080, an increase of 2151 over 1906.

Because of the increased activity in metalliferous mining and the starting of iron smelting at Lithgow, the output of coke was well in excess of that of 1906. The quantity produced in 1907 amounted to 254,609 tons valued at £159,316, an increase of 68,549 tons and £48,709 in value over 1906. The kerosene shale raised during the year amounted to 47,331 tons valued at £32,055 and shows an increase of 14,885 tons and £3585

in value over 1906. The number of persons employed getting shale during 1907 was 276.

Belgium.—The production of coal in Belgium in 1907 was 23,824,499 metric tons, or 254,639 tons in excess of 1906. The details of the production in Belgium for 1906 and 1907 are shown in the accompanying table.

COAL PRODUCTION OF BELGIUM.

District.	1907.	1906.	District.	1907.	1906.
Mons	5,020,413	4,896,240	Namur	896,070	860,740
Centre	3,605,596	3,593,000	Liege	5,823,310	6,014,140
Chaleroi	8,479,110	8,205,740	Totals	23,824,499	23,569,860

The stock of coal at the end of 1907 was estimated at 441,000 tons, against 355,000 at the close of 1906, and 129,000 tons at the end of 1905, whereas it amounted to 1,000,000 tons on Dec. 31 of both 1903 and 1904. The total number of miners employed at the end of 1907 was 142,610, as compared with 139,481 on Dec. 31, 1906; this includes both surface and underground workers. In the Province of Liege, there were 44 collieries in operation, the total number of employees, both underground and on the surface, amounting to 34,976. The average daily pay was 88.8c. per day. The Belgian Government advertises semi-annually for bids for a six-months supply of fuel for the use of the Belgian state railways and marines. The last proposal of this sort called for 520,000 metric tons of fine coal or slack, 130,000 tons of briquets, or small lump coal, 700 tons soft coal and 1,000 tons of hard coal. English dealers obtained a small percentage of the orders. The most engrossing question in connection with the coal mining industry in Belgium is the difficulty of procuring sufficient labor.

Borneo.—This island is developing a coal mining industry of considerable importance. The production at present amounts to about 1000 tons weekly, and consists of a good quality of bituminous coal. Some of the product has already been exported to Hong Kong, China, but it is probable that the future trade will be principally with Manila and Singapore. Extensive coal measures are reported as having been found on the Brunai river, in the northeast province of the island. The river, which is a broad and navigable stream, runs across the coal outcrop. Sea-going vessels are able to anchor near the mouth of the mine. This makes it possible for the coal to be marketed at a low cost. Capital for exploiting the deposits has been furnished principally by Hong Kong investors, who have secured concessions which include 10 seams of bituminous coal of various thicknesses; the work now being carried on is in a seam 18 ft. thick.

Canada.—The coal industry in Canada showed a steady advance throughout 1907. The greatest part of the production comes from Nova

Scotia and British Columbia. Owing to the lack of railway transportation facilities in other parts of the coal-bearing districts of southeast Kootenay, British Columbia, the only operating company at present is the Crow's Nest Pass Coal Company, which is now producing at the rate of more than 1,000,000 tons per year. During the last 10 years this corporation has produced more than 5,000,000 tons of coal. Other companies which have valuable coal lands in this part of British Columbia are as follows: The Imperial Coal and Coke Company has obtained leases and a total of 53,851 acres of coal lands. The preliminary survey of a railroad route to these properties has been completed. The Elk Valley Coal Company has valuable leases on Upper Elk river and is continuing the exploration of its lands. Coal has also been discovered and leases have been granted, on a tract covering some 30,000 acres of land, lying north of Lot 4588, on Upper Elk river.

New regulations for the disposal of coal mining rights in Canada have been promulgated. No more coal lands will be alienated, but leases will be granted for 21 years at an annual rental of $1 per acre. No applicant is allowed to lease more than 2560 acres. The leases will include mining rights only, but lessees may purchase the surface rights at $10 per acre. The contracts are subject to cancellation in case of default. Actual settlers are entitled to buy whatever coal they may require for their own use at the pit's mouth at a price not to exceed $1.75 per ton, and a royalty of 5c. per ton is to be paid on the output in addition to the rent.

One of the most promising of the hitherto unworked coalfields of British Columbia is found in the Telkwa valley, Cassier district. When prospecting for gold and copper, in 1901, William Limin, discovered these valuable deposits of coal. Allowing one million tons for each foot to the square mile, the amount of coal contained in this field is estimated to be 25,100,000,000 tons. The present holders expect to mine 3,000,000 tons a year. The seams dip toward the north and east and are considerably faulted. The Cassier Coal Development Company, composed principally of Toronto and Hamilton men, have a Government lease for 52 square mines and is negotiating with English capitalists to take a large interest. The Grand Trunk Pacific Railway is expected to pass through or near the property, while a charter exists for a road to Kitamaat Arm. This district lies 80 miles distant and has a landlocked harbor connecting with the Pacific Ocean.

The Crow's Nest Pass Coal Company intends to increase its output, from 3500 tons daily, to 7000 tons. One of the newest Canadian coal enterprises is the Great West Coal Company, with headquarters at Port Arthur, Ontario. This corporation acquired 4000 acres of coal-land in Alberta. The company has also acquired the charter of the Great Western

Railway, which extends from Calgary through the coalfields to the international boundary.

Chile.—More coal was mined in Chile in 1907 than in any previous year; however, the output was not sufficient to meet the increased demand. The slight falling off in the coal production during 1906 was due to labor troubles, and the difficulty of removing water from one of the largest mines, which extends far out under the ocean at Lota. Several excellent deposits of coal have been found in the interior, 50 to 60 miles from the coast; these beds will be opened up as soon as transportation facilities are provided; this, however, is not likely to be accomplished within the next two or three years.

The value of the importations of coal in 1906 was about $7,464,531 gold, of which the United States supplied only about $69,722 worth, against England's $5,420,658. It is estimated that Chile has paid over $30,000,000 for foreign coal since 1900, which is a heavy drain on a country of but 3,500,000 people. It has been announced that a 7-ft. seam of good coal has been discovered near Valvidia, at a depth of 100 ft. This is the thickest bed yet discovered, that at Lebu being from 4 to 6 ft., and that at Lota is from 4 to 5 ft. thick. Steps are being taken to develop the mine. Native coal of inferior quality sells at $8@9 per ton.

China (By T. T. Read).—An estimate of the total annual production of coal is 15,000,000 tons, valued at $75,000,000 for which I am indebted to Dr. N. F. Drake of the Department of Geology of the Imperial Pei-Yang University. The probable error is about 25 per cent. A large part of this is the aggregate of many small native producers, but there are also many large producers under foreign control or supervision. The largest of these is the Chinese Engineering and Mining Company, organized nearly 30 years ago. There are three mines at Tongshan and Linsi, about 50 miles northeast of Tientsin, in Chili province. The production of these mines was 1,118,000 tons in 1907. The product of the mines goes to supply the Imperial railways of north China, the coasting and trans-Pacific steamship lines, the neighboring cement, fire-brick and brick industries, and for domestic and industrial purposes in the nearby city of Tientsin. The coal dust is made into coke, of which it furnishes a good grade. The next largest producers are the government mines at Pinghsang in the province of Kiangsi. These supply coal and coke to the government iron works at Hanyang, across the river from Hankow. They are being developed to a capacity of 1,000,000 tons per year. The exact production in 1907 could not be ascertained. The mines owned by the Shantung, Bergbau Gesellschaft, at Fangtze and Poshan near Tsinan-fu in Shantung, are also large producers, the total for 1907 being 180,000 tons. The mines at Lincheng in western Chili are connected with the Peking-Hankow railroad by a branch line 14

miles long, and supply that line with coal. Mining on a small scale is also active in Chinghsing, in Shansi, and in other provinces.

France.—The year 1907 was the best on record for coal mining in France. Strikes were few in number, the output was excellent, and business was satisfactory. The larger companies spent considerable time and money in improving their mines and in ameliorating labor conditions. Considerable friction, however, occurred because of new legislation, which affects the hours of labor. It is said that the bill now being agitated, and whose passage is likely, will bring the actual hours of work down to 7¾, and by 1910 the hours of labor will be 7½.

At the Courrières mine where so many lives were lost in the 1906 disaster, the scarcity of labor has been severely felt. In order to fill up the vacancies left by the catastrophe, the company found it necessary to engage many new men, principally recruited from the fishing community of Brittany. The change from the breezy sea life to the dangerous underground work, was not particulary relished by the new employees, and most of the recruited miners returned to their former vocations.

The indications at the close of 1907 pointed to dissatisfaction and conflict in the coal mining industry of France during 1908. It is almost certain that bills will be drafted dealing with the mining law of 1810, the limitation of concessions, the responsibility of colliery owners and the "emancipation" of the miner. Much agitation has already been started in the mining districts, and it is likely that there will be a return of the strike fever, with the result that the output will be curtailed.

The Mining Society of Lens in the Pas de Calais, is the largest coal mining company in France. It produces over 3,000,000 metric tons per year, or between 9 and 10 per cent. of the entire production of the country. There are 16 working pits, employing more than 13,000 miners. There are 19 miles of main underground railroads and the equipment includes 29 mine fans, 20 air compressors, 39 haulage locomotives and over 350 rock drills. Practically all the coal is mined by hand. The seams pitch at a steep angle, are badly faulted and average about 33 in. in thickness. Horizontal galleries, 6 ft. 6 in. high and about 9 ft. wide and from 55 to 175 ft. apart, are driven along the seam from the main roadway; the upper portion lies in the solid rock and the lower portion in the coal. The coal produced in the Pas de Calais district in 1907 amounted to 17,804,836 metric tons, which is an increase of 2,018,106 tons over 1906. The coal production of the Nord district, the second largest producing field in France, amounted to 6,900,657 metric tons, in 1907 or an increase of 690,805 tons over the production of 1906.

Germany.—The coal production in Germany during 1907 amounted to 205,542,688 metric tons, or an increase of 12,009,429 tons over 1906. The output of coke in 1907 was 21,938,038 tons, an increase of 1,677,456

tons over 1906. During 1907, Germany made 16,414,478 tons of retort coke, as compared with 14,500,851 tons in 1906.

Although it can hardly be said that there was a coal famine in Germany during 1907, it is nevertheless true that there was a scarcity of fuel throughout the year. Many persons are beginning to fear similar disastrous results in the German industry to those experienced during the coal famine of 1901. The cause for the present unsatisfactory state of affairs is laid at the door of the coal syndicate, the selling policy of which is said to be greatly at fault. The intense industrial activity which has prevailed in Germany during the last two years has caused an increased demand and consequently higher prices for capital, labor and raw materials. One fact that is vital to the coal industry in Germany is the insufficiency of available labor, although large numbers of foreign laborers have been imported. The average pay for miners during 1907 was $1.43 per day; other underground workers averaged about 95c. per day, and common laborers 90c. The mine workers are required to pay large sums for the purpose of invalid and accident insurance. In 1905, a member of the Coal Miners' Association of Bochum had to pay per annum about $18.04. To these funds, the mine owner is required by law to contribute as much or more. Recent experiments have been made in Prussia to determine the comparative values of Pennsylvania coal and the German product; the results were flattering to the Pennsylvania coal. All German coals for coking have to be washed, while such is not the case with the American coals tried at the test.

Since 1895, the consumption of coal in Germany has risen from 100,000-000 to over 200,000,000 tons. If the consumption is reckoned per head of population, this would show in five years an increase from 5500 to 7260 lb. per person. The price of coal is fixed by the coal syndicate, and has advanced from $3.88 in 1895 to $4.35 in 1907. The Kingdom of Prussia is a large owner of coal mines and is represented in the syndicate, but has not a controlling vote.

India.—The coal industry in India during 1907 was continued with even greater activity than in 1906. At the end of 1885 the average annual output for the previous decade was only 1,227,197 tons; in the next decade the average had risen to 2,758,640 tons, and in that decade ending with 1905, the average output was 7,626,592 tons. The price of coal in India has advanced within the last two years to more than twice its former price, but the cost of production remains about the same as in 1895, when it was calculated at 64c. per ton. The price of coal in the Calcutta market in 1895 was $1.66@2.33 per ton according to quality. The present price for Bengal coal ranges from $4.82@5.49. The prospects are that Indian coal will go still higher.

The earliest records of coal mining in India come from Bengal, and

it is the vast deposits of this valuable mineral in Bengal which today are responsible for seven-eighths of the entire output of the country. Several coal seams are also worked at Singareni, in the Madras presidency, and also at Margharita in Assam, but in the former case the quality is far inferior to that of the best of the Bengal coal, and in the latter case the product, although of a higher quality, is so crushed by geological changes in the strata as to lose much of its value. The mines in Assam are also distant from the great commercial centers and transportation lines. The records of the oldest company now working extend back to 1837, and it was about that time that capital was brought in to work the seams of coal on a systematic plan.

All Bengal collieries are worked on what is known as the pillar-and-room system. Most of the modern mines are equipped with machinery and at several collieries, electrical underground haulage and pumping have been installed with excellent results. The average output of coal per man per day in India is 0.5 ton, which compares with 2.5 tons in the United Kingdom and five tons in the United States. Great hopes are placed on the introduction or suitable mechanical coal cutting appliances. The best mine so far developed in Assam, has been leased by the Assam Railways and Trading Company, which corporation controls a railroad from Brahmaputra to the mine. This company mined about 280,000 tons of coal in 1907. Coal of good quality has been discovered in the Garo and in the Kashia hills, but it is not likely that these deposits will be worked extensively during the next few years.

The Jheria mine, in Bengal, produced approximately 4,100,000 tons of coal during 1907; this amounts to more than 40 per cent. of the total production for the Bengal district.

Calcutta exports more coal than any other port in India. The total shipments for 1907 amounted to more than 100,000 tons. It is interesting to note that Indian coal is used on nearly all steamers that touch India. In some cases, it is mixed with Welsh coal, but in many instances the Indian coal is used alone and the results have been entirely satisfactory. During 1906 and 1907 there were about 28 new coal companies organized with a nominal capital of approximately $3,000,000. This increased activity is generally attributed to the increase in the price of Indian coal.

Japan.—The coal mining industry of Japan is of ancient origin. There is evidence that coal was mined as early as the seventh or eighth century; during the fifteenth century, many important mines were opened up, including those of Miike and Cakashina in the Island of Kyuslu. In 1880, machinery for use in coal mines was first introduced into the country. In 1889, Japanese coal was first exported in any considerable quantities to foreign countries; the output has more than doubled since that time. Japanese coal is generally bituminous. The greater part of the coal beds

belong to the Tertiary formation and none to the Carboniferous age. The total area of the coalfields worked in Japan is about 300 square miles. In addition there are more than 700 square miles of unproductive and undeveloped coal lands.

About 100,000 persons are employed in the coal mining industry of Japan. The average rate of wages paid per month is $7.29 for men and $3.58 for women. The principal coal mines are situated in the following districts: Chiku-ho coalfields, in the provinces of Chikuzen and Buzen in Kyushiu, accounting for more than half the total output of Japan. In these fields are the Namazuda, Katsuno, Meiji, Akaike, and other mines. The Miike coalfields in the Fukuoka and Kumamoto prefectures in Kyushiu. The Takashima coalfields on three small islands near Nagasaki, which supply the best steamship coal, used by the liners calling at Nagasaki. Nearly all the mines in the island of Kyushiu are reported to have passed into the hands of a trust, and the foreign sale of the ouput to be chiefly in the hands of the Mitsui Bussan Kaisha and the Mitsu Bishi Kaisha. The Hokkaido mines, including the Yubari, Sorachi and Poronai, are controlled by the Hokkaido Tanko Kaisha. Anthracite coal mines are located in the province of Nagato, and supply the Japanese navy.

The total output of coal, in 1907 was 12,890,000 metric tons, as compared with 12,500,000 tons in 1906. The exports of Japanese coal amounted to nearly 3,000,000 tons. The largest shipments were to China. The better grades of coal sell at the mines for $1.64 per ton.

The general method of mining coal in Japan is the pillar-and-room system, the size of the pillars being about 66 ft. square. The cutting of coal is almost entirely done by manual labor. The amount of coal mined per miner per 10 hours is 2.5 tons under normal conditions, and when robbing pillars the output per miner per 10 hours is increased to three tons. Timbering is seldom necessary in the mines as the roof is generally of hard sandstone. Steel and bricks are principally used in building overcasts. As most of the mines are wet, pumping water is an important factor. In the Miike mines, where the daily output is about 4000 tons, there are 80 pumps in use. About 90 per cent. of this number are used in collecting water at the main sump at the foot of the shaft. The daily output of water is about 4000 gal. per ton of coal hoisted, against a 900 ft. head.

Mexico.—There was but small activity in coal mining in Mexico in 1907, although the Mexican Coal and Coke Company, at Esperanzas, Coahuila, and the Sadinas Coal Mining Company, at San Juan, and the San Juan de Sadinas, just north of Esperanzas, continued development work. The great hindrance to the coal mining industry in Mexico is the lack of transportation facilities, as most of the coal deposits lie at points distant from present railway lines. Considerable interest is manifested

in the fact that the large peat bogs near Mexico City are to be developed by a company recently formed in New York. Mexico City, like many other muncipalities, has for years past been depending for its fuel supply upon the neighboring woods, with the result that there has been an unnecessary deforestation of woodland. The company proposes to build a 2500-ton plant, and it is expected that the peculiar qualities of peat as a fuel, its smokelessness, its producing no clinker, with its freedom from any of the sulphur products of combustion joined to the fact that its ash has fertilizing qualities, will all tend to assure a ready sale for the product. The company proposes also in addition to selling fuel, to furnish power. Surveys of the peat bogs not far from the lake of Tezcoco have shown that these deposits have more than 8,000,000 tons available. About $2,000,000 will be invested in the exploitation of the bogs.

Philippine Islands.—The coal deposits of the Philippine Islands received considerable attention during 1907, and recent explorations have shown that the beds are not only of considerable extent, but also that the fuel is of fairly good quality. Considerable development work has been done on the property owned by A. U. Betts, on the Island of Batan, with the result that a coal deposit of much importance has been opened, and arrangements are now being made to install an electric plant for lighting, pumping and hauling. The property is being opened by two slopes, one of which is now in 500 ft. The plant ordered is sufficient for an output of 300 tons daily. The coal produced at this plant has been tested in a number of coasting steamers, and is of good quality, although not equal to the best Australian coal. It is clean, low in ash, and generally free from sulphur. The mine is situated near the water, and two wharves have been built, extending out to a depth of 33 ft. of water; one wharf is 1100 ft. in length and the other 1400 ft. There is a secure harbor, almost completely landlocked, with good holding ground and mud bottom. The harbor is capable of accommodating the largest ship that visits the island.

Russia.—According to recent statistics, the fuel requirements of Russia, reduced to a coal basis, amount to more than 26,000,000 tons per annum, of which about 13 per cent. is imported. During 1905 and 1906, the output of petroleum in the Baku district, owing to strike and other disturbances, was so materially reduced that a fuel famine began during 1906. As affairs in Baku did not quickly improve, users began to look for fuel from other points at less ruinous prices, and at the instigation of the Government, the question of cheaper and more regular transportation of coal from the Donetz and Pula coalfields was made the subject of a conference. The output of coal in the Donetz district is steadily and rapidly increasing, it being 1,750,000 tons more in 1907 than in 1906.

Through her concession from the East Chinese Railway Company, Russia is entitled to carry on mining operations within 66 miles of the

Manchurian railway, so that this gives her access to large coal deposits in Manchuria. The war with Japan cost Russia the rich coal mines at Iantai and Funchan; but the Dschalainor coal mines, which are less than two miles distant from the railway, remained in Russian possession. These mines are being worked for the railway companies, which have also commenced working the coal deposits at Udsiminsk. There are also rich coal deposits in northern Manchuria, within the concession area, but the working of these has not yet been taken in hand.

The output of soft coal in South Russia during 1907 amounted to approximately 15,000,000 short tons; the production of anthracite was nearly 2,000,000 tons. The carrying capacity of the railroad was greatly increased during 1907, and the improved transportation facilities caused a material decrease in the reserve supplies of coal at the mines. In general the mining industry of Russia showed a gradual improvement in 1907, but the manufacture of coke showed but little increase, although this product is greatly needed by the metallurgical plants. The number of employees used to produce coal in Russia during 1907 was 81,260, as compared with 80,063 in 1906.

(By I. I. Rogovin.) According to the statistics of the council of mine operators of Southern Russia, the yield of coal in 1892 was 424,000,000 poods; in 1899, during which the new industrial tax law went into force, it was 853,000,000 poods. Since 1899 the production has increased, except in 1902, reaching in 1904 approximately 1,185,000,000 poods. In 1905, the coal mining industry, as well as all other industries, received a great set-back, the yield falling off by 40 million poods. In 1906 the production was 1,345,000,000 poods, and in 1907 it increased to 1,527,000,000 poods. In other words, the production of coal doubled during the period from 1892 to 1899, and during the next eight years it increased, as compared with 1892, more than 3.5 times, and as compared with 1899, by 80 per cent. Figuring on the population of the country, these figures show that in 1899 there was produced 7.4 poods per man, 9 poods in 1904, and over 11.5 poods in 1907.

In quantity and quality of production the various regions of Russia take the following order, their yield in 1907 being given in millions of poods: (1) Southern Russia (Donetz basin), 1,019.66; (2) Poland (Dombrovski basin), 330.3; (3) Siberia (the district tributary to the Siberian Railroad), 127; (4) Ural, 42.3; (5) Moscow (brown coal), 8.

The yield of coke in 1907 was 105.86 million poods, an increase of 11.62 million poods over 1906. The exportation of bituminous coal by rail during 1907 (figured in millions of poods) was 637.15, an increase of 96.73 over 1906; anthracite coal, 110.11 an increase of 21; coke, 76.19, an increase of 2.59. The following figures show, in millions of poods, the stock of bituminous and anthracite coal on hand on Jan. 1, of 1906, 1907

and 1908: 1906, bituminous 43.62, anthracite 20.32, total 63.94; 1907, bituminous 35, anthracite 18, total 53; 1908, bituminous 31.9, anthracite 10.5, total 42.4.

The number of men employed in the coal and coke industries on Jan. 1, 1908, was 118,000, as compared with 108,457 on Jan. 1, 1907. One of the features of the industry during 1907 was the increase in the exports; there was exported in 1907 a total of 13,152,000 poods, as compared with 5,709,000 in 1906, and 3,252,000 in 1905. The chief markets for export were Turkey, Greece, Italy and Austria. It is to be noted that Russian coal is beginning to supplant British coal in the home market, the Baltic fleet purchasing 5,580,000 poods of Donetz coal instead of the British product. The contract was obtained by the South Russian Coal Syndicate, which was established in 1906, and controls over 50 per cent. of the total yield of the Donetz basin.

The yield of the 29 coal mines of Poland was 330.3 million poods in 1907, against 282.8 in 1906. About 20,649 men were employed during 1907 in the bituminous and anthracite mines, at an average daily wage of 1.35 rubles per man; in the brown coal mines 395 men were employed at a daily average wage of 81 kopecs per man.

During 1907 a modern life-saving station was erected in the Donetz basin. A school for training the workmen in life-saving methods was also opened.

South Africa.—The known deposits of coal in South Africa amount to about 60,000 square miles, although this estimate will undoubtedly be increased when the country has been explored more thoroughly. The seams are generally near the surface and vary from 10 to 20 ft. in thickness. The most important producer of the South African colonies is the Transvaal. During 1907 there were about 25 collieries in active operation. The new coaling appliances at Durban, Natal, were expected to be ready for use about the end of March, 1907, but were delayed in completion. The plant will load into steamers at the rate of 400 tons per hour, either directly from trucks or from storage bins of a capacity of 10,000 tons. The coal will be either dumped into ship-holds from trucks, lifted bodily and dumped sideways, or handled by transporters which carry drop-bottom buckets of six tons capacity. The dumpers have handled loads of nearly 80 tons. Loading will be done by the Harbor Department at a cost of 30c. per ton for bunkers and 20c. per ton for holds. Two steamers can lie alongside and coal at the same time. The storage bins have sliding roofs to prevent deterioration of the coal, and the whole plant will be operated by electricity.

Sweden.—According to official reports for 1907, the coal output in Sweden was hoisted from 15 shafts, of which one was situated in the Kristianstad district, and 13 were in the Malmöhns district. The total

quantity of rock, coal and clay broken in connection with the coal industry amounted to approximately 600,000 tons, from which there was obtained 300,000 tons of coal and 105,000 tons of fireclay. The output in 1907 was about the same as the production in 1906.

United Kingdom.—The quantity of coal mined in the United Kingdom during 1907 amounted to 267,828,276 long tons, of which England furnished 187,383,846; Wales, 40,252,178; Scotland, 40,092,548, and Ireland, 99,704 tons. The report of the Mine Inspector showed that there were 260 coke-works in the United Kingdom; in 52 of these plants by-products were saved. The general tendency was to replace the old bee-hive ovens with different types of retort ovens. The largest producing district was Durham, while Yorkshire, Lancashire and Wales followed in the order named. A little less than 2,000,000 tons of briquets were made, of which 80 per cent. were manufactured in the South Wales district.

The coal industry in Wales during 1907 was highly prosperous. The foreign trade greatly increased, but the coastwise trade decreased; this condition is attributed to high prices. France is the most important purchaser of Welsh coal, buying in the first six months of 1907, over 3,000,000 tons. At the end of the year the value of large coal was nearly 61c. per ton more than at the beginning. In the early part of February, 1907, coal sold for $4.86 per ton. Small coals at the close of the year were about 12c. above January prices. In June, 1907, all Welsh miners working in the bituminous coalfields received an increase of wages of 11¼ per cent. The wages of the miners have been forced to a high level by the pertinacity of the leaders of the men. As soon as one advance in wages was completed, a new request was immediately made for an additional increase, and in every case the demand was granted. At the close of the year the men were still being advised to make further claims, although at present it seems almost impossible, owing to the backward condition of the trade generally, that additional concessions can be made.

The areas of coal supply of the various districts have been materially enlarged by the opening up of new fields and the extension of old workings. Fifeshire was the scene of the greatest new development work, and it is likely that extensions in this district will continue for a considerable time. Many new mines were opened in the Nottingham district. Electric power and compressed air are generally supplanting primitive methods of hauling and mining. The operators are also using ingenious methods of carrying miners to their work, thus saving time and strength they would lose in walking. Mechanical appliances are now used for cleaning mine cars, which soon become caked with fine coal and dirt, and have hitherto been cleaned by hand. The most expeditious device is a circular scraping tube worked by an electric motor, which cleans a car in a

minute. It is estimated that the total capital employed in British coal mines is fully $500,000,000, and that the wages paid annually amount to $300,000,000.

The Coal Markets in 1907.

Anthracite.—No especial feature marked the anthracite market in 1907. There were no changes in the control and management of the trade. Perhaps the chief feature was the decline in the activity manifested in 1906 in the prospecting and purchase of coal lands in the anthracite region by independent companies. The shipments of anthracite were the largest in the history of the trade.

SHIPMENTS OF ANTHRACITE.
(Tons of 2000 lb.)

	1905		1906		1907	
	Tons.	Per ct.	Tons.	Per ct.	Tons.	Per ct.
Reading	12,574,502	20.5	11,258,295	20.2	14,018,795	20.9
Lehigh Valley	10,072,120	16.4	8,536,254	15.4	11,532,255	17.2
N. J. Central	7,983,274	13.0	6,983,217	12.5	8,714,113	13.0
Lackawanna	9,554,046	15.6	9,201,875	16.5	10,237,919	15.2
Del. & Hudson	5,640,628	9.2	5,346,695	9.6	6,562,768	9.8
Pennsylvania	4,890,635	8.0	4,856,004	8.8	6,203,171	9.2
Erie	6,225,622	10.1	5,636,537	10.2	7,151,683	10.7
N. Y. Ont. & Western	2,864,096	4.6	2,444,273	4.4	2,689,089	4.0
Del., Susq. & Schuylkill	1,605,378	2.6	1,435,445	2.4	(a)	....
Total	61,410,291	100.0	55,698,595	100.0	67,109,393	100.0

(a) Shipments included in total of Lehigh Valley Coal Company.

All the companies showed increases, with the largest gains made by the Reading and Erie. The Lehigh Valley figures now include the shipments of Coxe Brothers & Co., or the Delaware, Susquehanna & Schuylkill Railroad, purchased in 1906. The New York, Ontario & Western remained under control of the New York, New Haven & Hartford Company. During 1907 an option on the stock was given to the New York Central, but subsequently was withdrawn. Later in the year the New York, New Haven & Hartford Company announced that it would receive no more all-rail coal for New England points by way of Jersey City and the Harlem river terminal. All shipments must be routed by Reading or Easton and over the Poughkeepsie Bridge line. Whether this will be fully carried out is uncertain. The New Haven's facilities for handling coal were always defective, and the road has been perpetually in quarrels with its connections over car delays and charges.

In the anthracite trade the demand for the small steam sizes was remarkably strong throughout 1907, except for two periods—July and November—and there was a shortage except at these times. The scarcity was brought about partly by the efforts, on the part of the producers to reduce

the production of these small sizes. Prices remained constant until the last part of August when an advance of 25c. per ton was made on all sizes. This increase was maintained to the close of the year in spite of the falling off in trade which began in October. Prepared sizes experienced a variable market. The demand was brisk at times although occassionally the market was flat. Steady contract business took care of a large tonnage and the trade enjoyed a profitable year as a whole. Prices of prepared sizes through the year were on the basis of $5 per ton at tidewater. The customary discount of 50c. per ton was made on Apr. 1, decreasing by 10c. each month until the $5 price was restored on Aug. 1. The year closed with a dull, heavy market, due chiefly to mild weather.

Seaboard Bituminous Trade.—Up to the time of the general business depression the bituminous trade along the Atlantic seaboard was characterized by a tremendous demand except during a portion of the summer months. The demand for coal in the extreme East was especially persistent and consumers clamored for supplies. Early in 1907 shipments by rail were hampered by severe climatic conditions which caused coal to freeze in the cars, creating delays in unloading at tidewater. In March the main line roads increased through rates 5c. per ton, and shortly afterward prices began to advance on all grades. Southern West Virginia coals became scarce and the market was largely supplied from Pennsylvania districts, when the heavy demand began in August.

In October certain George's Creek coals, which earlier had brought $2.85@3.15, were advanced to $3.55@3.70, and even at these prices the demand exceeded the supply. Consumption of coal in New England continued heavy throughout the year and there was not the disposition to carry heavy stocks seen in former years. In November factories began to shut down and the demand fell off sharply, which lowered prices accordingly. Good steam coals, offered at $2.50 per ton New York harbor, proved unattractive and the market closed heavy.

Coastwise Trade.—The demand for vessels, which supply most of the trade in eastern New York and New England points, was active during

COASTWISE COAL SHIPMENTS.
(In long tons.)

	1906.			1907.		
	Anthracite.	Bituminous.	Total.	Anthracite.	Bituminous.	Total.
New York (*a*)	14,150,811	10,572,464	24,723,275	16,753,914	11,691,101	28,445,015
Philadelphia	1,794,773	3,977,909	5,772,682	2,411,521	5,095,473	7,506,994
Baltimore	238,162	3,176,710	3,414,872	266,062	3,804,066	4,070,128
Newport News	....	2,791,404	2,791,404	....	2,396,406	2,396,406
Norfolk	....	2,080,087	2,080,087	....	1,951,747	1,951,747
Total	16,183,746	22,598,574	38,782,320	19,431,497	24,938,793	44,370,290

(*a*) New York includes all the New York harbor shipping ports.

most of 1907. At times the market was severely affected by shortage of vessels, and freight rates, which usually fall off in the summer months, were maintained at winter rates for practically the whole year. The demand from Maine set in earlier than usual and continued until navigation closed, on account of increased activity in manufacturing.

Pittsburg (By S. F. Luty).—The last two months of 1907 spoiled a remarkable year in the coal industry in the Pittsburg district and western Pennsylvania. Production was greater than former years despite the fact that the output of the Pittsburg Coal Company was fully 1,000,000 tons less than in 1906. Great gains were made by the Monongahela River Consolidated Coal and Coke Company as the rivers were navigable every month in the year for the first time in the history of the river trade.

There were no labor troubles of any consequence as the wage scale in force was arranged for two years and did not expire until Apr. 1, 1908. The only interruption due to a strike was at the Youghiogheny mines of the Pittsburg Coal Company, over 2000 miners going out on account of a dispute over the terms of the agreement. A serious strike was threatened owing to the introduction of the Pate dump at the majority of the mines, the objectionable feature being the ratchet sword attachment. A strike was averted by the companies on June 3, discontinuing the use of the dump.

The production of the Pittsburg Coal Company in 1907 was approximately 15,000,000 tons. The other large interest, the Monongahela River Consolidated Coal and Coke Company, produced in round numbers 7,500,000 tons, a gain of 1,500,000 tons compared with 1906. Of this tonnage nearly 3,000,000 tons went to lower river ports. Other river coal interests produced about 3,000,000 tons, two-thirds of which was by the Vesta Coal Company, a subsidiary of the Jones & Laughlin Steel Company, none of which went into the open market but was consumed at the plants of the company.

The lake shipping season opened early in April. The lake freight rates to the northwestern markets were increased 5c. a ton on April 15, making the new rate on cargo coal 88c. and on vessel fuel coal 98c. a ton. While the shipments to the northwestern markets were greatly in excess of the previous year, it is figured that contracts amounting to about 1,000,000 tons remained unfilled at the close of navigation.

Prices were advanced at the opening of the year on all contracts, the increase amounting in most instances to 20 per cent., and prices ranging from \$1.15@1.20 on the basis of mine-run coal f.o.b. mine. The general quotation to the trade was \$1.45 a ton. The advance was due to the higher price for mining. The operators had expected a reduction in the rate of 5c. a ton for mining coal over 1¼-in. screen, and 1906 contracts were made on that basis. Instead the mining rate was advanced to 90c. on April 1,

1906, for a two-year period. A number of important contracts were made in January and a lower rate was named in February. On Oct. 1, the large producers fixed the price on a basis of $1.25 for mine-run coal at the mine. On Oct. 15, the rate was advanced to $1.40, but the depression in the iron and steel industry which resulted in the closing of many mills caused a serious decline in the demand, and on Dec. 2 prices were reduced to a basis of $1.30 for mine-run coal and a week later the rate was cut to $1.15 a ton and even this low price was shaded. Slack which brought 85@90c. at the opening of the year could be had at 40c. in December and some sales were made at a lower figure.

The Pittsburg Coal Company had a prosperous year and in November resumed the payment of dividends. The company in addition to its regular annual contracts booked a number of important new ones, including 250,000 tons for the Southern Pacific Railroad and Steamboat Company to be delivered at its wharves in New Orleans, and 200,000 tons for the St. Louis Gas Company. A sudden storm early in July damaged a number of boats and coal tipples along the Monongahela river, the loss being about $500,000. The March flood also caused a loss, but despite these losses the year was a most profitable one for the Monongahela company.

Connellsville Coke.—The year 1907 opened with prices of coke abnormally high and production to capacity in the Connellsville cokefield; at the close of the year most of the ovens were idle and there was no demand for the product. Early in January furnace coke was quoted at $3.40 to $3.50 and foundry at $4.10 to $4.25, but in December, $2 to $2.25 for furnace and $2.50 to $2.75 for foundry were the quotations, with some sales reported at lower prices. The production of coke in the Connellsville region, including the lower field, in 1907 was about 18,000,000 tons, or nearly 1,000,000 tons less than in 1906. For the first 40 weeks the production was in excess of 400,000 tons weekly, and in the last 12 weeks the average production did not amount to 200,000 tons weekly. On Feb. 18, the H. C. Frick Coke Company ordered an advance in wages for its 18,500 employees in the coke region ranging from 8 to 12½ per cent. effective March 1. This made the rate the highest in the history of the great cokefield and increased the pay roll of the company for the year by $1,500,000. All the other companies in the region adopted the new wage scale. In December three small concerns made a reduction in wages of 12½ per cent. but the other companies announced that no cut was contemplated.

Chicago (By E. Morrison).—In certain respects the market of 1907 was the best that Chicago coal dealers have known. The consumption of steam coals for the first six months was so heavy as to remove all complaint about sacrifices due to demurrage rules. Prices were steady, though not high for Western coals, and there was no serious interruptions

to shipments and sales through labor troubles. Transportation also was better than in previous years, to judge from the absence of general complaints. At the end of the year, however, the trade suffered from financial conditions and the lack of severe weather in December.

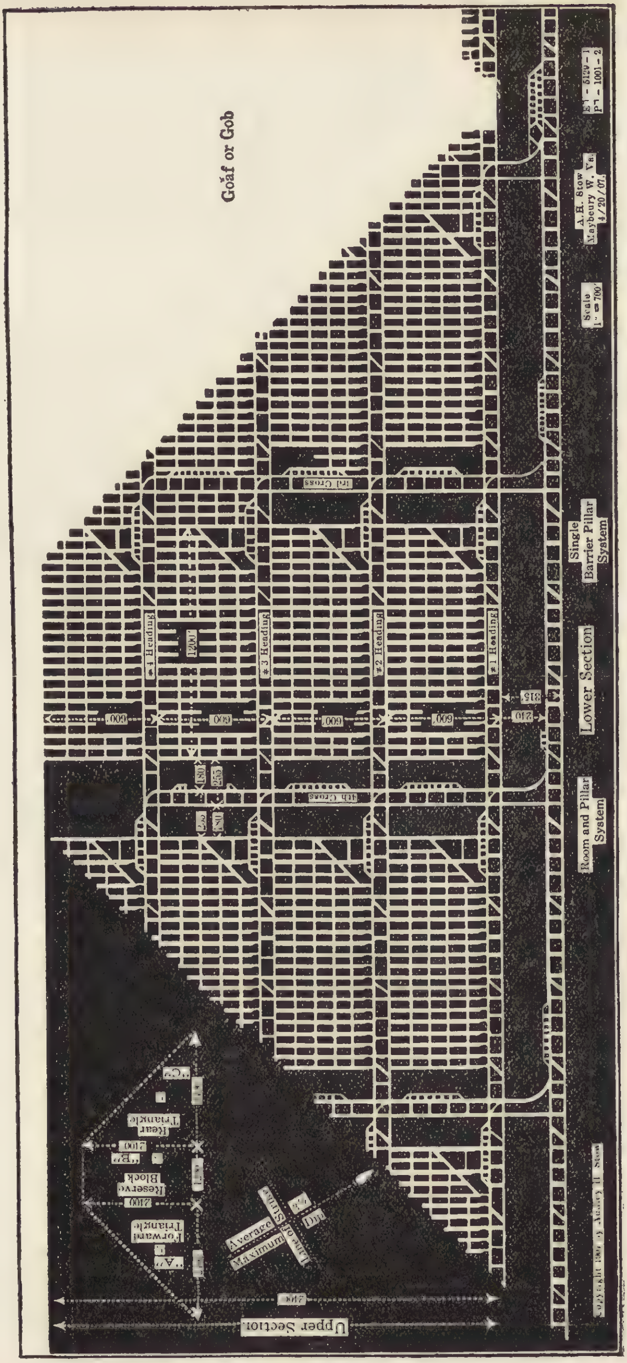
The mild weather of the winter of 1906-1907 caused light sales for domestic purposes in January and February, and by March weakness of prices began. By the end of March this trouble was removed by a general restriction or diversion of shipments, and comparative stability followed from the same causes. A cold and late spring aided sales of domestic coals greatly. The regulation of shipments to Chicago is the great thing needed to insure steadiness. With large producing interests in Illinois and Indiana becoming more conscious of this need every year, and conscious of the fact that common benefit means individual benefit, regulation is more nearly approximated. Enduring regulation, however, can hardly be looked for while the productive area is so great and the number of producers is unrestricted. Even in the summer period, when demurrage troubles are commonly expected, the Chicago market had little weakness, so far as Western coals were concerned. Eastern coals—the term including all from east of Indiana—were more irregular, but hardly suffered seriously from demurrage. Nor were sales of Eastern interfered with by lack of cars so seriously as in previous years.

Average car prices in Chicago in 1907, for Illinois and Indiana coals, which furnish about 60 per cent. of the total handled in the local market, were as shown in the accompanying table.

PRICES OF WESTERN COALS, CHICAGO.
(In long tons.)

Month.	Lump and Egg.	Run of Mine.	Screenings.	Month.	Lump and Egg.	Run of Mine.	Screenings.
January	$2.00@$3.00	$1.75@$2.25	$1.30@$1.65	July	$1.75@$2.65	$1.65@$2.00	$1.25@$1.65
February	1.85@ 2.75	1.75@ 2.25	1.25@ 1.50	August	1.75@ 2.65	1.65@ 2.00	1.25@ 1.50
March	1.75@ 2.75	1.65@ 2.15	1.25@ 1.50	September	2.00@ 2.65	1.65@ 2.00	1.10@ 1.50
April	1.75@ 2.50	1.65@ 2.15	1.25@ 1.75	October	2.25@ 2.65	1.75@ 2.25	1.10@ 1.50
May	1.85@ 2.65	1.65@ 2.25	1.25@ 1.75	November	2.25@ 2.75	1.75@ 2.25	1.10@ 1.40
June	1.85@ 2.65	1.65@ 2.15	1.25@ 1.65	December	2.25@ 2.75	1.75@ 2.25	1.10@ 1.40

The total amount of coal received in Chicago, of all kinds, was about 12,000,000 tons, against 11,000,000 tons in the previous year. The proportions of coal remained about the same, about 65 per cent. bituminous from the Illinois and Indiana fields and the remainder from mines east of Indiana. Within the city much attention was paid to smoke consumption. Leading dealers sought to aid consumers of coal in preventing offensive smoking of chimneys, and so to popularize the less expensive coals, though it is not apparent that such efforts had much effect on the trade as a whole.



PLAN FOR WORKING A COAL SEAM BY THE ROOM-AND-PILLAR SYSTEM, WITH SINGLE BARRIER PILLARS.

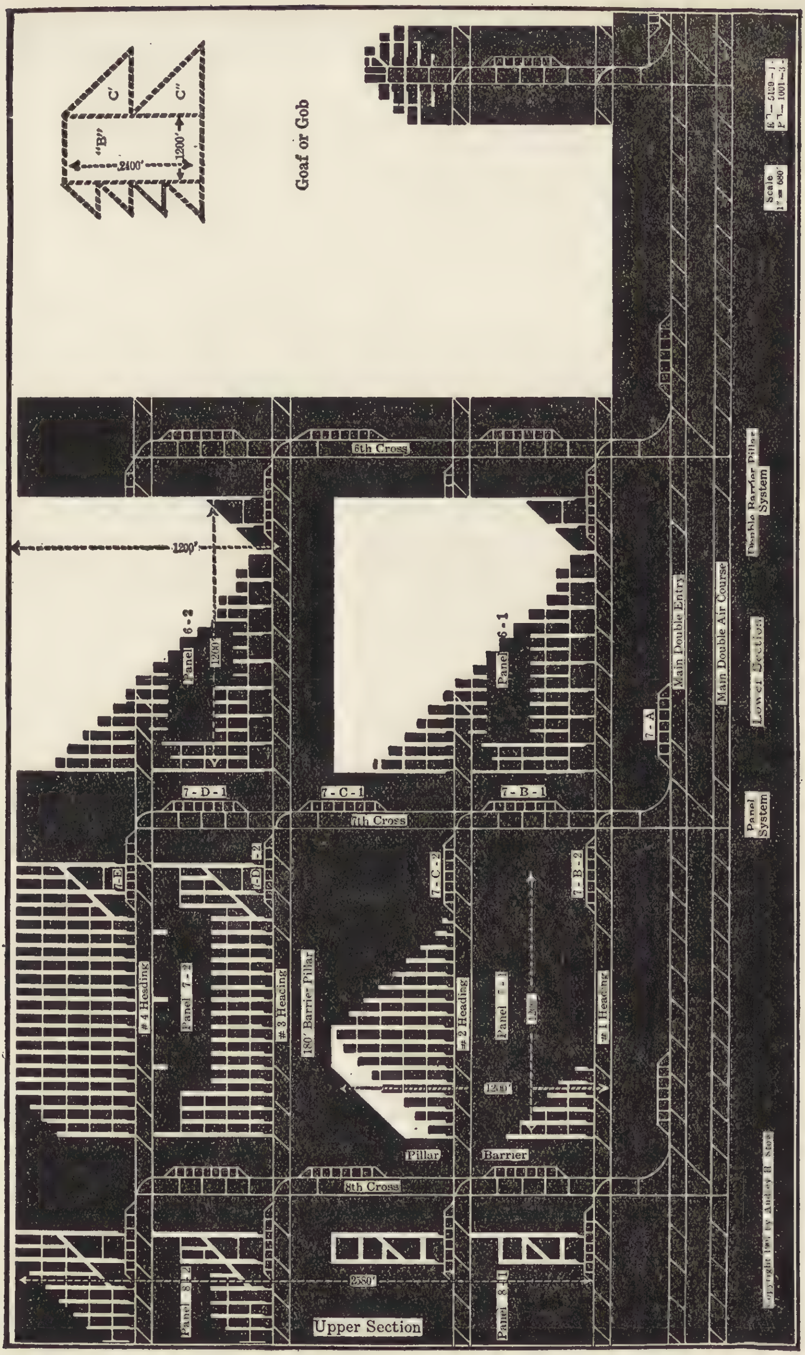
The use of Eastern coals was large, owing to general prosperity and heavy consumption, though in the closing months of the year these coals suffered more than Western. Smokeless ranged from $3 to $3.65 for Pocahontas and New River run-of-mine, the lowest prices being for December and the highest for early summer, though most sales were for $3.25@3.40. Hocking Valley coal sold at $2.75@3.50 for run-of-mine and was in good demand throughout the year. Other Eastern coals sold at about the same prices as in the previous year. Anthracite sold at the same prices as in 1906—$6.50 for egg, stove and chestnut and $6.25 for grate, with No. 2 nut in large demand at $4.50, for delivery after Sept. 1. The graduated discount system was in use between April 1 and Sept. 1, giving 50c. off on April sales, 40c. on May, 30c. June, 20c. July, and 10c. on August sales. There was a scarcity of chestnut throughout the year, but no lack of other anthracite.

Recent Practice in Coal Mining.

Although the attention of coal mine operators in 1907 was directed principally to affording better ventilation and to removing elements of danger, many mining companies found time to improve their systems of underground development by paying more attention to the preservation of room and barrier pillars, at the same time abolishing the uneconomical methods of driving all entries water level in order to prevent the initial expenditure that results from taking up bottom or grading a haulway. Two excellent systems for working low bituminous coal seams are shown in the accompanying illustrations. The general plans were made by Audley H. Stow, and were designed with especial reference to the southern West Virginia coal beds.

The wasteful system of gobbing all bony coal in the anthracite mines continues, and this waste coal or bone, as it is improperly called, is still considered without commercial value, although its composition shows it to be superior, so far as heating efficiency is concerned, to many grades of Western coal now mined. Members of the technologic branch of the U. S. Geological Survey are now investigating the waste in coal mining, and they will, no doubt, give attention to this important matter.

One of the most commendable advances recently made and which will undoubtedly result in much good, is the educational work being done by the Philadelphia & Reading Coal Company, and many other large anthracite corporations. The coal department of the Young Men's Christian Association of Western Pennsylvania is also holding numerous meetings in various localities, and much benefit is resulting from the discussions and papers that are presented. The first-aid organizations which are also being encouraged by the anthracite companies are reaching a degree of



GENERAL PLAN FOR WORKING A COAL SEAM BY THE PANEL SYSTEM, USING DOUBLE BARRIER PILLARS.

efficiency that speaks well for the good they will do when serious accidents occur. Such work appeals to the coal miners themselves, and it is not difficult for any operating company to secure the enthusiastic coöperation of its employees in such humane work.

Mine Timbering.—Among the many mining problems that were taken up in detail during 1907 is that of mine timbering. The reports made by the Forest Service of the U. S. Department of Agriculture were of such a nature that mine operators are becoming alarmed as to the future problem of mine supports. During 1905 the anthracite mines of Pennsylvania consumed about 53,000,000 cu.ft. of round timber and nearly 122,000,000 board ft. of sawed timber, costing approximately $6,000,000. In the same year the bituminous mines of the United States used about $7,000,000 worth of timber at an average cost of $2170 for each bituminous mine, and $20,524 for each anthracite mine.

It is estimated that during 1907 10 per cent. more timber was used than in 1905, and that the cost of this timber was at least 55 per cent. more than in 1905. Although the cost of other mine material has fluctuated, the price of timber has steadily increased. The work of reforesting has progressed so slowly that its results are not yet appreciable. Much publicity was given to the results obtained by the Forestry Service of the Government, in its experiments on mine timbers at several anthracite properties. The preservative methods devised are worthy of the attention of all mine operators.

Steel Supports.—The use of steel supports and props underground is no longer an experiment, for the mines of France, Germany and England have proved that it is eventually most economical, especially where timber is scarce. In those mines of America where steel supports are used, the results were successful beyond expectation. The Carnegie Steel Company is devoting much attention to this problem and has recently had an experienced colliery engineer design several styles of props for use in coal mines. The designs submitted permit of adjustability, and if carried out will make the steel prop almost as flexible as the wooden timbers now used. It is possible that the future will find steel used underground as largely as it now enters into the construction of head-frames, etc., on the surface.

Mine Explosions.—The losses of life in mine explosions during December, 1907, was appalling. Coal mines are dangerous and always will be, but to hush up these matters and belittle the fact is almost a crime on the part of those who are in a position to know. Mine managers, State inspectors, legislators, and engineers, deplore the losses of life resulting from these accidents; but all are waiting for others to provide a remedy. When the probable causes of an explosion are discussed, mining men seem to think it necessary to drop their voices to a whisper for fear someone

will find out something tending to involve the company or perhaps show a weakness in the State mining laws. If an explosion is thought to be due to the derangement of an electric wire and the consequent ignition of dust or gas, many intelligent men will immediately caution great secrecy for fear of what they call hysterical legislation. Legislation concerning mines will be as sane as the evidence submitted to the men who make the laws, and if facts are hidden, the legislators are compelled to do considerable guessing and consequently are not to be blamed for prescribing harmful and useless laws. The majority of mine operators are anxious and willing to see improved conditions introduced that will tend to increase the safety of mining operations in general.

The example set by the management of the Fairmont Coal Company in giving publicity to every detail of the explosion at the Monongah mine, might be followed advantageously by other coal-mining companies. The officers of the company knew that they were complying with the mine laws and felt confident that the truth could not harm them. Out of 100 mines controlled by this company, none was more nearly ideal in its plan of development than the Monongah operations. The ventilating fans were modern in design and threw large volumes of air through the entries and rooms of each mine, providing a system of ventilation second to none in the district. The recent explosions have not taught us new lessons, but have only emphasized with great force, the need of observing the many precautions that experience has already shown to be necessary. Attention has repeatedly been called to the dangers resulting from connecting the underground workings of two adjacent mines, and there is no question as to the advisability of enacting laws to prohibit this practice. If the provision had been considered and enforced at the Monongah mines, the loss of life would have been only half as great. A long list of recent accidents might be cited as examples in corroboration of this statement.

If the general public were to realize how little is known concerning the nature of gas and dust explosions and the phenomena which accompany them, it is most likely that a general demand would be made for some action to clean up the confusion of ideas now prevailing. Some practical men claim that the Monongah No. 8 mine was the seat of the recent explosion because the greatest damage was done in its workings; this very fact is one of the strongest proofs that the origin of the explosion was in the No. 6 mine. All dust explosions gather in force as they travel and do the least damage at the point of initial occurrence.

Evidence gathered from the many explosions in bituminous mines during recent years tends to show that dust has been the main factor in nearly all of these accidents. The trouble may have been started by a local gas explosion, a blow-out shot, the ignition of a cloud of dust by an open

light or from the short-circuiting of a live wire and many other accidental causes. When an explosion occurs in an anthracite mine it is usually local, affecting one portion of the mine; nearly all explosions in our bituminious mines have practically extended through the entire underground workings and generally executed the greatest damage at the very mouth of the mine. Since anthracite dust does not contain much volatile matter while bituminous dust is high in volatile hydrocarbons, which make a dust dangerous, does this not tend to prove that the extension of explosions in soft-coal mines is caused by the dust present?

A better understanding of these matters and the greatest good must come from actual experiments and observation and for this reason I wish to call attention to a part of an interesting paper read before the Midland Institute of Mining Engineers of England by Mr. Neil. The ignition of coal dust in one of the collieries was described as follows: "The seam was dry and dust accumulated rapidly. Early in September a deputy had opened a saftey lamp at a lamp station. A train of full cars drawn by ponies was passing and raised a considerable quantity of dust. Just before the train reached the last station, the deputy removed part of the burning 'snuff' from the wick of the lamp and it fell harmlessly to the floor. As soon as the train had passed he knocked away the remaining portion of the snuff and as this fell to the floor there was an ignition of the coal dust. The flame rose to the hight of about $2\frac{3}{4}$ ft., and spreading over a width of 3 ft., followed the train with a peculiar rolling motion apparently corresponding with the excessive clouds of dust raised, making a peculiar hissing sound. The train was stopped about 45 ft. beyond the last station, and when the flame reached the last car it ascended to the roof, returned along the upper portion of the roadway and finally extinguished itself within 3 ft. of the point of ignition. The color of the flame was described as being similar to that of a candle or oil lamp."

Mr. Neil had thought that nothing short of an ignition or an explosion of gas or gas and dust could cause a coal-dust explosion, and that was his opinion until this incident occurred. Such observations as this are interesting, and when coupled with other phenomena such as the explosion and wrecking of a coal mine when no one is underground and no flame is present, proving almost conclusively that explosions may be due to the ignition of gas by a heavy fall of roof, a better understanding of the dangers ever present may result.

There is no longer any doubt as to the advisability of wetting our coal mines; in fact it is as necessary to remove dust and carefully sprinkle all entries and rooms in our bituminous mines as it is to provide a large circulation of air by an efficient fan. Remove the likelihood of a dust explosion and our mines will be 90 per cent. safer than they are at present.

The Coal Industry of France.

By Ed. Lozé.

Coal Basins.—France contains 18 geographic groups of coal basins in which bituminous coal, anthracite or lignite is mined. The principal groups, six in number, producing bituminous coal and anthracite are as follows: 1. Nord and Pas-de-Calais, a group constituted by the Basin of the Nord and of the Pas-de-Calais, much the most important in area, resources and production, and extending over a part of the departments of the same name, and by the small basin of Boulonnais. 2. Loire, comprising Saint-Etienne and Rive de Gier, Communay, Sainte-Foy-l'Argentizré, le Roannais. 3. Gard (Alaise, Aubenas, le Vigan). 4. Bourgogne et Nivernais (Le Creusot et Blanzy, Epinac, Decize, la Chapelle-sous-Dun, Bert, Sincey). 5. Tarn et Aveyron (Aubin, Carmaux et Albi, Rodez, Saint-Perdoux. 6. Bourbonnais (Commentry et Doyet, Saint-Eloy, l'Aumanee, la Quene).

To these six groups there should be added seven of less importance, which nevertheless produce bituminous coal and anthracite and also in some places, lignite, namely: 1. Auvergne (Brassac, Champagnac et Bourg-Lastic, Langeac). 2. Alpes occidentales (le Drac, Maurienne-Tarentaise et Briançon, Chabbais et Faucigny, Oisans et le Grésivaudan). 3. Hérault (Graissessac). 4. Vosges méridionales (Ronchamp). 5. Creuse et Corréze (Ahun, Bourganeuf, Meymac, Cublac). 6. Ouest (Vouvant et Chantonnay, Le Maine, Basse-Loire, Saint-Pierre-la-Cour). 7. Corse (Osani).

Also five geographic groups producing lignite, namely: 1. Provence (Fuveau, Manosque, la Cadiére). 2. Vosges méridionales (Gouhenans, Norroy). 3. Comtat (Bagnols, Orange, Banc-Rouge, Barjac et Célas, Méthamis). 4. Sud-Ouest (Millau et Trévezel, Le Sarladais, Estavar, la Caunette). 5. Haut-Rhône and others (Hauterives, la Tour-du-Pin, Joigny).

There is some question in regard to recent discoveries made in Lorraine (Meurthe et Moselle) and extending towards the south of the basin of the Pas-de-Calais, and of prospects more or less promising in different places, notably towards the region of Rouen. In Lorraine there appears to be an extension of the German coal basin of Sarrebruck. A certain number of drillings have been made of which several have given positive results. Thus, coal has been found at Abancourt, at a depth of 896 meters; the seam here measures 2.65 meters in thickness; at Dombasle, a bed of 2.50 meters at a depth of 893 meters; at Pont-à-Mousson, a bed of 0.70 meter at 819 meters, one of 1 meter at 1282 meters and several thin seams at about the same depth; at Gezainville, several seams at a

depth of about 1000 meters; at Atton, a series of seams from 0.30 to 0.65 meter in thickness at depths of 1000 to 1300 meters.

The extension of the coal basin of Pas-de-Calais towards the south appears to have been proved over an extent of about 30 kilometers with an area estimated at 6000 hectares. It is premature to estimate the resources of this extension. The content in volatile matter of the product is from 30 to 40 per cent. It appears at the outset that the exploitation will be costly; the depth of the necessary shafts is estimated at 1200 meters. There will be some trouble in harmonizing in this extension, five new concessions having an average extent of 1800 hectares.

Resources.—The question of the coal resources of France is not settled and on this subject we can only refer to several sources of information more or less reliable and to several articles which have appeared in the *Economiste Francais* in 1903 and 1904.[1] Here we give some general statements. On January 1, 1906, there were in France 649 mines of combustible minerals with a total area of 560,759 hectares; in Algeria two with an area of 1981 hectares, making a total of 651 concessions of combustible minerals with a total area of 562,440 hectares. Not all of these are worked.

The greater part of the French basins belong to the upper Carboniferous, yet the principal basin, that of the North and of Pas-de-Calais belongs to the middle, or lower, Carboniferous, and is the most continuous. This basin is an extension of the basin of Westphalia and of Belgium. It is without dispute much the most valuable both for the extent of its resources and for the variety and value of its products. It contains, in fact, nearly the whole series of coals from anthracite to cannel coal, passing through all grades of bituminous coals from low to high volatile contents, which are found in the order stated, in going from north to south. We do not have the entire series of beds and we do not believe rigorous correlations have been established between them for all the basin. We have however previously given a tentative correlation for the Pas-de-Calais basin.[2]

The company d'Anzin, the oldest company, works the group of Vieux-Condé containing 11 beds of a total thickness of 7 meters of anthracite coal; the group of Fresnes-Midi of bituminous coal low in volatile matter; the group of Saint-Louis, Thiers et Alescon of semi-bituminous coals; the group of Saint-Waast of bituminous coking and smithing coals; those of Renard-Sud of bituminous coal rich in volatile matter and those of Renard-Nord and of Luvette of coals highly charged with gas. In the center of the basin the companies of Courriéries and of Lens are working a group of bituminous coals consisting of 14 beds with a total thickness

[1] See in this publication the number for June 6, 1903, pp. 816-817.
[2] See "Mines and Metallurgy at the Exposition of the North of France," by Ed. Lozé, pp. 2 and 3. Paris, Ch. Dunod, Publisher.

of 7 meters; in the northeast a group of semi-bituminous coals comprising about 7 meters of coal in nine seams; in the south a group of highly bituminous coals with 22 beds, having a total thickness of 21 meters.

At Bruay there are three groups, the upper, of four seams, of little thickness; the middle containing three beds together amounting to 2 meters, and the lower containing nine beds with 10 to 11 meters of coal.

In 1873 in a report to the Minister of Public Works, M. de Clerg, Chief Engineer of the mines, admitted the existence in the basins of the Nord and of Pas-de-Calais, of about 60 seams with total thickness of 60 meters, and an average thickness of 20 meters for the whole basin. He estimated the total resources at 20,000,000,000 tons which he reduced on account of irregularities in the beds and loss in mining to 13,000,000,000 tons. In 1889 M. E. Vuillemin, Director of Mines of Aniche, after a study of a report on the exploitation of the mines of Aniche, placed the resources of the basin of the Nord at only 6- or 7,000,000,000 tons. Following the same method, M. L. Aleyrac, Manager of the Courriéres, estimated, in 1899, the resources of the basin of Pas-de-Calais at 4,500,000,000 tons.

The basin of the Loire was for a long time considered the most important of France. Its total thickness is 3000 meters. It contains 30 beds, each more than a meter thick. Their total thickness varies from 50 to 60 meters of coal; but to reach them all it will be necessary in the center of the basin, to go to a depth of 2000 meters. It is doubtful if this basin may be counted on for more than 2,000,000,000 tons.

The basin of Gard is very irregular and cut by numerous faults; a total thickness of 2500 meters is assigned to it. The supply is variable. At Besseges and at Lalle there are more than 20 beds with a thickness of 25 meters; at Rochelle there are also 20 beds with 40 meters of thickness; at the Grand Combe, 12 beds with 18 meters of coal; at other places six beds with 16 meters of coal. The supply is placed at about 1,000,000,000 tons.

At Blanzy (Bourgogne et Nivernais) four principal beds have been recognized; their thickness is 15 to 18 meters, 8 to 10 meters, 25 to 30 meters and 16 to 20 meters. In a general review, necessarily brief, we are unable to go into details.

The estimation of the total coal supply of France is a perplexing question. Yet it appears that the total supply of workable coal cannot exceed 12- to 15,000,000,000 tons, if it even reaches these figures. Although such supplies are not negligible they do not occupy a notable place in the coal supply of the world, and their influence outside of the national domain is rather small. It is not necessary however to regard what precedes as an avowal of poverty in the sources of energy of France. We know, in fact, that if the coal resources of France are limited, and to bespeak impoverishment, it is important to glance at the various existing agencies, notably its watercources, renewed incessantly by the atmospheric agents, and the

continual exchange between sky, earth and ocean, which defy impoverishment and permit us to look into the future without any apprehension. The *dying* nation will without doubt live a long time yet, with a pity for the crocodilian tears of its neighbors. Besides her existence does not depend upon the coal question, because France does not, like some other nations, have coal for the principal source of her power.

Production.—The coal production of France at the time of the Revolution (1789) hardly reached 250,000 metric tons per annum. A century later it had increased a hundred fold. Now it amounts to about 37,000,000 metric tons per year. The steps in this progress are shown by the figures given below, which show the tonnage delivered for consumption, allowing for the variation in stock.

TOTAL COAL PRODUCTION OF FRANCE.
(In metric tons.)

Year.	Metric Tons.	Year.	Metric Tons.	Year.	Metric Tons.
1789..	240,000	1880..	19,507,700	1898..	32,356,000
1802..	844,180	1885..	19,511,000	1899..	32,863,000
1811..	773,694	1890(a	26,083,000	1900..	33,404,000
1820..	1,093,358	1891..	26,025,000	1901..	32,325,000
1830..	1,862,665	1892..	2.,179,000	1902..	29,997,000
1840..	3,003,382	1898..	25,651,000	1903..	34,906,000
1850..	4,433,567	1894..	27,417,000	1904..	34,168,000
1860..	8,309,622	1895..	28,020,000	1905..	35,928,000
1896..	13,509,745	1896..	29,190,000		
1875..	16,956,840	1897..	30,798,000		

(*a*) In 1891 and following years, the output includes the tonnage consumed at the mines.

The output of 1904 consisted of 33,502,000 tons of anthracite and bituminous and 666,000 tons of lignite; the output of 1905 included 35,218,000 tons of anthracite and bituminous and 710,000 tons of lignite.

A detailed examination of the production in each of the six principal geographic groups or basins enumerated above shows that for a score of years this production has been increasing. This increase is important so far as it concerns the Nord and the Pas-de-Calais fields, and especially the Pas-de-Calais. Below is given in round numbers the annual production in metric tons of this group since 1885.

COAL PRODUCTION OF NORD AND PAS-DE-CALAIS.

Year.	Nord, Metric Tons.	Pas-de-Calais, Metric Tons.	Together. Metric Tons.	Year.	Nord, Metric Tons.	Pas-de-Calais, Metric Tons.	Together. Metric Tons.
1885......	3,583,000	6,127,000	9,710,000	1896......	5,202,000	11,871,000	17,073,000
1886......	3,910,000	6,463,000	10,373,000	1897......	5,524,000	12,807,000	18,331,000
1887......	4,198,000	7,119,000	11,317,000	1898......	5,699,000	13,588,000	19,287,000
1888......	4,416,000	7,877,000	12,293,000	1899......	5,660,000	14,201,000	19,861,000
1889......	4,719,000	8,614,000	13.333,000	1900......	5,669,000	14,595,000	20,264,000
1890......	5,135,000	9,076,000	14,211,000	1901......	5,336,000	14,354,000	19,690,000
1891......	4,863,000	8,621,000	13,486,000	1902......	5,077,000	13,185,000	18,262,000
1892......	4,637,000	9,802,000	14,439,000	1903......	5,889,000	16,172,000	22,081,000
1893......	4,707,000	9,180,000	13,887,000	1904......	5,906,000	15,812,000	21,718,000
1894......	4,983,000	10,633,000	15,616,000	1905......	6,189,000	16,985,000	23,174,000
1895......	5,010,000	11,110,000	16,120,000	1906(*a*) ...	6,243,000	15,828,000	22,071,000

(*a*) Provisional figures.

The increase in the other five groups is less than in the Pas-de-Calais and the Nord basins, as shown in the following table; on the whole it appears that in the most of the groups the production tends to become stationary. The numbers are expressed in thousand of metric tons.

COAL PRODUCTION OF FIVE LESSER DISTRICTS.

Year.	Loire.	Gard.	Bourgogne and Nivernais.	Tarn and Aveyron.	Bour-bonnait.
	1000 m.t.	1000 m.t.	1000 m.t.	1000 m.t.	1000 m.t.
1885	3,001	1,728	1,501	1,093	831
1886	2,831	1,739	1,479	977	898
1887	2,980	1,831	1,497	1,076	994
1888	3,185	1,851	1,594	1,147	951
1889	3,378	2,009	1,726	1,234	926
1890	3,587	2,055	1,915	1,453	1,070
1891	3,823	2,192	1,977	1,552	1,119
1892	3,543	2,061	1,946	1,311	1,106
1893	3,507	2,006	1,979	1,420	1,106
1894	3,364	2,060	2,061	1,479	1,098
1895	3,484	9,984	2,074	1,476	1,089
1896	3,578	1,889	2,166	1,551	1,123
1897	3,750	1,894	2,203	1,635	1,128
1898	3,912	1,974	2,341	1,781	1,123
1899	3,858	2,011	2,045	1,844	1,135
1900	4,032	2,045	2,010	1,700	1,094
1901	3,867	2,038	1,556	1,851	1,007
1902	3,106	1,958	1,901	1,589	989
1903	3,689	1,960	1,995	1,881	1,036
1904	3,599	1,833	1,975	1,783	998
1904	3,743	1,983	1,978	1,805	943

It is not worth while, because of their comparative unimportance, to give in detail the production in the other geographic groups. The figures below referring to the six principal groups or basins show an increase in production in the last 20 years. The percentage of this increase was as follows: Nord and Pas-de-Calais, 123; Tarn and Aveyron, 85; Bourgogne and Nivernais, 34; Loire, 32; Gard, 14; Bourbonnais, 5 per cent.

Principal Producing Departments.—In 1905[1] combustible minerals were mined in 41 departments. The Pas-de-Calais field is the most productive, next comes the Nord field. These two departments furnished 64.5 per cent. of the total production of France. If there is added to their production that of the eight departments ranking next in importance we have a total of 34,159,000 tons, which represents 95 per cent. of the

COAL OUTPUT OF TEN LEADING DEPARTMENTS.
(In thousands of Tons.)

Departments.	1904	1905	Changes.	Departments.	1904	1905	Changes.
Pas-de-Calais	15,812	16,985	i. 1,173	Aveyron	1,064	1,082	i. 18
Nord	5,906	6,189	i. 283	Tarn	724	720	d. 4
Loire	3,532	3,678	i. 146	Allier	684	614	d. 70
Gard	1,786	1,936	i. 150	Bouches du Rhône	554	589	i. 35
Saône-et-Loire	1,824	1,798	d. 26	Puy-de-Dôme	535	568	i. 33

[1] It seems better to report for the year 1905, the last for which the figures have been definitely ascertained.

entire product. Above is given the production of these 10 departments during the years 1904 and 1905:

Bituminous coal and anthracite have been produced in 31 departments and lignite in 16. The greater part of the latter product, 83 per cent. in 1905, comes from Bouches du Rhône.

Principal Developments.—Below are given some statistics relating to the principal mining companies in 1904, 1905 and 1906, the figures for 1906 being provisional.

Companies.	1904 1000 Metric Tons.	1905 1000 Metric Tons.	1906 1000 Metric Tons.
In the Pas-de-Calais			
Lens	2,928	3,161	3,030
Courrières	2,265	2,409	1,503
Bruay	2,149	2,326	2,419
Béthune (Bully-Grenay)	1,484	1,644	1,640
Liévin	1,435	1 589	1,504
Marles	1,367	1,413	1,459
Nœux	1,333	1,391	1,425
Dourges	1,014	1,099	989
Drocourt	495	521	469
Ostricourt	424	442	404
Meurchin	376	399	365
In the Nord			
Anzin	2,915	3,052	3,102
Aniche	1,385	1,524	1,552
L'Escarpelle	686	656	693
Douchy	343	357	378
In the Loire			
Roche-la-Moliere et Firminy	807	821	....
Mines of the Loire	744	767	....
Montrambert and the Bésaudière	622	635	....
Saint-Etienne	539	556	....
The three railroad concessions of Paris-Lyon-Mediterranean.	273	281	....
In Saône-et-Loire			
Blanzy	1,430	1,438	....
Epinac	144	153	....
Creusot and Montchanin-Longpendu	126	114	....

Companies.	1904 1000 Metric Tons.	1905 1000 Metric Tons.
In the Gard		
Grand Combe	749	750
Robiac et Meyrannes (Bessèges)	441	469
Rochebelle et Cendras	266	258
In the Allier		
Commentry and Montvic (Commentry-Fourchambault)	315	277
Ferrières, Doyet, L'Ouche-Bézenet, Bézenet and Noyant (Châtillon-Commentry)	248	222
In the Aveyron		
Commentry-Fourchambault	474	487
Aciéries de France	281	313
Campagnac	247	222
In the Tarn		
Carmaux	555	543
Albi	169	1,777
In the Nievre		
Decize	133	146
In the Puy-de-Dôme		
La Roche et la Vernade	239	231
In the Bouches-du-Rhône		
New company (Lignite)	318	336
Valdonare (Lignite)	182	188
Company of the Grand Combe	53	61

State of Exploitation.—The "Statistique de l'Industrie Minerale," published by the Minister of Public Works, reviews the condition of exploitation. In 1904 out of 290 concessions for anthracite and bituminous coal at work, 81 were worked by drifts opening to the surface, and in the same way 20 concessions for lignite. The corresponding figures for 1905 were 280, 90 and 17. In other concessions there were in the same two years 386 shafts for raising coal; 63 were being dug in 1904 and 56 in 1905. The number of shafts used for other purposes was for the same years 312 and 311. The total number of shafts accordingly was 761 for 1904; and 753 for 1905.

The deepest shafts are found in Haute-Saone at the mine of Êboulet (1000 meters); in the Loire at the mines of Plat-de-Gier (880 m.), of

Villeboeuf (663 m.), of Treuil (630 m.), of Comberigol (620 m.), of Méons (600 m.), and of Montrambert (520 m.); in the Gard at the mines of Salles-de-Gagnieres (830 m.); in the Nord at the mines of Anzin (800 m.), of Douchy (684 m.), of Denain (600 m.), of Aniche (595 m.), of Crespin (580 m.), and of Azinodurt (545 m.); in the Pas-de-Calais at the Drocourt (750 m.), at Camblain-Chatelaine (614 m.) , at Lievin (600 m.), at Nœux (550 m.), and at Dourges (525 m.); and in Saone-et-Loire at the mines of Epinac (620 m.).

The average level of the thirls is at no very great depth. The average depth of the seams worked for bituminous coal and anthracite by shafts is 201 meters, and for lignite 47 meters, a general average of 168 meters. For Algeria the average of the seams worked is 32 meters. At Anzin in the Nord the average is 396 meters. For all the mines of the Nord and of the Pas-de-Cailil the average is 376 meters. Thirls, of which the mean depth is very considerable, are found in the mines of Eboulet (880 meters), of Plat-de-Gier (762 m.), of Drocourt (666 m.), of Villebaeuf (613 m.) and of Camberigol (606 m.).

A few beds are worked in the basins belonging to the upper Carboniferous of Stéphanien, 14 in the basin of Saint-Etienne and 17 in that of Alais. The so called basin of Valenciennes (Nord) belongs to the middle Carboniferous or Westphalien; its beds are much more numerous, but of less thickness. In the concession of Aniche there are even 47. The mean thickness of the beds in the basin of Valenciennes does not exceed 0.83 meter, while in that of Saint-Etienne it is 2.81 meters and in that of Alais it is 1.36 meters. At Saint-Eloy the thickness of certain beds reaches 20 meters and at Aubin even 35 meters. For the bituminous coal and anthracite the average thickness of the basin is 2.69 meters, and for the lignite 2.17 meters, which gives a general average of 2.55 meters. For Algeria the average thickness (lignite) is two meters. The Company of Escarpelde in the Nord for several years worked some seams from 0.40 to 0.49 meter thick, but the working of these thin layers has been abandoned for the present.

Average Price.[1]—The value at the surface of the mines of 33,502,394 metric tons of bituminous coal and anthracite, production of 1904, was given in the official statistics at 448,123,643 francs, which gives an average price per metric ton for that year of 13.37 francs. The 665,572 tons of lignite for the same year were valued at 6,305,845 francs which gives a mean price per ton for that product of 9.47 francs. The 105 tons of lignite from Algeria were valued at 1,260 francs or an average of 12 francs per ton. For the year 1905, the value at the surface of the mines of 35,218,237 tons of bituminous coal and anthracite was according to the same statistics,

[1] The values and prices here discussed should not be confused with the prices of sale to the public, and due reserve must be made for the mode of determination of the values and prices. I shall not repeat this remark.

457,519,485 francs, or an average price per ton of 12.99 francs. The 709,467 tons of lignite were valued at 6,532,649 francs, or 9.21 francs per ton. The 85 tons of lignite of Algeria were valued at an average price of 12 francs per ton.

The statistics cited give the average price for the different basins separately. Below are the values and average prices for bituminous coal and anthracite in the six principal geographic groups or basins for the years 1904 and 1905.

VALUE OF OUTPUT FROM PRINCIPAL BASINS.

Geographic Groups or Basins.	Value at the Surface of the Mine.		Average Price.	
	1904 Francs.	1905 Francs.	1904 Francs.	1905 Francs.
Nord and Pas-de-Calais....	280,045,260	289,598,049	12.89	12.50
Loire....................	56,548,524	56,708,806	15.71	15.15
Gard.....................	25,455,423	27,327,708	13.89	13.78
Bourgogne et Nivernais....	26,391,298	26,355,255	13.36	13.32
Tarn et Aveyron..........	25,084,457	23,325,112	14.07	12.92
Bourbonnais..............	13,261,034	12,384,898	13.28	13.14

Personnel.—The employment of women in the underground workings of mines is prohibited in France. So also for children under the age of 13 years. Below is given for the years 1904 and 1905 the number of workers, men, women and children, separating the workers underground from those on the surface, but excluding the workers in conjoined industries.

WORKING FORCE AT FRENCH MINES.

	1904					1905				
	Men	Youths 16 to 18 yr.	Children 13 to 16 yr.	Women	Total	Men	Youths 16 to 18 yr.	Children 13 to 16 yr.	Women	Total
In the mine.	109,000	7,200	6,800	none	123,000	112,300	7,200	7,500	none	127,000
On the surface.......	36,100	2,200	4,200	6,100	48,600	35,800	2,000	3,900	6,400	48,000
All together.	145,100	9,400	11,000	6,100	171,600	148,000	9,200	11,400	6,400	175,100

The figures for the different basins show that the personnel is almost stationary in all the basins except that of the Nord and that of Pas-de-Calais in which the numbers of workers are always on the increase.

Days of Work and Wages.—The Administration of Mines does not guarantee the figures it gives as to the days of work, the wages, and the production per workman. The statistics collected by it are derived from the operators and some of the elements may vary as accounts are differently kept in the different basins. These figures then are not comparable between the basins, but comparison with preceding figures relating

to the same basins is not without interest. The Administration states that the wages given in its statistics are the wages in silver to which are added various allowances. There are differences between the basins justified by varying mining conditions, by the greater or less facility for recruiting the miners, etc., so the wages bear no necessary relation to the average production per workman. The mode of working, the thickness, and purity of the beds worked, the character of roof, the distance from faces worked to the shafts, the mode of conveyance of the coal, are among the causes exercising their influence upon wages.

The total number of days worked, and the total wages for the year 1904 were: 49,559,000 days and 224,255,000 francs; and in 1905, 50,626,000 days and 229,091,000 francs. During 1904 and 1905, which years were identical in respect to the following data, the average number of days worked was 289; the average daily wages were 4.53 francs; and the average annual wages were 1309 francs.

Taken all together the coal basins of France show the following general results in regard to days worked, wages and cost.

AVERAGE DATA PER WORKMAN.

	In the Mines.		On the Surface.	
	1904	1905	1904	1905
Days worked	287	284	293	303
Annual wages, francs	1,412	1,401	1,034	1,064
Daily wages, francs	4.93	4.96	3.53	3.51
Annual output, met. tons	277	283	199 (a)	205 (a)
Daily output, kg	967	997	689 (a)	710 (a)

(a) Average for workmen in mines *and* on surface.

The cost for labor per ton produced was 6.56 francs in 1904, and 6.38 francs in 1905.

Accidents.—The figures given below have reference only to accidents in the coal mines and by injuries must be understood victims incapacitated permanently for labor and those disabled temporarily for more than four days.

NUMBER OF DEATHS AND INJURIES.

	Underground.		On Surface.		Totals.	
	1904	1905	1904	1905	1904	1905
Killed	153	147	31	35	184	182
Injured	20,593	21,768	3,249	3,660	23,842	25,428
Totals	20,646	21,915	3,280	3,695	24,026	25,610

A comparison of the number killed and the whole number of workers at work permits the establishment of the ratio per

10,000. This ratio for 21 years past, and the ratio due to firedamp, is as follows:

NUMBER OF DEATHS PER 10,000 WORKERS.

Years.	Total Ratio.	Due to Firedamp	Years.	Total Ratio.	Due to Firedamp	Years.	Total Ratio.	Due to Firedamp
1885........	16.8	4.1	1892........	9.5	4.9	1899........	13.5	0.1
1886........	13.	2.3	1893........	9.3	4.9	1900........	14.2	0.6
1887........	17.3	8.2	1894........	8.5	4.9	1901........	12.1	0.9
1888........	17.7	5.3	1895........	11.9	0.4	1902........	10.9	0.5
1889........	30.1	20.3	1896........	13.	0.2	1903........	10.2	0.1
1890........	25.8	9.6	1897........	10.7	0.4	1904........	10.7	0.2
1891........	16.7	4.9	1898........	10.7	0.4	1905........	10.4	0.2

Taking as a basis 10,000 workers employed underground and considering the accidents in the underground workings in which there were victims in 1904 and 1905, we have the following figures:

CAUSES OF ACCIDENTS PER 10,000 UNDERGROUND WORKERS.

Causes of Accidents.	Year 1904			Year 1905		
	Accidents.	Killed.	Injured.	Accidents.	Killed.	Injured.
Falling roof..	621.7	5.8	616.7	577.0	6.8	571.4
Firedamp	0.5	0.2	0.3	0.7	0.4	1.1
Accidents at Shafts						
1. Falls..	20.5	1.6	19.8	23.2	1.6	22.8
2. Breaking of cables, falls of cars, etc..........	0.3	1.6	1.9	1.9	1.6	1.9
Explosions..	5.8	1.4	6.2	4.8	0.5	5.4
Transportation in the mines........................	475.9	2.6	474.0	504.9	1.5	507.6
Manual labor..	264.9	2.6	264.9	305.9	0.1	305.9
Other causes..	287.5	0.8	288.7	298.7	0.7	298.5
Total..	1,677.1	12.4	1,672.5	1,721.1	11.6	1,714.6

In France machine mining is very little in vogue. The characteristic of the industry is the careful preparation of the coal extracted. Manufacture of coke and of briquets has reached some development.

Peat.—The production of peat is annually about 100,000 tons; 95,000 in 1904, and 98,500 in 1905, produced by about 2000 workings. The turf pits belong either to the commune or to individual owners. The former produced a little more than half of the product. The Department of Somme is the principal place of exploitation; it furnishes about 30,000 tons per annum. The remainder is furnished in decreasing order by the departments of Loire Inferieure, Aisne, Doubs, Isère, Pas-de-Calais, and Oise. The total value of the production for 1904 was 1,210,000 francs and for 1905, 1,188,000 francs. The price of peat is variable; the average for the two years mentioned was from 12.72 to 12.06 francs per ton. In the department of Oise, there is produced by carbonization of the peat 20 tons of coal, called parisian, of which the average price in 1905 was 150 francs per ton.

Importations.—It is seen from the figures given on page 8 that the production of coal in France has risen from about 20,000,000 tons in 1885 to almost 36,000,000 tons in 1905. There are times of arrest in this progress and times of retrogression, but these are only incidents, explained by various occurences, such as strikes. The consumption is almost parallel but it necessarily exceeds notably the production. This annual excess continued for a long time between 10- and 11,000,000 tons, but has increased since 1899. It reached in 1902, a year short in coal production, almost 15,000,000 tons, but has now returned to an almost normal figure of 13,000,000 tons. This excess, slightly increased by some exports discussed farther on, is naturally supplied by imports.

The imports of coal in 1904 were 13,984,000 tons and of coke, replacing the coke by the corresponding weight of coal calculated on the basis of 135 tons of coal for 100 tons of coke, 14,562,000 tons. The following figures give the information in detail by countries from which imported.

IMPORTS OF COAL INTO FRANCE IN 1904.

	Coal	Coke	Total of Coal & Coke	Total expressed in Coal	Difference from 1903	Proportional part from each country
	Tons	Tons	Tons	Tons	Tons	Per cent.
United Kingdom.....	7,162,000	16,000	7,178,000	7,183,000	*d.* 192,000	49.3
Belgium............	4,241,000	528,000	4,769,000	4,953,000	*d.* 63,000	34.0
Germany............	918,000	1,113,000	2,031,000	2,420,000	*i.* 18,000	16.6
Other Countries.....	6,000	....	6,000	6,000	*d.* 3,000	0.1
Total..........	12,327,000	1,657,000	13,984,000	14,562,000	*d.* 240,000	100.0

The imports in 1905 were 13,436,000 tons of coal, and of coke, represented in coal, 14,007,000 tons. These figures indicate a decline in importation, corresponding to an increase in home production.

The following are the figures in detail of these imports with the countries from which imported.

IMPORTS OF COAL INTO FRANCE IN 1905.

	Coal	Coke	Total of Coal and Coke	Total expressed in Coal	Difference from 1903	Proportional part from each country
	Tons	Tons	Tons	Tons	Tons	Per cent.
United Kingdom......	7,176,000	17,000	7,193,000	7,199,000	+16,000	51.4
Belgium............	3,739,000	501,000	4,240,000	4,415,000	-528,000	31.5
Germany............	877,000	1,115,000	1,992,000	2,382,000	-38,000	17.0
Other Countries......	11,000	....	11,000	11,000	+5,000	0.1
Total...........	11,803,000	1,633,000	13,463,000	14,007,000	-555,000	100.0

For some years it has been feared or hoped, depending upon one's point of view, that the United States, the greatest producer of coal in

the world, would contribute to these imports, and for a couple of years the arrivals from across the Atlantic seemed to be growing. The movement has not been maintained, and the tonnage which the United States contributes to France is in fact insignificant, in 1904, 3000 tons and in 1905, 6000 tons. It appears from the figures above that the United Kingdom of Great Britian and Ireland is the largest exporter of coal to France. There should also be added to the figures of British imports 915,000 tons for 1904, and 907,000 for 1905 loaded in French ports on board of French ships for their own consumption.

The following figures, in round numbers, permit us to follow since 1885 the annual imports into France, expressed in coal, from the United Kingdom, from Belgium and from Germany. These figures show, in the aggregate, the growth of British imports, the slow decrease, with a tendency to increase since 1898 and the giving away again in 1903 and 1904, of the imports from Belgium, and the aggressive German movement in recent years.

FRENCH IMPORTS OF COAL FROM LEADING SOURCES.

Year.	United Kingdom 1000 tons	Belgium 1000 tons	Germany 1000 tons	Year.	United Kingdom 1000 tons	Belgium 1000 tons	Germany 1000 tons
1885	4,079	5,321	1,511	1896	5,089	4,593	1,907
1886	3,921	5,086	1,368	1897	5,491	4,402	2,077
1887	4,046	5,244	1,272	1898	5,486	4,605	1,806
1888	4,108	5,104	1,336	1899	6,720	4,752	1,871
1889	3,840	5,035	1,104	1900	8,375	5,692	2,020
1890	4,865	5,341	1,394	1901	7,900	5,493	1,956
1891	4,863	5,279	1,544	1902	7,528	5,512	2,055
1892	4,897	4,807	1,820	1903	7,375	5,061	2,402
1893	4,434	4,889	2,037	1904	7,183	4,953	2,420
1894	4,843	4,556	2,231	1905	7,199	4,415	2,382
1895	4,944	4,516	2,041				

Exportations.—The French exports of combustible minerals amounted in 1904 to 1,153,000 tons (983,000 tons of bituminous coal and anthracite, 8000 of lignite and 62,000 of coke) and in 1905, not including re-exports, to 1,778,000 tons (1,540,000 tons of bituminous coal and anthracite, 9000 of lignite, and 229,000 of coke). This increase is due to strikes occurring abroad. The exporting basins or regions are the Nord and the Pas-de-Calais, and to a less extent, Loire, Alais, Creuost and Blanzy, Ronchamp, Graissessac, and Aubin, enumerated in decreasing order. These exports are principally to Belgium, and in small amounts to Switzerland. Then come, with small amounts, Algeria and the French colonies, Spain and Germany, and Italy. Some foreign steamships take 40,000 tons per annum. These exports do not include coal used by the French ships, nor that, varying from 50,000 to 80,000 tons, sent to the country of Gex and neutral Savoy (French Zone). The total exports in 1904 were 3.5 per cent. of the product, and in 1905 they were 5.2 per cent.

Consumption.—The French consumption since 1885 is as follows:

CONSUMPTION OF COAL IN FRANCE.

Year.	1000 Metric Tons.	Year.	1000 Metric Tons.	Year.	1000 Metric Tons.
1885...	30,035	1892...	36,516	1899...	45,228
1886...	29,619	1893...	36,379	1900...	48,803
1887...	31,191	1894...	38,000	1901...	46,773
1888...	32,674	1895...	38,640	1902...	44,310
1889...	32,511	1896...	39,995	1903...	48,180
1890...	36,653	1897...	41,341	1904...	47,030
1891...	36,573	1898...	43,295	1905...	48,669

The production bears the following relation to the consumption:

PERCENTAGE OF CONSUMPTION SUPPLIED BY PRODUCTION.

Year.	Per cent.	Year.	Per cent.	Year.	Per cent.
1894	72	1898	75	1902	66.5
1895	72.5	1899	73	1903	72.4
1896	73	1900	68	1904	72.7
1897	75	1901	69	1905	73.8

Consumption in Algeria, including the railroads does not reach altogether 300,000 tons per year.

Distribution of Consumption.—The departments consuming the most coal in France are the following, the figures being those of 1905, and are stated in thousands of tons: Nord, 6841; Meurthe-et-Moselle, 5031; Seine, 4124; Pas-de-Calais, 3558; Loire, 1550; Bouches-du-Rhône, 1481; Seine Inférieure, 1318; Rhône, 1213; Saône-et-Loire, 1098; Total, 26,244 tons. It appears thus that these nine departments in 1905 absorbed 61.3 per cent. of the consumption of France, deduction being made for tonnage required by the railroads. After these come in descending order the departments with a consumption less than 1,000,000 tons but above 500,000 tons, as follows: Seine-et-Oise, Somme et Loire Inférieure, Aisne, Gard, Ardennes, Allier, and Oise-et-Isère. The other departments have a consumption less than 500,000 tons, and the consumption of 20 to 30 of these departments, depending upon the years, does not reach 100,000 tons.

CONSUMPTION OF IMPORTED COAL IN 1905.

Departments	Total 1000 Tons	From United Kingdom	From Belgium	From Germany	Departments	Total 1000 Tons	From United Kingdom	From Belgium	From Germany
Meurthe-et-Moselle.......	2,651	....	882	1,769	Bouches-de-Rhône.........	607	577[1]	....	30
Seine...........	1,174	402	646	126	Seine-et-Oise.....	333	91	240	2
Seine-Inférieure.	1,000	1,000	....	....	Ardennes.........	318	....	316	2
Nord...........	798	2	795	1	Calvados.........	314	276	....	38
Loire-Inférieure.....	794	579	200	15	Aisne............	238	....	232	6
					Gironde..........	218	216	....	2
					Landes..........	202	202	....	....

[1] Not including the 6000 tons imported from the United States of America.

The departments making use of foreign coals number about 70. The principal consumers of these coals in 1905 were as above.

The other departments consumed less than 200,000 tons of foreign coal. British coal is imported into 55 departments and German and Belgian into 28 departments.

Mean Value of Fuels.—The determination of this mean value is a very intricate matter, and without repeating the note on page 17, here are the figures given by the official statistics, first, for the average price of domestic coals at the mine, based on the average price of orders and estimates of the coal used by the mines themselves, and second, the average prices at the place of consumption of foreign and domestic coals since 1885, by the metric ton:

VALUE OF COAL AT MINING AND CONSUMING POINTS.
(In francs.)

Year.	Producing Point.	Consuming Point.	Year.	Producing Point.	Consuming Point.
1885	11.73	20.83	1896	10.84	19.44
1886	11.19	19.79	1897	10.85	18.73
1887	10.63	19.55	1898	11.22	19.46
1888	10.31	19.12	1899	12.41	22.89
1889	10.42	20.38	1900	14.95	26.57
1890	11.94	22.54	1901	15.69	25.59
1891	13.25	21.61	1902	14.55	23.72
1892	12.40	20.38	1903	14.01	22.72
1893	11.49	20.03	1904	13.30	21.83
1894	11.22	19.73	1905	12.92	21.53
1895	11.01	19.66			

The average prices at the mines were given on page 18 for each of the six principal basins in 1904 and 1905. Accepting 21.83 francs and 21.53 francs for the approximate average price of coals in 1904 and 1905, at the places of consumption, the value of the coal consumed in France was in 1904, 1,027,000,000 francs, and in 1905, 1,048,000,000 francs, which include respectively 15,683,350 francs and 15,045,430 francs for duties and fees collected at the frontier upon imported coal.

Consumption at the Mines, Metallurgical Plants and Railroads.—The official statistics give also data in regard to the consumption by mines, by metallurgical works and by railroads. It will suffice to note this as follows: The consumption of collieries is about 10 per cent. of their production; in 1904 it was 3,250,000 tons, and in 1905 it was 3,588,000 tons. That of metallurgical establishments is about double, namely: In 1904, coal 2,892,000 tons, and coke 3,614,000 tons, altogether representing in coal approximately 7,677,000 tons; and in 1905, coal 2,864,000 tons, and coke 3,918,000 tons, altogether approximately 8,153,000 tons of coal, not including wood fuel (15,000 tons in 1904 and 12,000 tons in 1905).

The consumption of coal, briquets, and coke by the railroads, including steam tramways, is shown by the following figures:

CONSUMPTION OF FUEL BY RAILROADS.

	Year 1904.				Year 1905.			
	Coal Tons	Briquets Tons	Coke Tons	Total Consumption, Tons	Coal Tons	Briquets Tons	Coke Tons	Total Consumption, Tons
Locomotives	4,311,000	1,091,000	96,000	5,498,000	4,440,000	1,289,000	96,000	5,825,000
Road and workshop engines	467,000	22,000	29,000	518,000	509,000	34,000	31,000	574,000
Heating of trains, stations, etc.	119,000	28,000	29,000	176,000	115,000	31,000	28,000	174,000
Total	4 897,000	1,141,000	154,000	6,192,000	5,064,000	1,354,000	155,000	6,573,000

This consumption was met in 1905 by domestic and foreign products as follows: Coal, domestic 3,622,000, foreign 1,442,000; briquets, domestic 772,000, foreign 582,000; coke, domestic 151,000, foreign 4000 tons.

In conclusion, the national consumption of coal was employed in 1904 and 1905, by the several industries in the following amounts:

MANNER OF CONSUMPTION OF COAL IN FRANCE.

Industries.	1904 Metric Tons.	1905 Metric Tons.
Metallurgy	8,111,000	8,490,000
Railroads	6,246,000	6,627,000
Mines	3,594,000	3,678,000
To which must be added:		
Gas works (about ⅓ returned for consumption in the form of coke)	3,097,000	3,480,000
Marine	1,477,000	1,133,000
Various Industries	14,105,000	15,261,000
Domestic Use	10,000,000	10,000,000
Total	47,030,000	48,669,000

By-product Coke Ovens.

By C. G. Atwater.

The by-product coke oven plants in existence and under construction in the United States and Canada are given in Table 1, together with their situation, size, and the uses to which the coke and gas are put. This table shows an increase of 596 ovens over the list given last year. The installation of Koppers ovens constitutes the only new establishment, all the others being extensions of existing plants. In addition to the ovens mentioned in the list, there is a plant of 152 Newton-Chambers ovens, built by the Vinton Colliery Company, at Vintondale, Penn. These ovens are not of the retort type, strictly speaking, as they are

of the bee-hive shape and are heated by combustion in the coking chamber, but they recover certain amounts of gas, tar and ammonia. This plant was in operation for a part of 1907. The increase of 596 ovens, or including the Newton-Chambers ovens, of 748 ovens, is an indication of the growing appreciation of the by-product oven on the part of iron and steel manufacturers.

TABLE 1. BY-PRODUCT COKE OVENS BUILT OR BUILDING IN THE UNITED STATES AND CANADA. 1907.

Company.	Situation.	No. of Ovens.	Use of Coke.	Use of Gas.
Otto-Hoffmann and United-Otto Ovens.				
Cambria Steel Co.	Johnstown, Pa.	372	Blast furnace	Fuel
Pittsburgh Gas and Coke Co.	Glassport, Pa.	120	Blast furnace and domestic	Illum. and fuel.
New England Gas and Coke Co.	Everett, Mass.	400	Domestic and locomotive	Illuminating.
Dominion Iron and Steel Co.	Sydney, N. S.	500	Blast furnace	Fuel.
Hamilton Otto Coke Co.	Hamilton, Ohio	50	Foundry and domestic	Illuminating.
Hamilton Otto Coke Co. (*a*)	Hamilton, Ohio	50	Blast furnace	Illuminating.
Lackawanna Steel Co.	Buffalo, N. Y.	188	Blast furnace	Fuel.
Lackawanna Steel Co. (*a*)	Buffalo, N. Y.	376	Blast furnace	Fuel.
Lackawanna Steel Co.	Lebanon, Pa.	232	Blast furnace	Fuel.
South Jersey Gas, Elec. & Traction Co.	Camden, N. J.	150	Foundry and domestic	Illuminating.
Maryland Steel Co.	Sparrows Pt. Md.	200	Blast furnace	Illuminating.
Michigan Alkali Co.	Wyandotte, Mich.	30	Lime kilns	Fuel.
Sharon Coke Co.	So. Sharon, Pa.	212	Blast furnace	Fuel.
Zenith Furnace Co.	Duluth, Minn.	50	Blast furnace	Illuminating.
Semet-Solvay Ovens.				
Solvay Process Co.	Syracuse, N. Y.	40	Lime kilns	Fuel & power.
Semet-Solvay Co.	Dunbar, Pa.	110	Blast furnace	Fuel.
Carnegie Steel Co. (*b*)	Sharon, Pa.	25	Blast furnace	Fuel.
National Tube Co.	Benwood, W. Va.	120	Blast furnace	Fuel.
Semet-Solvay Co.	Ensley, Ala.	240	Blast furnace	Fuel.
Peoples Heat and Light Co. (*b*)	Halifax, N. S.	10	Domestic	Illuminating.
Solvay Process Co.	Delray, Mich.	120	Lime kilns	Illum. & fuel.
Philadelphia Suburban Gas Co.	Chester, Pa.	40	Blast furnace	Illuminating
Empire Coke Co.	Geneva, N. Y.	30	Foundry and domestic	Illuminating.
Central Iron and Coal Co.	Tuscaloosa, Ala.	40	Blast furnace	Fuel.
Pennsylvania Steel Co.	Lebanon, Pa.	90	Blast furnace	Fuel.
Pennsylvania Steel Co.	Steelton, Pa.	120	Blast furnace	Fuel.
Milwaukee Coke and Gas Co.	Milwaukee, Wis.	160	Blast furnace and foundry	Illuminating.
By-products Coke Corporation	So. Chicago, Ill.	160	Blast furnace, foundry and dom.	Illum. & fuel.
Rothberg Ovens.				
Lackawanna Steel Co.	Buffalo, N. Y.	282	Blast furnace	Fuel.
Retort Coke Oven Co.	Cleveland, Ohio	105	Blast furnace	Fuel.
Koppers Ovens.				
Illinois Steel Co. (*a*)	Joliet. Ill.	280	Blast furnace	Fuel & power.
Total		4,902		

(*a*) Not completed. (*b*) Not operated in 1907.

The Western trend of the iron business, toward the Lake ores and the Western iron markets and away from the coke supply of the Connellsville region, and the practical withdrawal of the standard Connellsville coke from the market, due to the approaching exhaustion of the Connellsville coal, have made it necessary (or economical in many cases) to resort to coking coals that do not make as good coke in the bee-hive oven as in the by-product oven. The ability of the by-product oven to treat inferior coals successfully, the increased coke yield, and the saving through by-product recovery and mechanical operation, are now quite generally recognized.

Indeed, while genuine Connellsville coke may still be regarded as the standard in quality, it has long ceased to hold the dominating position as a blast furnace fuel in point of quantity. The iron business has grown far beyond the capacity of this once pre-eminent region. Even the United States Steel Corporation, to whose furnaces the output of the Connellsville district proper is almost exclusively shipped, has definitely undertaken the construction of by-product coke ovens at the Joliet plant of the Illinois Steel Company, which it controls, and a large plant of by-product ovens of a type not as yet announced will be installed at the new works building at Gary, Ind. The by-product oven plant at Joliet is the first installation of this character undertaken by the Steel Corporation on its own initiative, though it has controlled the plants at Sharon, South Sharon and Benwood since its inception, these plants having been built by the individual companies before the consolidation. This step on the part of the Steel Corporation has long been thought inevitable by many who have closely watched the progress of the iron industry, and it does not now seem unreasonable to expect that it will be followed by other steps in the same direction, both on the part of the corporation itself and of the independent concerns.

There is another point of view from which the introduction of the Koppers oven in this country is of interest. The oven, as it is being built here, is of the regenerative type, and this is indisputably the choice of the Steel Corporation committee in charge of the selection, since the same designer began by building a waste-heat or non-regenerative oven, and still builds them when desired. There is therefore no question of their having selected the man rather than the system. It is much more reasonable to infer that the strong tendency to revert to the regenerative type of oven that has lately manifested itself in Germany and elsewhere, because of the increasing confidence in the large gas engine and the enhanced value of coke oven gas for use in it, is making itself clearly felt here as well. Regenerative ovens have, indeed, been in the majority on this side of the Atlantic since the early days of the by-product oven industry. Table 1 shows that there are 3210 regenerative ovens (the United-Otto, Otto-Hoffman and Koppers types) as against 1792 of the non-regenerative Semet-Solvay and Rothberg types. This preference for the regenerative oven has hitherto been largely because of its adaptation to the production of illuminating gas, a peculiarly American development. If, then, the large gas engine for power development is to find a field here as in Europe, it would seem that, with both these influences operating in the same direction, the regenerative oven should continue to grow in favor.

The present season of comparative inactivity in the iron and steel trade may well be regarded as favoring the construction of by-product

oven plants. Economy in cost of production has become of more importance than mere volume of output. Relief from the tension of keeping production up to its maximum point brings opportunity for the consideration of hitherto neglected economies, the benefit of which is brought into clear relief by a background of falling prices. These in their turn mean reduced cost of new construction. Such times have been taken advantage of in the past by manufacturers whose success has passed into a by-word, and it is not likely that their example will lack followers in this instance.

The demand for coke as a fuel for other than blast furnace purposes shows steady progress. By-product coke for foundry use is regularly quoted at higher prices than 72-hour bee-hive coke in several middle-Western cities, and seems to give the satisfaction requisite to maintain this advanced price. Crushed and sized domestic coke is giving universal satisfaction as a substitute for anthracite coal in the Eastern cities where it has been introduced, while in those of the middle West where soft coal is still allowed, the results of the anti-smoke agitation are bringing coke more and more into general use.

By-Products.—The demand for ammonia during 1907 does not seem to have suffered any diminution because of the constant increase in supply. In addition to the home production the country has constantly imported sulphate of ammonia, over 30,000 tons in 1907. The strength of the demand for this product is in its use in agriculture, and the experience of other countries conclusively shows that a much wider extension of this field may be looked for. The demand for tar and tar products has not, however, kept pace with the production, and tar is still used to a considerable extent for fuel purposes. For steam raising 5 ℔. of tar are equivalent to 8 ℔. of coal, allowing for the lower cost of handling the tar, or tar at 2c. per gal. is equivalent to coal at $2.50 per ton. Therefore even at this low value, tar recovery is profitable. The present depression in the building trades, in which large amounts of tar roofing and pitch are usually consumed, has checked progress in this direction, but the increasing use of pitch for fuel briquetting purposes, an industry now apparently well grounded here, and of tar for surfacing macadam roads and for sidewalks may be counted upon to reduce the surplus very materially.

Illuminating gas from by-product coke ovens has steadily grown in general favor. The prescribed standards of quality have been maintained, and 50 of the ovens under construction, those being added to the Hamilton plant, will deliver illuminating gas when in operation. When we consider that it was not much more than 10 years ago that an eminent gas engineer announced his conclusion that a surplus of gas from by-product coke ovens was non-existent, though there were then plants

in operation that could demonstrate the contrary, we are able to appreciate the progress that has been made in this direction. There is no reason to doubt that there will be regular and healthy extension of this field.

The use of coke-oven gas for power generation in the gas engine has also made some progress. One 500 h.p. unit has been in operation for some time, and another plant of 3000 h.p. is well on toward completion. Provision is also being made for a gas power installation at the Illinois Steel Company's oven plant at Joliet.

The first plant referred to is at the works of the American Iron and Steel Company, at Lebanon, Penn. It consists of one 22x30 Westinghouse horizontal double-acting tandem gas engine of the four-cycle type having a rated capacity of 500 brake horse-power and 10 per cent. over-load. The engine is directly connected to a 300 kw. 40-cycle Westinghouse generator operating at 440 volts and 150 r.p.m. This unit develops current for general power and lighting purposes and is frequently run in parallel with a steam unit at times of large power consumption. The gas used is from the Semet-Solvay coke oven plant at Lebanon, Penn. The gas is first passed through a tower scrubber containing a water spray and then through an oxide purifying box to remove the sulphur. In running with gas that had not passed through the oxide box, the main trouble encountered was with the piston rod, on which the products of combustion appeared to condense, vitiating the lubrication and causing corrosion. This trouble was practically obviated, it is stated, by keeping the water with which the rod was cooled at a high temperature, but there was also some clogging in the valves, so that the oxide purification was continued, and is considered as advisable as yet.

The 3000 h.p. installation is to be at the plant of the Cornwall Ore Banks Company, at Lebanon, Penn. It is to comprise six Westinghouse units of the same type as the plant already described, the engines using gas from the same source. The current will be used for general light and power purposes in the ore mines operated by the company.

Coke Production.—The amount of coke produced in by-product ovens for each year since their introduction in this country in 1893 and the total annual production of the country for the same period, together with the percentage for the by-product coke, are shown in Table II.

The production of by-product coke shows an increase of 934,298 tons for 1907 over 1906, or 20.5 per cent., while the proportion of by-product coke has risen from 12.5 per cent. in 1906 to 15.0 per cent. in 1907. While the actual increase of tonnage from the by-product ovens was not quite as great as in 1906, yet they have more than held their place in the industry. This is strikingly apparent when the proportion of the increased production from by-product ovens to the total increase in output of the country is considered. While the by-product coke has increased nearly a million

tons, the total production has increased by little more than one-third as much. The increased production of the United States, therefore, is entirely due to the by-product oven.

TABLE II. COKE PRODUCTION OF THE UNITED STATES. (a)
(Tons of 2000 lb.)

Year.	By-product.	Total.	Per cent. of total.
1893	12,850	9,477,580	0.13
1894	16,500	9,203,632	0.18
1895	18,521	13,333,714	0.14
1896	83,038	11,788,773	0.7
1897	261,912	13,288,984	2.0
1898	294,445	16,047,209	1.8
1899	906,534	19,668,569	4.6
1900	1,075,727	20,533,348	5.25
1901	1,179,900	21,795,883	5.4
1902	1,403,488	25,401,730	5.5
1903	1,882,394	25,274,281	7.4
1904	2,608,229	23,661,106	11.1
1905	3,462,348	32,231,129	10.7
1906	4,558,127	36,401,217	12.5
1907	5,492,425	36,665,400	15.0

(a) This table is based on data taken from "Mineral Resources of the United States" except for 1907, the figure for by-product coke being made up from reports by the producers to the writer and that for total coke being the report of *The Mineral Industry*.

If we omit from Table I the ovens in Canada and those that were not completed, and make allowance as far as possible for the ovens not in operation during 1907, we have about 3424 ovens from which the output of by-product coke for that year was derived. Using the figure for the total by-product coke produced in 1907 as given in Table II we find that the annual product of each oven was about 1606 net tons. The official figures for the previous years were 1356 tons for 1906, 1159 tons for 1905, 896 tons in 1904 and 962 tons in 1903. This steady increase in the annual output per oven shows both the higher efficiency with which the plants are being operated and the effect of the ovens of larger capacity more recently built. The individual production of the bee-hive ovens operated in 1906 is given officially as 373.6 net tons per oven, a slight gain over the year before, which was 365.8 tons. The gain in output for the bee-hive oven, however, can only be ascribed to the more continuous operation of the plants, as there has been no change in the capacity or method of operation of the bee-hive oven sufficient to affect these figures. A few rectangular bee-hive ovens of large capacity have been built, and some progress has been made in the introduction of the mechanical coke drawer, but these are as yet unimportant factors in the industry as a whole.

The amount of coal coked in by-product ovens in 1907 was 7,391,285 net tons, as reported to me by the operators with but one exception, which was closely estimated. The yield of this coal in coke was 5,492,425 net tons, as given in Table II, or 74.3 per cent. of the coal. The amount of coal carbonized in by-product ovens and the yield in coke as given in

the official records for previous years is shown in Table III, there being unfortunately a gap from 1898 to 1903, for which no figures are available.

TABLE III. COAL COKED AND COKE YIELD IN BY-PRODUCT COKE OVENS IN THE UNITED STATES.
(Tons of 2000 ℔.)

Year.	Coal coked.	Coke yield per cent.	Year.	Coal coked.	Coke yield per cent.	Year.	Coal coked.	Coke yield. per cent.
1898....	402,297	73.2	1904..	3,572,949	73.1	1906..	6,192,068	73.6
1903....	2,605,453	72.25	1905..	4,628,891	74.8	1907..	7,391,285	74.3

The yield seems now to be about constant, averaging considerably higher than the general average of the coking industry, which is given in the official reports about as 64 per cent. for 1907, but with the comment that even this figure is probably in excess of the facts.

Production of Tar.—The production of tar in the by-product coke ovens of the United States is reported to me as 53,393,495 U. S. gallons in 1907. No official figures are at hand for 1906, but for 1905 the production is given as 36,379,854 gal., and for 1904 as 27,771,115 gal. Therefore there was an increase in production of 8,608,739 gal., or 31 per cent., from 1904 to 1905, and of 17,013,641 gal., or 47 per cent., from 1905 to 1907, showing that while the rate of increase has not been maintained, the actual amount has been about the same.

The yield of tar per 2000 ℔. of coal carbonized was 7.22 gal. in 1907, 7.86 in 1905, and 7.77 in 1904. It should be remembered that these figures include the results from low volatile coals, which yield but little tar, though making an excellent coke; indeed, the falling off in 1907 is principally due to the additional ovens put into operation on these low volatile coals. Several of the plants using coals of moderately high volatile show recoveries of 10 gal. per ton for the year's operation.

Production of Ammonia.—The production of ammonia from by-product coke ovens and other sources is fully discussed elsewhere in this volume. The total recovery from by-product coke ovens is given as 62,700 net tons of ammonium sulphate or sulphate equivalent for 1907. This is equivalent to 17 ℔. of sulphate per 2000 ℔. of coal carbonized.

Production of Gas.—The amount of gas reported to me as made in by-product ovens and sold during 1907 is 12,587,743,500 cu.ft. This figure cannot be regarded as rigidly accurate, as some of the individual figures are made up from estimates, but it is believed that it is too low rather than too high. Nor does it include figures for gas produced from all the oven plants as some operate on low volatile coals which yield only enough gas for heating the ovens, and at other plants the gas is not sold, but is merely used for steam raising and general purposes around the plant, and therefore is not included in the above figure. The amount of coal carbonized at the plants producing this gas was 5,597,609 tons, or the

gas sold amounted to about 2250 cu.ft. per 2000 ℔. carbonized. Besides the gas sold, a certain amount of the surplus gas made at some of these works is reported as used around the plant. Allowing for these amounts, and estimating the gas produced but not measured on a conservative basis, it may safely be stated that the production of gas from by-product coke ovens in 1907 amounted to at least 15,500,000,000 cubic feet.

The Koppers Oven.—The system of by-product coke ovens designed and built by Heinrich Koppers, of Essen-Ruhr, Germany, is the latest comer in the American field. Although a number of these ovens have been built abroad, the plant of 280 ovens now building for the Illinois Steel Company at Joliet is the first installation in the United States. The description of the oven here given is prepared from the latest printed data obtainable.

The Koppers oven is of the usual retort oven form and dimensions, and is of the vertical flue type. The oven walls sustain the weight of the superimposed brickwork and contain a single system of vertical heating flues, with a horizontal connecting flue at the top, there being 28 to 32 vertical flues to a wall. The main object of the designer, it is stated, was to obtain an even and easily controlled distribution of the heat throughout the oven wall, and to this end each of the vertical flues is provided with a separate burner. This is a further development of the methods adopted in the underfired type of oven, known both in this country and abroad, but the Koppers oven is distinguished from these in that the distribution of the gas to the different burners was made by means of pipes and regulating valves outside the oven in the underfired ovens, but in the Koppers oven is made through specially formed refractory clay nozzles and sliding damper tiles placed inside the oven. On this method of inside gas distribution much stress is laid.

As built abroad, the Koppers ovens are of two types, the non-regenerative or "waste-heat" oven, in which the heat carried out by the exit gases is recovered in steam boilers, and the regenerative oven, in which the waste heat recovery is accomplished by means of chambers filled with checkerbrick and used to preheat the air for combustion. In both types the flame course through the oven heating flues is upward, then along the top connecting flue and then downward to the exit, but in the regenerative type the course is reversed at regular intervals, usually 30 minutes. The "waste heat" type was the first put forward and still finds application where steam supply is the desideratum, but where there is a profitable outlet for the gas, either for power development in gas engines, or other use, the larger quantities recovered by the regenerative oven make it preferable. As the regenerative type of oven is, generally speaking, more adapted to the conditions prevailing in this country, and is the type now building here, its construction will be described in detail.

Longitudinal and cross sections through the regenerative oven are shown in Figs. 1 and 2. In the first regenerative ovens built by Koppers there were two long regenerative chambers extending under each oven battery, parallel with its length, with a connection to each oven, as in the familiar Otto-Hoffman type. In the later type, known as the "cross regenerator," Koppers provides each oven with two separate regenerators, one for each end, together occupying the space immediately below

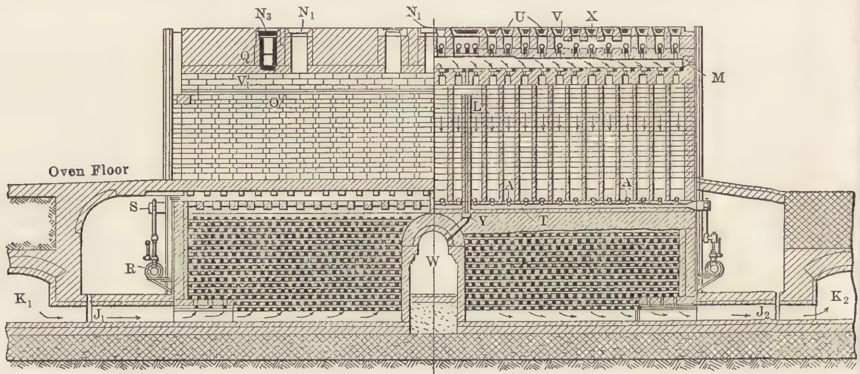


FIG. I. LONGITUDINAL SECTION OF KOPPER'S REGENERATIVE BY-PRODUCT COKE OVEN.

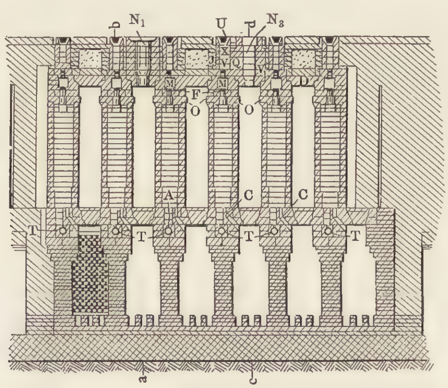


FIG. 2. CROSS-SECTION THROUGH KOPPER'S OVEN.

each oven. The walls of the regenerative chambers serve as the support for the oven walls, and extend the whole length of the oven and downward to the concrete foundation. Thus the construction resolves itself into a series of parallel walls joined at the top and at the oven floor level, the upper chamber forming the oven and the lower the regenerators. Each oven is provided with three charging openings *H* and an additional opening for the gas uptake connection.

Referring to the longitudinal section, Fig. 1, the path of the air for combustion, as shown by the arrows, is from the distributing canal K past the damper and along the canal J, then upward through the checker-brick, and by the ports C, shown in Fig. 2, to the vertical heating flues in the oven wall. The heating gas enters from the same side as the air through the distributing main R, and pipe connections S as shown, and passes along the canal T to the distributing nozzles A. Here it meets the air, and ignites, passing upward through the vertical flues and the slide dampers F to the horizontal canal M, again downward through the dampers F, vertical wall flues and portholes C to the regenerators, thence through the canal J_2 to K_2 and to the stack. The regulation of the draft passing through the vertical flues is accomplished by adjusting the slide dampers F which are placed at the top of each flue. These are reached through the stoppers U and V, which on being removed also allow the nozzles A at the lower end of the combustion flue to be removed and replaced if necessary; it is stated that the slide dampers do not fuse and stick in their places, and that the initial setting is usually sufficient. The same is true of the nozzles below, it being at most, necessary merely to stop the flue off for a little while in case one sticks.

The general regulation of the inlet gas pressure and the outgoing draft is accomplished, as usual, by the use of cocks on the gas pipe and dampers on each waste gas flue, as well as by the main chimney damper. It is the distribution of the gas and chimney draft in the individual oven flues that is sought by the regulating devices described above. With this same object in view the upper horizontal heating flue was built with an increased area in its middle portion, in the first ovens constructed as shown in Fig. 1, but it is stated that the other devices have proved sufficient for the purpose, so that this modification has now been omitted as superfluous.

For heating up the oven at the start Koppers provides the auxiliary uptake N_3, the opening Q and the distributing canal X. By removing the stoppers V and V_1 the gas generated in the oven is thus allowed to pass directly into the heating flues and the regenerators. In order to make it easy to push the finished charge of coke out of the oven, the oven walls are spaced a little wider apart at the discharge end than at the pusher end. The amount of this taper or "draw" varies with the quality of the coal. It also introduces a variation in the coking time of the two ends of the oven, due to the varying thickness of the coal body to be treated, and calls for correction in the method of heat distribution, if an even coking throughout the charge is to be obtained. To overcome this objection Koppers places the division between the upward and downward path of the heating gases, not in the center of the oven wall, but a little toward the discharge side, there being 14 vertical flues with their accompanying gas and air ports on that side and 17 on the pusher side. Thus

the egress for the heated gases from the larger number of burners is through the smaller number of outlets, the narrowing of the outlet area being counted upon to increase the intensity of the heating effect.

As practically two-thirds of the space beneath the ovens is taken up with regenerators, it may be readily appreciated that the regenerative surface thus afforded is ample for the heating of the air to the maximum degree obtainable in practice. As has long been known and may be readily demonstrated by theory, the amount of heat available in the waste gases, even allowing for stack temperatures that will ensure ample draft, is more than the air required for combustion will actually be able to absorb. In order to render some of this lost heat available, Koppers has introduced a supplementary offheat flue *W* underneath the oven battery, which withdraws a certain amount of heated waste gases from each oven through a flue *L*, connecting with the upper horizontal flue. The amount taken is regulated by a sliding damper at the top of the flues. The waste heat is conducted through *W* to *X* steam boilers where the heat is absorbed. A trial of this arrangement is being made at a plant in Germany.

COPPER.

By W. R. Ingalls.

Our revised statistics show that the production of copper in the United States in 1907 was 879,241,766 ℔., against 917,620,000 ℔. in 1906. The statistics for 1907 are based on reports from all of the Michigan mines and all of the other smelters as follows: (1) Refined copper obtained from the "mineral" produced by the Michigan mines during the year; (2) fine copper content of the blister copper produced by the smelters; (3) to a small extent, the copper content of ore and matte (with a suitable allowance for loss) treated by Eastern smelters and the copper content of bluestone made directly from matte. The Eastern smelters treat mainly

COPPER STATISTICS IN THE UNITED STATES.
(In pounds.)

	1901	1902	1903	1904	1905	1906	1907
Alaska	(a)	(a)	(a)	2,043,586	4,703,609	8,700,000	6,610,000
Arizona	126,183,744	119,841,285	153,591,417	191,602,958	222,866,020	263,200,000	256,866,761
California	33,667,456	25,038,724	19,113,861	29,974,154	16,697,486	24,421,000	34,398,823
Colorado	7,872,529	8,463,938	7,809,920	9,401,913	9,854,174	9,565,000	13,344,118
Idaho	480,511	(a)	(a)	5,422,007	6,500,005	9,493,000	11,471,101
Michigan	155,511,513	170,194,996	192,299,485	208,329,248	218,999,759	224,071,000	220,317,041
Montana	229,870,415	266.500,000	272,555,854	298.314,804	319,179,880	299,850,000	226,290,873
Nevada	(a)	(a)	(a)	(a)	(a)	426,000	1,462,450
New Mexico	9,629,884	(a)	(a)	5,368,666	5,638,843	6,262,000	8,652,873
Utah	20,116,979	23,939,901	38,302,602	47,062,889	51,950,782	49,712,000	68,333,115
Wyoming	2,698,712	(a)	(a)	3,565,629	2,393,201	146,000	2,919,137
Southern States	6,860,039	13,599,047	13,855,612	15,211,086	14,907,982	18,821,000	22,408,696
Other States	4,551,430	9,218,490	10,846,477	1,418,065	1,550,000	3,379,000	6,166,098
Total production	597,443,212	636,796,381	708,375,228	817,715,005	875,241,741	917,620,000	879,241,766
Stock, January 1	93,050,230	209,587,698	162,935,439	230,111,792	208,376,672	132,587,496	139,385,400
Imports	176,472,369	161,551,040	167,161,720	182,292,205	210,724,685	225,593,281	253,523,682
Total supply	866,965,811	1,007,935,119	1,038,472,387	1,230,119,002	1,294,343,098	1,275,800,777	1,272,150,848
Deduct exports	227,194,184	376,298,726	312,822,627	555,638,552	548,772,403	467,839,041	519,280,730
Deduct consumption	440,913,929	468,700,954	495,537,968	466,103,778	612,983,199	668,576,336	531,610,118
Stock, Dec. 31 (b)	197,857,698	162,935,439	230,111,792	208,376,672	132,587,496	139,385,400	221,260,000

(a) Included in "Other States." (b) Computed by method described in *Eng. and Min. Journ.*, July 25, 1908.

material from foreign sources, and rather than attempt to distribute their blister-copper production according to origin, it has been considered safer to reckon the copper content of the ore and matte which they received from domestic sources, the major portion of this material being matte, a smelter product which is already high in copper. The total of the copper thus reckoned is less than 2 per cent. of the whole production. Thus the statistics for 1907 have been computed on a more uniform basis than ever before by ourselves or any other statisticians.

Although the effort has been to conform to the above basis in previous years, it has been impossible to avoid the incorporation of some reports submitted on the basis of refined copper, which have introduced certain confusion as to time. Blister copper from the West is in transit, and process of refining for an average of 60 days after shipment from the works of origin. Consequently the statistics of the production of refined copper and of blister copper are for different periods of time. Statistics of the production of refined copper show the metal made ready in final marketable form, but such statistics for any calendar year represent the metal shipped from the Western smelteries during the year ending Oct. 31, or even earlier. The statistics of blister-copper production are therefore more

PRODUCTION OF COPPER ACCORDING TO CLASS.
(In pounds.)

Year.	Total Domestic. (*d*)	Total Foreign. (*e*)	Grand Total.	Lake.	Electrolytic.
1897	501,370,295	26,938,254	528,308,549	145,839,749	250,000,000
1898	535,900,232	36,055,352	571,955,584	156,669,098	314,107,776
1899	581,319,091	40,659,868	621,978,959	155,845,786	386,410,356
1900	600,832,505	62,484,290	663,316,795	144,227,340	466,092,663
1901	597,443,212	....	....	155,570,465	(*a*)485,016,400
1902	636,796,381	....	....	170,194,996	(*a*)606,270,500
1903	708,375,228	....	....	192,299,485	(*a*)617,293,600
1904	817,715,005	....	....	208,329,248	705,478,400
1905	875,241,741	183,252,259	1,058,494,000	219,000,000	(*b*)760,000,000
1906	917,620,000	(*c*)247,549,000	(*c*)1,165,169,000	224,071,000	(*b*)865,000,000
1907	879,241,766	(*c*)273,506,124	(*c*)1,152,747,890	220,317,041	854,441,000

(*a*) As estimated by the Metallgesellschaft, Frankfurt am Main. (*b*) Partly estimated. (*c*) Includes copper from domestic scrap and junk. (*d*) Entered the same as production of the mines. (*e*) Difference between the first and third columns.

Year.	Prime Lake.	Arsenical Lake.	Electrolytic. (*d*)	Casting. (*d*)	Pig Copper. (*a*)	Total.
1904	171,284,430	37,044,818	705,478,400	(*b*)45,000,000	44,408,000	1,003,215,648
1905	175,457,000	43,542,000	(*c*)760,000,000	46,000,000	33,495,000	(*c*)1,058,494,000
1906	180,273,426	43,797,574	(*c*)860,000,000	52,000,000	29,098,000	(*c*)1,165,169,000
1907	176,264,418	44,052,623	854,441,000	47,957,890	30,032,000	1,152,747,890

(*a*) Exported. (*b*) Estimated. (*c*) Partly estimated. (*d*) Includes copper from scrap and junk.

nearly representative of the production of the mines; in fact, as near, we think, as it is possible to go. Moreover, blister copper is a product that is never shipped from one smelter to another; its destination is invariably to a refiner; consequently, statistics computed on this basis eliminate danger of duplication about as far as is possible, but not entirely, inasmuch as certain refiners work up their by-products in such a way that their copper content appears a second time as blister copper. Many of the Eastern smelters purchase considerable quantities of scrap metal, which appears in their statistics. These items are deducted from their reports so far as possible, and in the statistics for 1907 we believe there is but little, if any, overstatement on these accounts. On the other hand, it is to be remarked that all the copper in blister does not appear finally as refined

metal, a portion of it (very small, however) going into bluestone, a by-product of the refining process.

ELECTROLYTIC COPPER REFINERIES OF THE UNITED STATES.
(Approximate annual capacity at ends of 1906 and 1907.)

Works.	Location.	1906 Capacity, Pounds.	1907 Capacity, Pounds.
Nichols Copper Company	Laurel Hill, N. Y.	288,000,000	300,000,000
Raritan Works	Perth Amboy, N. J.	288,000,000	300,000,000
American Smg. and Ref. Company	Perth Amboy, N. J.	144,000,000	144,000,000
U. S. Metals Refining Company	Chrome, N. J.	144,000,000	144,000,000
Baltimore. Cop. Roll. and Mfg. Co	Baltimore, Md.	130,000,000	130,000,000
Balbach Smelting and Refining Co	Newark, N. J.	75,000,000	75,000,000
Boston and Montana Copper Co	Great Falls, Mont	48,000,000	48,000,000
Tacoma Smelting Co	Tacoma, Wash	18,000,000	18,000,000
Mountain Copper Co	Oakland, Cal	3,000,000	3,000,000
Chicago Copper Refining Co	Blue Island, Ill.	2,000,000	2,000,000
Calumet & Hecla Mining Co. (a)	Buffalo, N. Y.	25,000,000	25,000,000
North American Lead Co	Fredericktown, Mo	....	2,000,000

(a) Refines Lake copper.

The distribution of production according to States is made on the basis of the reports of the smelters; all of the smelters report the origin of most of the ore and matte which they converted into blister copper. In the case of matte purchased from other smelters, the latter have reported the origin of the ore which they turned into matte. In general the reports of matte shipped and matte received have agreed very well, similarly as to ore received and matte shipped, but in some cases there are discrepancies, which are explainable, of course, by additions to, or drafts from, stock on hand. This shows why it is impossible to carry statistics of production clear back to the mines themselves. On the other hand, reports from the mines are of comparatively little value, because they seldom know how much of the metal is extracted from their ore, and often do not know how much metal their ore contains. The statistics based on the blister-copper production are most representative of the production of the mines in the aggregate, but the allocation of the production according to States of origin can never be precise, although in most cases it can be arrived at closely.

There is one particularly noteworthy feature in the statistics for 1907. This is the remarkably large output made by certain States which have not heretofore figured as copper producers of any consequence. Some of these have had to be grouped in order to avoid the disclosure of identity, but it may be remarked that Alabama, North Carolina and Virginia produced nearly 2,500,000 ℔. of copper in 1907; Wyoming produced upward of 2,900,000 ℔.; Nevada produced nearly 1,500,000 ℔.; Vermont produced nearly 700,000 ℔., and Missouri figured with a substantial output. These figures show the great stimulus of a high price for copper in bringing

out production from many small mines which normally can not be worked at a profit. The same condition is reflected in the statistics of the States like Colorado and New Mexico whose ore goes chiefly to custom smelters. The high price prevailing during the major part of 1907 led to the operation of many little mines, whose output was large in the aggregate. Since the price passed under 15c., many of these have ceased to operate.

EXPORTS OF COPPER FROM THE UNITED STATES. (a)

Ore, matte and regulus stated in tons of 2240 lb. Ingots, etc., in pounds.

Country.	1903	1904	1905	1906	1907
Ore, matte and regulus					
Exported to:					
United Kingdom	318	164	50	206	200
Germany	....	102	....	59	188
Brit. North America	....	3,486	24,690	36,700	82,016
Mexico	10,667	15,175	12,948	10,600	16,737
Other countries	1,306	....	....	54	....
Total	512,291	18,927	37,688	47,619	99,141
Ingots and scrap (b)					
Exported to:					
United Kingdom	47,140,717	112,224,871	60,945,794	55,097,670	81,409,441
Belgium	4,209,720	9,365,791	4,997,206	6,475,054	3,822,551
France	53,745,221	99,888,455	74,604,455	80,703,723	93,075,145
Germany	71,130,077	103,825,445	104,575,864	96,629,040	107,607,390
Italy	7,774,016	15,297,091	15,800,967	19,777,296	21,192,908
Netherlands	96,927,346	147,678,581	130,675,386	151,650,293	156,652,270
Russia	10,411,679	22,333,578	18,418,982	9,523,992	4,341,386
Other Europe	16,516,663	29,064,494	25,279,162	25,260,807	26,221,024
Brit. North America	2,644,831	3,472,614	3,019,450	4,176,135	3,747,410
Mexico	165,283	191,429	290,763	263,319	362,411
China	....	10,403,034	79,940,250	4,932,128	10,003,592
Other countries	63,971	804,647	16,359,751	262,561	493,873
Total	310,729,524	554,550,030	534,907,619	454,752,018	508,929,401

(a) The imports of ore, matte and regulus are reported as gross weight, the copper contents not being stated. (b) Includes bars and plates.

IMPORTS OF COPPER INTO THE UNITED STATES.

(In pounds.)

Country	1903	1904	1905	1906	1907
Ore and matte					
Imported from					
Brit. North America	(a)243,918	15,046,131	15,403,429	10,329,955	12,803,069
Mexico	(a) 39,261	20,803,961	28,890,239	31,690,058	33,476,046
South America	(a) 77	91,509	1,503,427	4,140,589	8,685,621
Other countries	(a) 1,656	3,006,121	4,308,205	2,874,289	5,657,679
Total	(a)284,912	38,947,722	50,105,300	49,034,891	60,622,415
Pigs and scrap (b)					
Imported from:					
United Kingdom	18,788,558	19,172,854	26,284,302	22,549,321	25,706,852
France	1,926,279	22,075	1,549,138	3,202,168	606,662
Germany	1,600,766	875,329	2,945,441	5,303,712	6,814,338
Other Europe	240,689	16,943	1,955,358	5,649,689	5,616,261
Brit. North America	15,923,760	17,690,656	23,636,843	30,398,369	30,902,596
Mexico	89,361,100	97,965,593	102,646,343	85,595,359	76,741,532
Cuba	467,832	368,634	433,440	513,240	767,184
West Indies (c)	317,112	373,743	278,502	399,569	401,585
Japan	3,604,643	80	....	6,752,486	9,809,569
Other countries	4,477,256	5,858,535	890,018	16,194,477	35,534,688
Total	136,707,995	142,344,433	160,619,385	176,558,390	192,901,267

(a) Tons of 2240 lb. copper content not given. The imports reported for 1904-1907 are the copper contents of ore, matte and regulus. (b) Includes also bars, ingots and plates. (c) Includes Bermuda.

It is not easy to arrive at an accurate conclusion respecting the stock of refined copper on hand at the end of 1907, inasmuch as producers were reluctant to report their holdings, for obvious reasons. Ten of the Michigan companies producing 87,647,135 lb. of copper in 1907, had on hand at the end of the year 21,331,110 lb., or 24.3 per cent. of their output during the year. The total stock of Lake copper may be estimated at 40,000,000 lb. On the basis of the reports received, and reliable trade information, I estimate the stock of refined electrolytic and casting copper at 80,000,000 lb. The stock of copper in transit and in process of refining is estimated at 101,260,000 lb. This gives a total estimated stock (in the United States, exclusive of copper in European warehouses for American account) of 221,260,000 lb. This estimate enables me to compute the domestic consumption of new copper at 531,610,118 lb., as appears in the accompanying table. An independent, and as I believe a more accurate, computation on the basis of refined copper indicates a consumption of about 539,000,000 lb. (See *Eng. and Min. Journ.*, July 25, 1908.)

THE WORLD'S COPPER PRODUCTION. (*a*)
(In metric tons.)

Countries.	1900	1901	1902	1903	1904	1905	1906	1907
Argentina	76	793	244	137	157	157	107	224
Australasia	23,368	31,371	29,098	29,464	34,706	34,483	36,830	41,910
Austria-Hungary	1,377	1,356	1,626	1,407	1,473	1,346	1,458	1,062
Bolivia	2,134	2,032	2,032	2,032	2,032	2,032	2,540	2,540
Canada	8,595	18,575	17,765	19,360	19,490	21,588	19,106	21,022
Cape of Good Hope: Cape Co.	4,491	4,064	2,794	4,704	5,563	5,105	4,003	4,298
Cape of Good Hope: Namaqua	2,337	2,439	1,727	610	2,337	2,337	2,642	2,540
Chile	26,016	31,299	29,373	31,424	30,592	29,632	26,157	27,112
Germany—Total	20,635	22,069	21,951	31,214	30,262	22,492	20,665	20,818
(Mansfeld)	(18,684)	(19,082)	(19,050)	(19,810)	(19,578)	(19,878)	(18,085)	(17,343)
Italy	2,797	3,048	3,424	3,150	3,388	2,997	2,911	3,353
Japan	28,285	27,916	30,251	3,861	35,408	36,485	40,528	49,718
Mexico—Total	22,473	33,943	36,357	46,040	51,760	70,010	62,690	61,127
(Boleo)	(11,297)	(10,956)	(10,958)	(10,480)	(11,120)	(10,341)	(11,002)	(11,506)
Newfoundland	2,929	2,800	2,906	2,753	2,235	2,316	2,332	1,758
Norway	3,998	3,429	4,638	6,010	5,502	6,406	6,218	7,122
Peru	8,353	9,673	7,701	7,925	6,863	8,763	8,641	10,744
Russia	8,128	8,129	8,814	10,485	10,871	8,839	10,658	15,240
Spain-Portugal—Total	53,718	54,482	50,587	50,536	47,788	45,527	50 109	50,470
Rio Tinto	36,304	35,916	35,032	36,382	34,016	32,796	34,644	32,832
Tharsis	8,092	7,546	6,817	6,421	5,710	4,415	4,816	4,206
Mason & Barry	3,515	3,789	3,383	2,469	2,997	2,764	2,504	2,662
Sevilla	1,483	1,313	1,570	1,123	1,351	1,300	2,073	2,337
Sweden	457	457	462	462	542	559	508	2,032
Turkey	2,341	1,665	1,118	1,422	965	711	432	1,270
United Kingdom	777	610	488	544	501	726	762	711
United States	274,933	270,998	288,833	316,239	370,892	397,003	416,226	398,736
Totals	496,819	532,148	542,209	602,832	663,327	699,514	715,523	723,807

(*a*) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by *The Mineral Industry* direct from official sources.

The fact that certain mines in Michigan yield a mineral so free from objectionable impurities that a high-grade copper is obtainable directly by a simple furnace scorification, early gave Lake copper a premium in the market that electrolytic competition even yet has not entirely wiped out. For almost all purposes there should be no hesitation whatever

today in specifying electrolytic copper. In conductivity it is superior to nearly all brands of Lake, and in mechanical properties it leaves little to be desired. There are a few classes of work in which the metal is subjected to very severe stress, such as in the making of cartridges, where Lake seems to stand up better than electrolytic, although even here it is a question how much to allow for the trade prejudices of the older generation of mill foremen.

Any advantage in Lake copper must be attributed to the fact that it is not so clean as electrolytic. A small quantity of arsenic, for example, while exceedingly detrimental to the conductivity, will appreciably improve the mechanical properties of copper; at one time, arsenic was considered necessary in English specifications for firebox copper. Each year a greater proportion of the world's output is electrolytically refined, and it seems probable that before long there will be but two grades of copper on the market—high-conductivity copper, which will include electrolytic and picked brands of Lake, and casting copper, covering all material which will not pass a conductivity requirement of 98 per cent., annealed.

WORLD'S PRODUCTION OF COPPER.(a)

Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.
1879	154,471	170,310	1889	265,516	292,741	1899	476,194	525,021
1880	156,500	172,547	1890	274,065	302,166	1900	496,819	547,761
1881	166,065	183,093	1891	280,138	308,862	1901	532,148	586,712
1882	184,620	203,550	1892	309,113	340,808	1902	542,209	597,805
1883	202,697	223,481	1893	310,704	342,562	1903	602,832	664,644
1884	223,884	246,840	1894	330,075	363,920	1904	663,327	731,342
1885	229,315	252,828	1895	339,994	374,856	1905	699,514	771,239
1886	220,669	243,295	1896	384,493	423,917	1906	715,523	788,890
1887	226,492	249,716	1897	412,818	455,147	1907	723,807	798,023
1888	262,285	281,179	1898	441,282	486,529			

(a) The statistics for 1879–91 are as reported by Henry R. Merton & Co.; 1892-1907 as per *The Mineral Industry.*

Copper Mining in the United States.

Alaska.—Mining development in southeastern Alaska during 1907 was greatest in the Ketchikan district, where several copper mines on Prince of Wales island are being exploited. Mining was actively advanced on this island in the spring and early summer, and considerable low-grade ore was produced, but most of the mines suspended operations in September and October. Many of the mines that were forced to discontinue operations were in good condition for profitable production at the time they closed, and at some it was not the price of copper alone that caused suspension, but unfavorable conditions as to labor and transportation.

The copper occurs in the form of chalcopyrite, either in irregular masses associated with magnetite, garnet, and epidote at the contact of intrusive

rocks with limestone, as at the Mamie, Mount Andrew, and Jumbo mines, or in the form of lenticular bodies associated with pyrite, quartz, and calcite along shear zones in the slates and greenstones, as at the Niblack and other mines. Developments at the Mamie mine disclosed new orebodies on the lowest level and considerable ore was ready for extraction in the upper levels at the time operations ceased. A similar condition existed at the Mount Andrew mine, where large ore reserves were being developed and plans for more extensive operations had been made. At the Rush & Brown mine the copper deposits were being explored 100 ft. below the present working level, which is 90 ft. in depth, and the results were encouraging. Though the developments at the Jumbo mine and Niblack mine are still being advanced and good orebodies are being opened, ore shipments have been temporarily suspended. Work has been discontinued for the winter at most of the prospects which gave promise of becoming copper producers.[1]

The Alaska Smelting and Refining Company, after a period of idleness in the early part of 1908, blew in its furnaces about May 1 and produced matte which was shipped to the smelters of Vancouver Island and to Tacoma. Shortage of fuel had previously been a difficulty.

Prospecting was done in the Copper River region, or more properly the Chitina valley, most of the ore deposits lying north of the Chitina river, on the south slope of the Wrangell mountains, extending from Kotsina river nearly to the head of Chitistone river, a distance of about 70 miles. The Bonanza mines are situated near the head of Kennicott river. Heretofore travel into the Chitina country has been by the way of Valdez, and thence by trail, but during the summer of 1908 it is proposed to put a line of steamboats on the Copper river, to run to the head of navigation on the Chitina river.

The Copper River country was visited by members of the U. S. Geological Survey in 1907, who reported that prospecting was being carried on most actively at places between Kotsina and Chitistone rivers near the contact of the Nikolai greenstone and the overlying Chitistone limestone. This contact can be traced with little difficulty, although it is in some places covered by later sediments, and it has served as a guide for prospecting ever since the region was opened.

Most of the ores of the Kotsina region are of the same general type, being chiefly bornite and chalcocite, and less frequently chalcopyrite or copper-bearing pyrites in greenstone. At one place the copper ore occurs in limestone. The greenstones have been crushed and faulted, and developments of secondary mineral have taken place locally. The copper ore occurs as lenses and stringers in the greenstone, at one place in the limestone, and in most places occurs in sheared zones of the country

[1] C. W. Wright, U. S. Geological Survey.

rock, or associated with joint or fault planes. In most places the ore occurs within a short distance of the Chitistone limestone.

The largest orebody so far known in the Chitina valley is the Bonanza, owned by the Kennicott Mines Company. The ore is almost entirely chalcocite and is a replacement of limestone along a series of perpendicular fractures or fault planes. This orebody is found about 6000 ft. above sea level. It occurs in the limestone only a few feet above the contact with the underlying greenstone, but so far as yet known does not penetrate the greenstone. The ore is well exposed on the surface, and has been crosscut by two tunnels, but no further development has yet been done. A wire tramway is under construction to carry ore from this mine to bins at the mouth of National creek, whence it may be transported to steamboats on Chitina river or to the railroad when that is completed.

East of the Bonanza mine is the Nikolai mine on Nikolai creek, which was the first mining claim patented in this region; since the patent was granted, no further work has been done. The most extensive operations on the Kotsina river in 1907 were those of the Great Northern Development Company, which had 75 or 80 men employed.

The great handicap to prospecting in this region has been the high cost of transportation. In July, 1907, a small steamboat made the first trip from Abercrombie Rapids to the mouth of the Nizina on Chitina river, but after making one trip was hauled out. She will probably be used in 1908 for transporting supplies for the Copper River & Northwestern Railroad. This road is now being built from Cordova Bay to Copper and Chitina rivers. From Abercrombie Cañon a branch line will run to Katalla and the Bering river coalfields.

Arizona.—The production of this Territory in 1907 showed only a small decrease, and in view of the fact that Montana experienced a large decrease, Arizona for the first time took the premier place. Some of the individual mining districts of Arizona showed increases. The works of the Consolidated Arizona Smelting Company, at Humboldt, were in operation during the early part of the year, but were closed during the fall, when the company went into the hands of a receiver. The Imperial Copper Company was engaged in the construction of a smelting works of 350 tons daily capacity at Sasco, near Red Rock, on the Southern Pacific Railway. These works went into operation early in 1908. The general conditions of copper production in Arizona in 1907 are carefully reviewed in the special article by Doctor Douglas, which follows.

The mining companies of Arizona are now taxed under a law, which provides that every mining company whose gross production in any one year is of a value of $3750, or more, must submit to the Territorial auditor and county assessors a statement of the products for a basis of assessment.

The reports to the Territorial auditor for 1906 are given in the accompanying table.

PRODUCTION OF THE MINES OF ARIZONA IN 1906.

Reported to the Territorial Auditor.

County.		Companies.	Location.	Copper, lb.	Gold, Ounces.	Silver, Ounces.	Value Product.
Cochise	1	Copper Queen Consolidated	Bisbee	71,711,813	7,573	382,211	$14,236,428
	2	Calumet & Arizona	"	36,934,387	4,241	194,451	7,337,748
	3	Superior & Pittsburg	"	9,044,875	143	32,564	1,768,377
	4	Shattuck Arizona	"	2,541,680	...	...	489,985
	5	Tombstone Consolidated Mines	Tombstone	...	7,043	577,221	650,218
	6	Two companies, no returns, assessed full value	...	...	...	...	17,500
	7	Miscellaneous, four companies	...	917,242	4,923	456,431	583,410
Coconino	8	Canyon Copper	Grandview	26,798	...	324	5,383
Gila	9	Old Dominion Copper	Globe	16,602,186	1,798	74,123	3,286,125
	10	United Globe	"	4,607,537	...	5,277	891,772
	11	Saddle Mountain Mining	Christmas	2,338,492	437	16,410	470,515
	12	Live Oak	Globe	1,714,074	...	...	330,439
	13	Gibson Copper	"	1,106,100	...	...	213,234
	14	Arizona Commercial Copper	"	939,102	...	...	181,040
	15	Miscellaneous, three companies	"	1,592,343	...	...	305,972
Graham	16	Arizona Copper	Clifton	25,937,006	...	...	4,973,421
	17	Detroit Copper	Morenci	19,588,412	70	2,407	3,762,671
	18	Shannon Copper	Clifton	9,592,773	698	23,340	1,868,963
	19	New England & Clifton	"	1,140,770	43	10,472	228,508
	20	Miscellaneous, five companies	...	698,013	...	24,744	150,130
Maric'pa	21	Relief Gold	...	...	738	...	14,778
Mohave	22	Gold Road Exploration	Gold Roads	...	26,299	12,013	551,629
	23	Miscellaneous, four companies	...	27,653	1,346	40,605	90,081
Pima	24	Imperial Copper	Silver Bell	4,385,246	...	44,327	874,994
	25	Twin Buttes Mining	...	700,329	...	11,294	142,553
	26	Helvetia Copper	...	379,823	28	3,898	76,446
	27	Tip Top Copper	Tip Top	298,890	...	...	57,620
	28	Miscellaneous, two companies	...	151,000	...	414	29,466
Santa C.	29	Miscellaneous, two companies	...	46,136	16	73,373	133,684
Yavapai	30	United Verde Copper	Jerome	38,827,365	12,913	428,317	8,038,126
	31	Ideal Mining & Development	McCabe	518,895	14,696	139,766	497,152
	32	De Soto Mining	Humboldt	1,128,327	903	19,259	249,047
	33	Congress Consolidated	Congress	...	10,410	11,407	222,794
	34	Commercial Mining	Prescott	694,126	...	...	131,114
	35	Miscellaneous, twelve companies	...	820,762	17,055	117,004	655,866
Yuma	36	King of Arizona	Kofa	...	13,643	4,395	284,593
		Totals	...	255,012,155	125,016	2,704,045	$53,801,781

The Calumet & Arizona Mining Company in its report for 1907 shows the results of operations in the five years of its productive career. In every

CALUMET & ARIZONA MINING COMPANY.

Year.	Net Earnings.	Copper Produced. lb.	Price Received.	Gold and Silver Value.	Gross Product.	Cost Per lb.
1903	$1,341,474	25,535,857	11.558c.	$144,862	$3,096,807	6.89c.
1904	1,682,518	31,638,660	12.562c.	195,926	4,170,374	7.86c.
1905	2,314,268	31,772,896	14.932c.	178,843	4,923,172	8.21c.
1906	4,827,872	37,470,284	17.96c.	238,464	6,968,127	5.71c.
1907	2,114,047	30,689,448	18.102c.	210,846	5,765,636	11.22c.

year the item of cost per pound includes all construction and development, the addition of machinery, etc., but for the year 1906 there was spent $374,357 in the development and partial purchase of additional and

separated properties, and this is not included in the cost of producing copper. The great increase in the cost per pound in 1907 was due to the strike in the early part of the year, when the company was fighting the Western Federation of Miners; to the decline of production; the increase of wages; to the fact that during a curtailment of 50 per cent. in production nearly all employees were retained; to a diminution of the copper content from 7.95 per cent. in 1906 to 5.85 per cent. in 1907; and somewhat to large expenditures for additional smelter capacity and additional automatic bin and charging arrangements. The company has now paid 18 dividends amounting to $9,500,000, the first having been in December, 1903. There is a blast furnace capacity of about 2000 tons of ore daily.

The Superior & Pittsburg Copper Company in 1906 produced 9,691,495 lb. of copper, which realized an average of 18.10c. per lb. The average yield of the ore was 3.69 per cent. copper against 4.91 per cent. in 1906. The company mined from 700 to 800 tons of ore per day.

(By James Douglas.) The anticipations formed in 1906 that the output of certain mines in the Warren district would be largely increased in 1907, and that the Copper Queen, for instance, while it might not increase its production, would probably, at least, maintain its former rate, and that other companies in the district would enter the field as large producers in 1907, have not been realized. During the early months of 1907 there was great activity and a production in excess of 1906. During March the Copper Queen works at Douglas turned out 10,160,732 lb. of copper, and the Calumet & Arizona smelter over 4,000,000 lb. The average of the first half of the year was considerably in excess of the second half, when the output of the Copper Queen works at Douglas averaged only 6,927,000 lb. and that of the Calumet & Arizona 3,136,030 lb. per month from their own product and also from custom ores.

The production of the Copper Queen works was 97,452,065 lb. in 1907, but for the first five months the matte from the Detroit works in the Clifton district, containing 8,633,415 lb. of copper, was reduced in the converters of the Copper Queen plant at Douglas. There was also treated at the Douglas smelter of the Copper Queen company some of the ores and concentrates of the Montezuma Copper Company of Nacozari, Sonora, Mexico, containing 6,861,900 lb. of copper, and from the mines of the Indiana-Sonora Copper Mining Company at Cananea, Sonora, ores containing 1,106,716 lb. of bullion. Deducting these three sources of production together with small quantities of other Mexican ores, the output of the Copper Queen smelter from ore of the Copper Queen mine and from custom ores drawn from southern Arizona, amounted to 87,745,284 lb. (including 903,064 lb. from New Mexico). Considering the large production of the early months of the year, these totals show a heavy shrinkage during the closing months.

While there has been a diminished production from the Warren district, the three smelting companies in the Clifton district increased their production over 1906 by approximately 6,000,000 ℔., making 63,000,000 ℔. in 1907. This increased production, however, was due in part to the increased activity in the district at large, a number of smaller companies and prospectors having contributed to the ores supplied from the mines of the smelting companies themselves. The Old Dominion smelter turned out 36,819,206 ℔. of copper, which is slightly in excess of the production of 1906. This amount is made up of 5,600,828 ℔. of copper in sulphides from Mexico and 31,218,388 ℔. from domestic ores, which represents an increase in copper from domestic and a decrease of copper from foreign ores. The United Verde will probably show a slight decline over last year, owing to the reduced activity during the latter months.

The drop in price, which became acute with the drop to 12c. in October, had the effect of shutting down a number of smaller mines throughout the whole Territory, and unfortunately closing the Humboldt smelter on the Agua Fria in Yavapai county, and thereby discontinuing the only open market for the product of a number of small mines within the Prescott district, and some as far south as the Saddle Mountain mines on the Gila.

The recent decline in production in both the Warren district and in northern Mexico has not been due to the depletion of the orebodies, but to good business judgment in restricting output until the accumulation of copper created last summer is worked off. This accumulation occurred through the producing mines not appreciating in time the decline of demand for the metal which they could supply, and therefore maintaining a maximum production for at least five months while they should have been running at slackened speed. It was generally felt that it would be imprudent, unless again urged by the extravagant price, to run the works as actively as was done during the early months of 1907, and that by reducing the production to a figure somewhat below the normal, opportunity is given not only to make necessary repairs, but also for the staff to recover from the excessive strain to which they were exposed. During 1908, therefore, there is little reason to suppose that the output in Arizona will exceed that of 1907, nor will it fall far below it, for it is unlikely that the copper trade can be supplied with an output from the mines as low as that to which it has fallen during the closing months of 1907. The Copper Queen works at Douglas will receive an accession from the Moctezuma Copper Company in Sonora, when its new concentrator is completed, before next summer; but unless the price of copper revives, this augmented quantity will, perhaps, be compensated for by falling off in custom ores from which the same works during 1907 made 17,517,518 ℔. of copper.

The Copper Queen built one large reverberatory furnace for experi-

mental purposes in cleaning slag, and disposing of flue dust and fine concentrates; but as yet no decisive results have been obtained. The same works has also installed two Walker tables, to handle the copper from the converters through an intermediate ladle. At the Calumet & Arizona works, in addition to the present plant of five furnaces, they are understood to be adding two very large furnaces in order to matte the ore production of the Superior & Pittsburg mines. The Arizona Copper Company is erecting an additional large 40-ft. furnace and the Shannon company has recently erected and put into service a new 30-ft. furnace, which is admirably designed for the swift and easy removal of defective jackets, and has other distinctive features worthy of imitation. The only other metallurgical innovation of note is the Mond gas plant erected by the Arizona Copper Company for supplying its gas engine plant with gas made from Gallup coal slack.

The most important copper discovery made in Arizona is that of copper ore in the bottom of two of the deepest workings of the Tombstone silver mines. The Development Company of America has for years been bravely re-opening this famous district, which was virtually abandoned 20 years ago, after the old companies reached water level, and when through lack of coöperation it became impossible to handle the water and to sink through the barren zone, which was encountered at or near the water level. It was in carrying out this exploration in depth that copper was struck in two of the deep workings in November last. Mr. Staunton reported on Nov. 17 "An encouraging feature during the past week has been the appearance of copper ore in a winze being sunk below the 9th level of the Emerald. A few days ago native copper was noticed in the ledge and yesterday a small streak of black copper sulphide running 46 per cent. appeared on the foot-wall. To-day the entire width of the winze is in copper ore, consisting of a pulverized quartz gangue with chalcocite all through it." Almost on the same day copper was struck in the Emerald claim as reported by Mr. Walker, who says, "In sinking this winze (below the 900 level) we struck a little native copper at about 35 ft., and in a couple of feet further we came into a fine showing of copper sulphide (chalcocite). We are now down about 6 ft. in the sulphide and it continues to improve with depth. The winze is all in ore and I do not think we have either hanging-or-footwall."

In another portion of the report General Manager Staunton says "Evidence indicating the addition of copper with depth to the other values in the ore has been frequently referred to in letters and is now finding marked confirmation in the Silver Thread and Emerald mines in both of which the copper has increased so as to be of commercial importance and in the case of the Emerald to be the metal of chief importance in the ore. This development in the Emerald is in a winze being sunk on

the vein below the 9th level of the mine to keep in touch with the recession of the water."

If these discoveries are more than mere indications, the size of the veins in the Tombstone district is so great that the copper product of the Tombstone mines may become a commanding feature in the copper interests of the Southwest. The coming year should determine this question.

The same company has started a smelter in connection with its Imperial mine at Sasco, a point 12 miles from the mine and eight miles from Red Rock, the junction point of the company's railroad, the Arizona & Southern, which unites the Imperial mine with the trunk line of the Southern Pacific. The smelter is two miles from the dry bed of the Santa Cruz valley, from which abundant water will be pumped through an 8-in. pipe. The furnace building as erected is designed for two furnaces, one of which is already in blast, the other of which is understood to be in contemplation. The furnace has a cross-section of 43x192 in. at the level of the tuyères, which are 16 ft. below the feed floor. It is provided with an elliptical settler 12x18 ft. Two stands of 84x126 in. electrically turned converters, with cast steel heads, take the matte from the settlers by means of a 50-ton Pawling & Harnischfeger electric crane having 45 ft. span. The converter plant has a separate dust chamber and stack from those of the furnaces. The boiler plant and steam plant consist of four Morrison internal-corrugated-furnace boilers of the marine type, 10 ft. 6 in. in diameter by 15 ft. length of flue; and the power will be developed from one 11 in. and 22 in. x 36 in. Nordberg Corliss condensing engine, direct connected to a No. 9 Connersville blower, one Nordberg, cross-compound, Corliss condensing blowing engine, having 15 in. and 30 in. steam cylinders with two 36 in. air cylinders and 42 in. stroke. Electrical power will be furnished by a 500 kw. Westinghouse turbo-generator, running at 440 volts, alternating current. The necessary direct current of 220 volts for operating the crane, converters, trolley line, etc., will be obtained by a motor generator set. The sampling mill, relining machinery, pumps, etc., will be operated by 440-volt induction motors. The surplus electrical power available from the turbine unit will be stepped up by transformers to 11,000 volts and sent to the mine for operating the mill. To provide against delay from a shut down of the turbine for any reason, an additional spare 150-kw., 440-volt generator has been provided, direct connected to a Westinghouse vertical compound engine. This will supply sufficient power for all purposes at the smelter. The switchboard is so arranged that the generators are interchangeable, or may be connected to run in parallel if desired. Condensers of the Alberger dry vacuum type have been provided, that for the turbine having 2000 sq.ft. of cooling surface, and that for reciprocating engines 1400 sq.ft.

It will be seen, therefore, that this new smelting plant, though as yet not of large capacity, is designed skillfully for economic work, and the explorations underground, both in smelting and concentrating ore, warrant the belief that the output from its own establishment will hereafter be considerably in excess of the four to five million pounds of copper per annum which have heretofore been smelted for the Imperial Copper Company by the Copper Queen furnaces at Douglas. To utilize the low grade ore the company is already erecting the first unit of a concentrating plant.

(By Robert B. Brinsmade.) The country north of the Grand Cañon in Arizona has copper mines situated on the Kaibab plateau, or Buckskin mountain, which lies just north of the great southern loop of the Colorado river. The copper ore lies near the surface of the plateau in a stratum of cherty sandstone of the Upper Carboniferous. The ore is found in runs along shallow gash veins; in the veins it sometimes reaches a depth of 30 ft. below the surface, but the base of the runs is seldom over 10 ft. deep. The minerals are malachite, azurite, chrysocolla, a little limonite, a trace of calcite and the remainder silica. The ore lies in certain layers, 2 to 4 ft. thick, and at vertical intervals of several feet when several layers occur in one run. The largest run observed was 600 ft. long and 50 to 100 ft. wide, with a total thickness of 4 to 8 ft. of ore, carrying about 6 per cent. copper.

Until recently it was thought by many that these deposits occurred as a continuous bed over the whole district. The early prospectors encouraged the belief, for they dug their trenches and pits only where there was ore. These wily deceivers avoided digging in barren places by first testing each proposed excavation with a churn drill hole which was easily affected on account of the nearness of the runs to the surface.

The Petoskey mine was acquired in 1900 by the Petoskey Mining Company which built a pumping station three miles away on the side of Warm Spring cañon. A leaching mill was erected to dissolve the copper in dilute sulphuric acid and precipitate it with scrap iron; but, before the process could be fairly tried, the plant was destroyed by fire and the property has since lain idle. The Coconino Copper Company acquired the ground adjoining the Petoskey mine in 1901. A reverberatory furnace was built at the mouth of Warm Spring cañon, but it was a failure on account of the highly silicious nature of the ore. Next a mill was erected to use the Neill process, but it also was unsatisfactory. Later the mill was leased by the Esmeralda Precipitation Company, of Chicago, which experimented for several months without productive results. At present the mine and mill are in possession of the Buckskin Mountain Copper Company which has renovated the mill with a view to active production if the experiments now being conducted are successful.

The district has always suffered from its isolated position, Marysville, 180 miles away, being the nearest railroad point, and from the silicious character of the ore for which a commercially successful leaching process has not yet been developed. Moreover, the limited quantity of ore available will require that any investment in plant be moderate in amount.

California.—As in 1906 the Mammoth and Mountain were the largest producers. The Balaklala failed to complete its smelting work, which proved to have been underfinanced, wherefore the company has had to raise more money. The Bully Hill made no production, its smelting works being still unfinished. They are expected to be in operation in the spring of 1908. The American Smelters Securities Company abandoned, temporarily at least, the construction of its San Bruno plant. The Mountain refinery was operated about nine months, being closed in the autumn. The Mammoth Copper Mining Company increased the capacity of its smeltery, and began the installation of a converting plant which will be in operation in 1908.

In 1907 the Mammoth Copper Mining Company was the larger producer of California, its matte being shipped to Salt Lake City for converting. The Mountain Copper Company was the only producer of blister copper. Other large producers were the Peyton Chemical Company, of Oakland, Cal., and the Great Western Gold Company, of Ingot, both of which shipped their matte to Salt Lake City; and the Penn Chemical Company, of Campo Seco, Calaveras county, which shipped matte to New York. The Arizôna-Mexican Smelting Company, of Needles, also shipped matte to New York. Among the counties, Shasta was the leading producer. Copper ore was also mined in Calaveras, San Bernardino, Amador, Eldorado, Inyo, Mariposa, Orange, Placer, Plumas and San Diego counties. The Mountain Copper Company in 1906 reported a profit of £94,948, against £158,168 in 1905. The sales of copper in 1907 were 3407 tons, against 2854 in 1906 and 4564 tons in 1905. The smeltery at Keswick was closed in 1905, and a new works, comprising an electrolytic refinery and a sulphuric acid plant, was erected near San Francisco. Phosphate rock also is acidulated at this plant. The injunction of the Government against smelting at Keswick having been lifted, operations have been resumed at that works.

The mines of the "Foothills Copper Belt," especially in Nevada, Placer, El Dorado, Amador, Calaveras, Tuolumne, Mariposa, Madera and Fresno counties were described by Herbert Lang in an elaborate series of articles in the *Engineering and Mining Journal* of Nov. 16, 23, and 30, 1907. In subsequent issues there were contributions by several writers in discussion of Mr. Lang's article.

Colorado.—The production of this State showed a considerable increase over 1906, the output in 1907 according to the State Commissioner of

Mines having been 11,256,291 lb., of which 5,366,759 lb. came from Leadville, and 2,572,764 lb. from the San Juan region, while the remainder was derived in small quantities from nearly every county of the State. It is believed that the San Juan region will make an increased production, the Red Mountain district promising especially well. Our own statistics, based on reports from all the smelters, give Colorado a materially larger production than the State Commissioner's.

Idaho. (By Robert N. Bell).—The amount of copper in ore produced in this State in 1907 was 10,620,000 lb.[1] The principal producer was the Snow Storm mine in the Coeur d'Alene, which turned out 95,435 tons of ore, of which 18,695 tons was oxidized ore assaying 2 per cent. and less, treated by the leaching plant at the mine, while the remainder was shipped to smelters for use as converter lining. The total production of the mine averaged better than 3.5 per cent. copper and 4.5 oz. silver per ton. The gross receipts for the year's shipment from this mine amounted to $1,623,689. The profit was $450,000 and the dividends paid amounted to $360,000. The Monitor mine, 10 miles east of the Snow Storm, shipped 500 tons of sulphide ore carrying 16 per cent. copper and about $5 gold per ton. This mine will be made much more accessible by the completion of the Chicago, Milwaukee & St. Paul railway which passes within half a mile of it. The mine is opened by a vertical shaft 400 ft. deep, from which four levels have been driven. At Loon creek, in Custer county the Lost Packer Mining Company operated successfully a 100-ton hot blast, pyritic smelter for 34 days during the summer, producing 425 tons of matte, assaying 51 per cent. copper, 10 oz. gold and 70 oz. silver per ton. This mine is situated 125 miles from railway, wherefore its operation is costly. Mackay is the nearest railway point. The Lost Packer ore is chalcopyrite in a gangue of quartz, together with some spathic iron. The principal ore shoot has been developed for a length of 500 ft. and to a depth of 500 ft. by a series of adit levels. The ore shoot varies in width from 1 to 8 ft. and averages 7 to 20 per cent. copper, 2 to 5 oz. gold and 7 to 15 oz. silver per ton.

At Mackay, the Macbeth Leasing Company and the Empire Copper Company operated the White Knob mine for eight months of the year, producing 2,750,000 lb. of copper. The ore of this mine is mostly of low grade, wherefore, operations were suspended in the autumn when the price of copper declined. The mine is in excellent condition and can be worked profitably when copper is at 15 to 16c. per pound.

The Weimer mine, in Fremont county, shipped several carloads of ore assaying 16 to 18 per cent. copper, the total shipments containing about 200,000 lb. of copper. Considerable low grade ore in bodies 20

---

[1] Our own statistics, based on smelters' reports, give a little higher total.

to 50 ft. wide was developed. The Copper Queen mine in Lemhi county made small shipments.

In the Seven Devils district, the Peacock mine shipped 500 tons of ore assaying 16 per cent. copper, while the Lockwood mine shipped 50 tons assaying 35 per cent. The completion of the extension of the Oregon Short Line through Snake River Cañon, which is now well under way, will put the Peacock mine within three miles of a railway. The Blue Jacket mine in the Seven Devils district was unwatered for examination, but the projected sale was not consummated.

At Cuddy Mountain, which is a bold uplift of porphyry, limestone and schist, 30 miles south of the Seven Devils district, there is a large area of monzonite, showing disseminated copper ore. The deposit makes a surface showing that is far superior to that of the Bingham monzonite. Another promising deposit of copper ore has recently been found at South Mountain in Owyhee county.

Maryland.—The Virginia Consolidated Copper Company did some exploration work in the Old Liberty copper mines, near Union Bridge, during the summer and fall of 1907 and conducted milling experiments to determine the value of its ore and method of treatment. These experiments proved satisfactory, although the ore is of extremely low grade. At present the company is not in a position to erect a suitable milling plant and operations have been temporarily discontinued.

Michigan.—Some of the mines of Lake Superior made slightly diminished outputs in 1907, but others increased. The net result was a small decrease in the total as compared with 1906. The intentional curtailment in this district in the last quarter was insignificant.

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1901	1902	1903	1904	1905	1906	1907
Adventure	....	606,211	2,182,608	1,380,480	1,606,208	1,552,628	1,244,872
Ahmeek	....	....	....	350,000	1,552,957	3,077,507	5,527,672
Allouez	....	....	....	....	1,167,957	3,486,900	2,934,116
Atlantic	4,666,889	4,949,368	5,505,598	5,321,859	4,049,731	1,439,082	*Nil.*
Baltic	2,641,432	6,284,819	10,580,997	12,177,729	14,384,684	14,397,557	16,704,868
Cal. & Hecla	82,519,676	81,248,739	76,490,869	80,341,019	83,812,370	94,529,821	88,055,723
Centennial	806,400	....	....	641,294	1,446,584	2,253,015	2,373,572
Champion	....	4,165,784	10,564,147	12,212,954	15,707,427	16,954,986	16,489,436
Franklin	3,757,419	5,259,140	5,309,030	4,771,050	4,206,085	4,571,570	4,401,248
Isle Royale	2,171,955	3,569,748	3,134,601	2,442,905	2,973,761	2,937,098	2,667,608
Mass	837,277	2,345,805	2,576,447	2,182,931	2,007,950	2,106,739	2,078,677
Michigan	....	166,898	275,708	2,746,127	2,891,796	2,875,341	2,665,404
Mohawk	160,897	226,824	6,284,327	8,149,515	9,387,614	9,352,252	10,107,266
Osceola	13,723,571	13,416,398	16,059,636	20,472,439	18,938,965	18,588,451	14,134,753
Phoenix	93,643	....	202,823	1,162,201	273,219	*Nil.*	*Nil.*
Quincy	20,540,740	18,988,491	18,498,288	18,343,160	18,827,557	16,194,940	19,796,058
Tamarack	18,000,852	15,961,528	15,286,093	14,961,885	15,824,008	9,832,644	11,078,604
Trimountain	....	5,730,807	9,237,051	10,211,230	10,476,462	9,507,933	8,190,711
Winona	....	101,188	1,036,944	646,025	*Nil.*	278,182	1,285,863
Wolverine	4,946,126	6,473,181	8,999,318	9,764,455	9,464,418	9,548,123	9,273,351
Victoria	....	....	....	....	....	....	1,207,237
Others	640,591	700,067	75,000	50,000	....	....	100,000
Totals	155,507,465	170,194,996	192,299,485	208,392,485	218,999,753	224,071,103	220,317,041

DIVIDENDS PAID BY MICHIGAN MINES.

Mine.	1905	1906	1907	Mine.	1905	1906	1907
Calumet & Hecla...	$5,000,000	$6,500,000	$6,500,000	Osceola..........	$384,600	$1,541,400	$1,249,950
Copper Range Con.				Quincy...........	600,000	1,250,000	1,485,000
Baltic..........	1,250,000	1,400,000	1,000,000	Tamarack.........	120,000	480,000	420,000
Champion...	1,000,000	1,200,000	1,000,000	Wolverine........	660,000	660,000	1,050,000
Mohawk..........	....	700,000	900,000				
				Totals...........	(a)$9,014,600	$13,231,000	$13,604,950

(a) Does not include dividends of $50,000 and $160,000 paid respectively by the Atlantic and Central companies.

In the following table the dividend disbursements are given for every half decade from 1850 to 1900, and subsequent years up to 1906:

Year.	Dividends.	Year.	Dividends.	Year.	Dividends.	Year.	Dividends.	Year.	Dividends.
1850	$84,000	1870	$700,000	1890	$3,415,000	1902	$3,440,000	1906	$13,231,000
1855	168,000	1875	1,920,000	1895	3,280,000	1903	4,980,000	1907	13,604,950
1860	120,000	1880	3,080,000	1900	9,811,200	1904	5,432,300		
1865	510,000	1885	1,970,000	1901	7,496,900	1905	9,224,600		

The Adventure Consolidated Copper Company in 1907 produced 1,842,145 lb. of mineral, yielding 1,244,874 lb. of refined copper for which was realized $220,409.75. The running expenses at the mine were $259,758.58; taxes, $7,932.43; smelting, transportation and all other expenses, $19,485.27; total, $287,176.28. Construction, development, and exploration accounts increased the deficit to $86,038.20.

The Ahmeek Mining Company recently issued a report for the period from Aug. 1, 1902, to Dec. 31, 1907. A total of 10,518,136 lb. of copper was produced of which 8,679,623 lb. was sold at an average of 17.63c. per lb., and 1,838,513 lb. remained unsold at the end of 1907. The running expenses at the mine were $1,111,358; smelting, transportation, commission and all other charges, $125,551; total, $1,236,909.

During the period of operations, 736,065 tons of rock were mined, of which 582,453 tons were stamped. From this 13,998,350 lb. of mineral were obtained, which contained 10,518,136 lb. of refined copper, an average of 75.138 per cent. of the mineral. Development consisted of 2644 ft. of sinking, including small vertical exploratory shaft; 313 ft. of cross-cutting; 15,801 ft. of drifting; total to Dec. 31, 1907, 18,758.6 ft. The Ahmeek property embraces 920 acres of mineral land, traversed by the following known veins: Allouez conglomerate, Kearsarge conglomerate, Osceola amygdaloid, and Kearsarge amygdaloid. Ground was broken for the present shafts late in 1903, and the first shipment of rock from the mine to the mill was made in May, 1904. The ore has been stamped on contract at the Osceola and Tamarack mills. During 1907 it was all treated at the Osceola mill. In the future it will be necessary to build

a stamp mill. On Dec. 31, 1907, No. 1 shaft was 1204.2 ft. deep on the incline. Excepting where the ground is poor a system of drift stoping has been adopted for all new workings. But little poor ground has been encountered in the entire mine.

The Allouez Mining Company in 1907 hoisted 227,481 tons of ore, of which 12,761 tons were discarded. The mill treated 214,720 tons of ore, producing 4,616,180 ℔. of mineral, from which was obtained 2,934,116 ℔. of refined copper, or 63.561 per cent. copper from the mineral and 0.683 per cent. from the ore. The cost of mining was $264,833; sorting, $12,990; transportation, $35,769; milling, $65,191; surface and incidental expenses, $23,192; smelting, freight and marketing copper, $48,887. The sales of copper were 2,934,116 ℔. at an average price of 18.9916 per ℔.

The Calumet & Hecla Mining Company in the year ended April 30, 1907, produced mineral equal to 46,297 tons refined copper, as against 43,652 tons in the previous year. The product of refined copper was 46,949 tons. For the previous year the refined copper was 50,515 tons. There were produced from the Osceola lode 6,892,548 ℔. of copper during the year. Excellent progress is being made in opening the Kearsarge amygdaloid. The ground thus far opened is promising. The irregular occurrence of pay ore in the Kearsarge lode, and the low average of the copper content, make it desirable to operate on as great a length of the lode as is practicable. In view of this and of the large stake in this lode in the LaSalle company's operations, the Calumet & Hecla directors determined that it would be best for their company to acquire interests in the Osceola, Allouez and Centennial companies, in the lands of which companies copper in paying quantities had been discovered in the Kearsarge lode in a tract of several miles. The contiguity of the lands of these companies with each other and their vicinity to the Calumet & Hecla mine and the contiguity of the land of the Osceola to that of this company, and of the LaSalle company, also seemed to offer the opportunity of economical coöperation in mining between these several companies. Induced by these considerations, the Calumet & Hecla purchased 42,978 shares of the Allouez, 46,080 share of the Centennial, and 22,671 shares of the Osceola companies, at a total cost of $7,389,267. The Calumet & Hecla now owns 42,978 shares of the Allouez Mining Company, 100,000 shares issued; 46,080 shares of the Centennial Copper Mining Company, 90,000 shares issued; 20,000 shares of the Frontenac Copper Company, 20,000 shares issued; 50,100 shares of the Gratiot Mining Company, 100,000 shares issued; 161,750 shares of LaSalle Copper Company, 302,977 shares issued; 18,000 shares of the Manitou Mining Company, 20,000 shares issued; 22,671 shares of the Osceola Consolidated Mining Company, 96,150 shares issued; 50,100 shares of the Superior Copper Company, 100,000 shares issued.

The Centennial Copper Mining Company in 1907 hoisted 225,271 tons of ore, of which 200,040 tons were treated in the mill, producing 3,604,970 lb. of mineral, from which was obtained 2,373,572 lb. of refined copper. The percentage of mineral obtained from the ore treated was 0.901, the refined copper in the mineral was 65.842 per cent., and the refined copper per ton of ore was 0.593 per cent. The working expenses at the mine were $433,727; smelting, freight and marketing copper, $36,255; construction and equipment at the mine, $64,193; total, $534,176. The price received for refined copper during the year averaged 18.4639c. per lb.

The Franklin Mining Company in 1907 hoisted 388,165 tons of ore and milled 383,290, producing 8,547,696 lb. of mineral and 4,401,248 lb. of refined copper, of which 2,020,070 lb. remained unsold at the end of the year. The copper sold (2,381,178) realized an average of 19.377c. per lb. The ore milled yielded an average of 11.48 lb. of refined copper. The cost of mining per ton of ore was $1.1622; surface expense, $0.0678; milling, $0.3258; transportation, $0.0948; rock house expense, $0.0549; general expense, $0.0169; office expense $0.0127; construction, $0.0478; total, $1.7829. The average number of men employed was 575.

The Isle Royale Copper Company in 1907 hoisted 227,688 tons of ore and milled 175,450, which yielded 3,614,799 lb. of mineral and 2,667,608 lb. of refined copper or 15.2 lb. per ton of ore milled. The cost of mining and milling per ton of ore milled was $2.08. The cost per pound of refined copper at the mine, excluding construction, was 13.70c.; for construction, 1.05c.; for smelting, freight, marketing, etc., 2.34c.; total, 17.09c.; explorations, railroad extension, etc., 3.17 c.; grand total, 20.26c. The cost of milling was 32.65c. per ton.

The Mohawk Mining Company in 1907 raised 744,361 tons of ore, and milled 640,777 tons, which yielded 13,164,360 lb. of mineral and 10,107,266 lb. of refined copper, an average of 15.77 lb. per ton of ore milled. The cost per ton of ore hoisted was $1.326; per ton of ore milled, $1.54. The cost of refined copper at the mine was 9.764c.; the cost of refined copper at the mine was 9.764c.; cost of smelting, freight, marketing, etc., 0.961c.; total, 10.725c.; total, including construction, 11.747c. The production of copper was sold at an average of 15.66c.

Osceola: In 1907 the mills stamped 811,603 tons of ore, yielding 18,607,747 tons of mineral, from which was obtained 14,134,753 lb. of refined copper. The cost of mining and milling was $1.84 per ton of ore. The total cost per pound of refined copper was 12.44c. against 10.89c. in 1906.

Quincy: The production in 1907 was 31,339,170 lb. of mineral, yielding 19,796,058 lb. of refined copper, for which was realized $3,717,501. Mining cost $2,012,085; smelting, transportation, etc., $164,289; total $2,176,374. The cost of production per pound of refined copper was about 11c. for direct operating expense, and 12.13c. for all expense,

including construction. Mining and milling cost about \$1.90 per ton of ore; construction 19c.; smelting, refining and marketing, 15c.; total \$2.24.

The Tamarack Mining Company in 1907 milled 533,600 tons of ore, which yielded 17,071,730 lb. of mineral and 11,078,604 lb. of refined copper. The sales during the year were 6,931,397 lb., leaving 4,147,207 lb. unsold at the end of the year. The cost of mining and milling was \$2.98 per ton, the cost of milling alone being \$0.2812. The cost of refined copper at the mine, excluding construction was 14.33c.; construction amounted to 0.0084c.; and smelting, freight, eastern expenses, interest, commissions and all other charges, 1.49c., making the total cost per pound of refined copper 16.66c. The percentage of refined copper per ton of ore milled was 20.8; per ton of mineral produced 64.89c. The amount of ore mined during the year was 659,821 tons. The average number of men employed per month was 1479. The high cost of production in 1907 was due to decreased grade of the ore and increased cost of labor and supplies.

The Victoria Copper Mining Company in 1907 hoisted 104,783 tons of ore and milled 95,035 tons, which yielded 2,062,210 lb. of mineral and 1,207,337 lb. of refined copper or 12.7 lb. per ton of ore milled. The cost per ton of ore milled was \$1.745. The cost per pound of refined copper at the mine was 13.778c.; cost of smelting, freight and marketing, 2.019c.; total cost per pound of refined copper, 15.797c. The amount of copper sold was 949,801 lb., leaving 257,436 lb. unsold at the end of the year.

The Wolverine Copper Mining Company in the year ending June 30, 1907, hoisted 359,915 tons of ore and milled 344,062, which yielded 12,523,210 lb. of mineral and 9,372,982 lb. of refined copper, an average of 27.24 lb. per ton of ore milled. The cost per ton of ore hoisted was \$1.58; per ton of ore milled, \$1.65; per pound of refined copper at the mine, 6.076c. The cost of smelting, freight and marketing product was 1.062c. The total cost per pound of refined copper, including construction, was 7.587c. The production of copper was sold at an average of 21.36 cents.

(By C. L. C. Fichtel.) A great deal of construction and development was undertaken and several new mines were opened and operations were resumed at abandoned properties. The year was free from any serious labor troubles. There were 19 producing mines in 1907, compared with 20 in 1906. The Atlantic did not produce owing to the caving in of the old workings. The largest individual gain was made by the Quincy, which returned to its former place as the second largest producer on the Lake. The greatest losses were recorded by the Wolverine and Mohawk mines, due partly to accidents at their mill. In the last quarter of the year wages were reduced to about the scale which prevailed in 1906.

There was considerable mining activity in Keweenaw county, and

several new mines were opened. On the land of Keweenaw Copper Company the Medora shaft was sunk to a depth of 950 ft. where the formation was found to be less fissured than higher up. There are now about 4000 ft. of openings, and a good grade of soft amygdaloid stamp rock, which is easy to mine and mill, has been disclosed. The acquisition of the Phœnix property in April gave this company ample stamping facilities for some time to come, and the Keweenaw Central Railroad has been in operation from the Mandan location to Mohawk. Work of grading from Mohawk to Calumet will probably be completed by June 1, 1908.

The Calumet & Hecla suspended operations on its Manitou and Frontenac properties after considerable exploratory work and diamond drilling. The showing is sufficient to warrant opening the property in a systematic manner and this will probably be done in the spring of 1908.

The Tamarack company developed the Cliff property in a systematic manner. The new shaft is being sunk and all necessary surface equipment is in place.

The Ojibway company, since its formation in March, sank two shafts; one is down 250 ft. but the other was started only a short time ago. These shafts are being sunk behind the foot-wall. In sinking No. 2, the first of the openings, a fissure vein was encountered, carrying good milling ore, with some shot copper. Diamond drill cores taken from this property were heavily charged with copper. The company is at present without railroad facilities, but surveys have been made for a line.

At the Mohawk mine all five shafts were in operation and sinking is going on in all of them. During 1907 the heads at the mill were changed from simple to compound, and the capacity was thus raised to about 3000 tons of ore per day.

The Ahmeek is the best equipped small mine on the Lake. A good deal of construction work was completed and the underground conditions are now in fine shape. Between 800 and 1000 tons of ore were shipped daily to the Tamarack mills.

The Allouez Mining Company came under the control of the Calumet & Hecla and development is progressing systematically. All ore came from No. 1 shaft and amounted to about 700 tons per day. No. 2 shaft is down more than 1000 ft., but has not yet cut the lode. It is being sunk at an angle of 80 deg., but will turn and follow the lode.

The Calumet & Hecla assumed the management of the Centennial and Allouez properties and at the same time purchased stock in the Osceola, but an injunction obtained by the management of the Osceola restrained the Calumet Hecla from voting the stock acquired. This case has not yet been settled.

At the main mine of the Calumet & Hecla all shafts were operated and the output of ore was fully maintained and in some instances increased.

On the Osceola amygdaloid all the shafts were in operation and a new shaft, No. 18, was started at the extreme north of the property. Shafts No. 19 and 20 on the Kearsarge lode were closed down and it is likely that this lode will be worked through drifts from the Osceola amygdaloid shafts or from the Centennial. Preparations are being made for a new electrical pumping station at No. 5 shaft, which will take care of all the water from the north end of the mine and eliminate water hoisting at the Red Jacket shaft. This will give another compartment for hoisting ore, which will probably be used to take care of the ore coming from the new sub-shaft. New construction and equipment included a foundry and pattern shop; recrushing mill, containing 175 tables and 48 Chilean mills; one 2000-kw. generator; two boiler houses and various other improvements. During the year Walter Fitch was appointed superintendent and John Knox succeeded J. B. Risque as head mining captain.

The Tamarack company extended its drifts to the extreme limits of its property and stoped back, allowing the ground to cave behind the workings, thus eliminating the cost of timbering. This, with the addition of electrically operated locomotives for tramming, which probably will be installed in 1908, should greatly lessen the cost of production. The Osceola showed a slight falling off in production due to the curtailment in October and November. Production was again nearly normal during December. The North Kearsarge branch was again opened after having been closed for some time because of the fire. The South Kearsarge maintained about normal production.

The Superior sunk below the 8th level. Drifts on the Baltic lode from the 6th and 7th levels encountered good milling ore, with some shot copper. This company has now approximately 4000 ft. of openings.

On the Quincy property, No. 8 shaft, on the Mesnard tract, showed good milling ore, and drifting was extended to the southern boundary of the newly acquired Arcadian land. This tract carries the outcrop and dip of the Pewabic lode for nearly 1½ miles. Some diamond drilling has been done and No. 9 shaft will be sunk here. Work at the Hancock consisted largely in cleaning out and retimbering the old workings and sinking the new (No. 2) shaft, which has now attained a depth of 550 ft. A new hoisting plant and compressors were installed. Isle Royale spent most of the year in sinking shafts and other development work.

The three companies which comprise the Copper Range Consolidated Mining Company had a prosperous year. The Champion and Baltic increased their production and more than offset the slight falling off at Trimountain. At the Globe tract the shaft has passed through the overburden and is well into the lode. The company experienced great quicksand difficulties in sinking this shaft, but these were overcome, and in June the lode was struck, after nearly two years of work.

The La Salle company is capitalized at 400,000 shares, 163,000 of which are owned by the Calumet & Hecla, and 97,000 are in the treasury. The company has $1,000,000 cash in the treasury and an agreement that the controlling company will furnish an additional $750,000 if necessary, which is to be paid back from the earnings of the company. Developments on the Tecumseh tract of this property are encouraging.

The feature of 1907 in Ontonagon county was the disclosure on the Lake property of a highly mineralized formation, 35 to 40 ft. wide, which proved to be the Baltic lode. Development was begun and the lode was opened by trenching, and was found to be rich. A shaft was started and is progressing satisfactorily. A diamond-drill core, taken from the shaft-site showed the same mineralized formation at a depth of 450 ft. This discovery has greatly encouraged property owners in Ontonagon county.

Missouri.—The North American Lead Company, of Fredericktown, on the southern edge of the disseminated lead district, made a considerable output of copper in 1907, which appears in the statistics credited to "Other States." This company operates an electrolytic refinery and marketed nearly all of its product as refined metal. An increased production from this source is to be expected in 1908. Besides copper, the North American Lead Company produces nickel and cobalt, both of which occur in association with the copper.

Montana.—The great decrease in the copper production of this State was due, in the first place, to the restriction of operations by the subsidiaries of the Amalgamated Copper Company in October, and finally by the closing of the Washoe smelter in December, which cut off the North Butte and Butte Coalition companies from a market for their ores. The closing of the Washoe works was to concentrate the curtailed production of the Amalgamated to the mines and works of the Boston & Montana Company, which could be operated more economically at full capacity than all the mines and works at reduced capacity. The Boston & Montana mines were then operated with two crews of men, each working half a month. This employed practically the whole of the best miners of the Amalgamated companies and preserved the organization so that operations could be resumed immediately at full capacity when desired.

The suspension of Butte was largely to correct the labor difficulties which had arisen. During the great activity in 1906-1907 the demand for men was such that many incompetent miners secured employment, and wages were raised to $4 per eight hours, the companies entering into a contract with the miners' union to maintain this rate so long as the price for copper should be 18c., or over. With this increase in wages and at the same time a reduced efficiency from the men the cost of production at Butte rose to an alarming figure. In 1905 the cost of mining per ton of ore was about $3.50, and the cost of carriage to the reduction works,

concentration, smelting, and refining was about \$2.50, making a total of about \$6 per ton of ore. In the case of ore yielding 60 lb. of refined copper per ton, which was approximately the average of the Anaconda company in 1905, the cost per pound of copper was a little more than 10c. Out of the mining and smelting cost of \$6 per ton about \$3.50 is due to labor. Consequently, the advance in wages from \$3.50 to \$4 per day, increased the cost of production by about 50c. per ton of ore, which is a little more than 0.8c. per lb. on 60 lb of copper, even if the efficiency of the labor had remained the same, but inasmuch as in general the efficiency of labor decreases with increase in wages, and this was distinctly the case at Butte, the cost of producing copper actually rose to 12c., or more.

When the price for copper appeared likely to decline to 15c. there was considerable doubt whether the union would stand by its contract, but in October the serious position of the copper industry became so clear that none could fail to recognize it, and when the reduction to the old rate of wages was made toward the end of October it was accepted without opposition. The suspension of operations in December by all of the large companies, except W. A. Clark, enabled the inefficient men to be weeded out, and when work was resumed at the end of March, 1908, the men retained were given to understand that they had to work harder than formerly, wherefore it is expected that copper will now be produced at Butte at 10@11c. per lb., as it was previous to 1906. Thus a very difficult labor situation was ably handled by the managers of the companies, led by the Amalgamated.

The report of the Amalgamated Copper Company for the fiscal year ending Apr. 30, 1908, showed net income of \$6,680,557, against \$14,154,400 in the previous year. The net income in the fiscal year 1906-1907 was equivalent to a little more than $4\frac{1}{4}$ per cent. on the \$153,887,900 outstanding capital stock of the company. The output of the Washoe and Great Falls works was 212,000,000 lb. in the calendar year 1907, of which the production of the Amalgamated companies was 178,000,000 lb., the remainder being derived from custom ores. The Amalgamated smelted the entire output of the North Butte and the Red Metal companies, besides which it obtained considerable supplies of ore from smaller companies. The combined production of the Washoe and Great Falls works was just 24 per cent. of the total production of copper in the United States in 1907, while the production of the constituent companies of the Amalgamated Copper Company was a little more than 20 per cent. of the total. These figures are far in excess of those of any other copper producing company of the United States. The Washoe works is the largest single producer of copper in the United States. The second place is held by the Great Falls works, and the third place by the Douglas works of the Copper Queen company, but between these two the difference is small.

The Anaconda Copper Mining Company in 1906 produced 1,450,601 net tons of ore. The reduction works treated for all companies 3,006,910 dry tons of ore and other cupriferous material. From the ore treated for the Anaconda company there was produced 94,963,835 lb. of copper, 2,979,908 oz. of silver and 15,985 oz. gold. The gross earnings were $26,968,871 and the expenses were $18,384,702. The company treated at the reduction works an increased amount of ore for other companies, thus reducing the cost of treating the product of its own mines. On account of the high price for copper a greater proportion of lower-grade ores was mined, although the reserves of the higher-grade ores were increased largely. Through the economies introduced in the operation of mines and works and the higher price for copper, it was possible during the year to mine at a profit ore as much as 1 per cent. lower in copper than the lowest grade considered profitable during the last few years. It was determined that the working of the tailings in the dump of the old works by leaching and precipitation can be done at a good profit. With any reasonable price for copper, the yield from working the tailings should be large for many years to come. The product of copper, silver and gold in 1906 sold for $20,955,532. The total quantity of cupriferous material treated in the reduction works, was 3,006,910 dry tons. The material treated for the Anaconda was 1,494,828 tons ore; 18,496 tons of slimes; 3324 tons flue-dust; 166,516 tons slags; 654 tons miscellaneous cleanings from old works; a total of 1,683,818 tons. The average yield per ton of ore was 63.5 lb. copper, 1.99 oz. silver, and 0.011 oz. gold per ton. An increased product of arsenic was recovered from flue dust.

In 1907 the Anaconda mines produced 1,123,692 tons of ore at a cost of $5,241,704 for mining. The reduction works treated 2,582,611 tons, of which 1,153,645 was for the Aanconda company, the cost of treatment being $3,640,296. The yield was 63,055,661 lb. of copper, 2,001,351 oz. silver, and 8290 oz. gold.

(By Edwin Higgins.) The later part of 1906 witnessed the advent of an unprecedented number of new mining companies in Montana and activity continued well into 1907. Most of the new companies being in the development stage, their operations had little or no effect on the production of 1907. The high price of copper stimulated exploration in every mining district of the State and many abandoned silver mines, the ores of which carry copper in commercial quantities, with that metal selling above 20c. per lb., were reopened and equipped for operation on a large scale. One of the noteworthy features of 1907 was the gradual change in the development of power for mine and mill operations. Steam is rapidly being discarded in favor of electric power, especially in the Butte district. At the close of 1907 the larger Butte operators were driving all compressors and pumps electrically and this same power had been

installed in several of the mines for surface and underground haulage. In the other districts of the State many mining plants and mills are using electrical power and it will be only a short time before it will come into general use, except in remote places.

The Butte district produces from 80 to 85 per cent. of the ore mined in the State. In the early part of 1907 there were employed in the mines of the district 16,000 men and the average monthly pay roll was close to $2,000,000. The tonnage mined by the large companies was as follows:

Amalagated, North Butte and Red Metal companies, 10,000 tons of ore per day, treated at the Washoe smelter, Anaconda; Boston & Montana, 3000 tons treated at the Boston & Montana works, Great Falls; the Clark mines, 1200 tons, smelted at the Butte Reduction Works, Butte; other companies and leasers, 2000 tons of ore per day, shipped to the works previously named and to the East Helena plant of the American Smelting and Refining Company; total, 16,200 tons of ore per day.

During January and February the extremely cold weather, coupled with shortage of cars, caused a loss of 20 days of actual mining operations. However, for the first six months of 1907 the production of the mines was about normal and equal to that of the corresponding months of 1906.

On April 1 the miners' wages were raised from $3.50 to $4 per day, the Amalgamated and other large companies entering into a five-year agreement with the Butte Miners' Union to pay the $4 scale as long as copper sold as high as 18c. per lb., the old scale, $3.50 per day, to be resumed when the metal sold below 18c. On Aug. 1 the machinists, who had been receiving $4.50 per day, struck for $5 per day. They were repeatedly offered $4.75 but would not accept, and it was not until Dec. 1 that they returned to work. By that time they were glad to get $4.50 per day. In the meantime, under the five-year agreement, the miners had been reduced to $3.50 per day, the new scale going into effect Nov. 1.

On Sept. 16, orders were given by the larger companies to curtail production to 40 per cent. of the normal. By the end of November all the smelters of the State had decided to accept no more custom ore for treatment, except the East Helena smelter, which continued to accept ore from mines with which it had contracts. This was a severe blow to the smaller operators and leasers, not only in Butte but throughout Montana.

On Dec. 10 it was decided to close the Washoe smelter and all the mines of the Amalgamated, North Butte and Red Metal companies, except the mines and smelter of the Boston & Montana company (a subsidiary company of the Amalgamated). This company worked to full capacity, mining and shipping 3700 tons daily to the Great Falls smelter. Toward the end of December the production of ore in the Butte district had dropped to about 5000 tons per day. With the exception of about 3000

tons produced by leasers and small operations, this tonnage came from the Boston & Montana company's mines and those of W. A. Clark. Many of the new companies which had started operations earlier in the year with such bright prospects were forced to suspend on account of a lack of funds with which to pay their men.

A matter of great importance in connection with the mining industry of the State is the attitude of the ranchers of the Deer Lodge valley toward the Washoe smelter at Anaconda. These ranchers ask for an injunction closing the smelter and for the payment of damages to livestock and land. A vast amount of trouble and expense has been incurred by the Anaconda company in the preparation and submission of its side of this now famous case. Master-in-Chancery Crane, before whom the case was argued, gave an opinion in 1907 to the effect that the sulphur in the smoke causes no damage to animal or vegetable life, but that the arsenic is damaging to both; and that if the injunction were granted and the smelter were closed the damage to the ranchers would be greater than if the plant were allowed to continue in operation. A stronger opinion in favor of the company has been given by Judge Hunt, of the Federal court, and it is probable that this vexatious litigation will be concluded in such a way that the Washoe works will continue to operate as usual.

Nevada.—The Steptoe Valley smelting works were delayed in construction, wherefore the mines at Ely did not become producers in 1907. Considerable prospecting work was done in other parts of the State, particularly near Luning, where there are some good surface showings. No mine has yet been developed, although the district has been known for the last 30 years. There was considerable activity in developing mines in the Yerington district, and several lots of rich ore were shipped from there by the Nevada-Douglas company. Operations came largely to a standstill in the autumn. Some prospecting was done at Ubehebee, 60 miles west of Bonnie Clare where there are said to be good surface indications.

The mines of Ely, together with the Steptoe Valley mill and smelter, and the productive and earning capacity of the several companies were described in *Eng. and Min. Journ.*, Oct. 12, Oct. 19 and Nov. 2, 1907. In these articles it was estimated that the mines of the Nevada Consolidated Copper Company had positive and probable ore to the amount of 10 or 11 million tons while the Cumberland Ely had an orebody not fully developed, which was considered likely to prove to be of several million tons. The ore of the Ruth mine (Nevada Consolidated) was reckoned as concentrating in the ratio of 7:1; of the Eureka mine (Nevada Consolidated) 10:1; and of the Veteran mine (Cumberland Ely) 3½:1. The plans of the two companies contemplate the milling of 2000 tons daily of Eureka ore and 1000 tons each of Ruth and Veteran. The cost of pro-

duction to the Nevada Consolidated was estimated at a little over 7c. per lb., basis New York, and to the Cumberland Ely a little over 8.5c. per lb. The productive capacity of the Nevada Consolidated, on the basis of plans then being carried out, was estimated at 39,817,000 lb. per annum; of the Cumberland Ely, 16,000,000 lb.; and of the Giroux, 8,000,000 lb.

In comparing Bingham and Ely, it was remarked that all of their production would be comparatively cheap copper, the probable cost to the two companies of Bingham being in the neighborhood of 7c. per lb., while the cost for a large part of the production at Ely will be about the same, for a part a little higher. In many respects the conditions at the two districts are similar. In both cases the ore is chalcocite and chalcopyrite finely disseminated in monzonite; in both cases the ore bodies are of immense size; and in both cases the conditions are favorable to cheap mining. The two mines at Bingham will produce 10,000 tons of ore per day; the four mines at Ely will produce 4500 tons per day. The ore of Ely will assay 2.2 to 3 per cent. copper; that of Bingham will go about 1.8 per cent. The disadvantages of the lower grade of the ore at Bingham, and the finer dissemination of the mineral (which leads to a lower percentage of extraction in milling) are offset by the ability to mine more cheaply, a larger proportion of the ore being capable of extraction by steam shovel than at Ely. In so far as ore reserves are concerned, the tonnage of developed ore at Bingham is far superior to that at Ely, although the orebodies at the latter place are gigantic. In this comparison it is merely a difference between giants, in which Bingham overtops all others, even after allowing for the prospective resources of Ely. The mines of Ely are certainly great, but those of Bingham are greater. It is evident that the two districts will make an immense addition to the world's supply of copper and will have an important influence upon the market for the metal.

The official report of the Nevada Consolidated Copper Company for the year ended June 30, 1907, contained a condensed report of Pope Yeatman, consulting engineer of the company, under date of Oct. 1, 1907, as follows:

"The area of the Nevada Consolidated Copper Company amounts to about 850 acres, consisting of 63 claims. It is divided into two general sections, the Eureka group, comprising 27 claims, and the Ruth group of 36 claims. The copper ore occurs mainly as chalcocite as a secondary enrichment; this chalcocite, in the form of specks, veinlets and bunches, being in porphyry, of which there are large intrusive masses in both the Eureka and the Ruth groups. Besides copper in the form of chalcocite, there is also some chalcopyrite. Iron pyrites is associated with both minerals.

"The orebodies in the Eureka and Ruth groups have been developed

and show ore blocked out by shafts, levels, raises, and drill-holes, aggregating 14,432,962 tons, with an average content of 40 lb. copper per ton, and some gold and silver, for which an extraction of 15c. per ton of ore is estimated.

"Outside of the developed ore bodies there are great possibilities. These are indicated: (1) by the typical porphyry croppings; (2) by underground workings, which have not yet been carried out beyond the limits of the ore; (3) by development by means of drill-holes, which have penetrated orebodies.

"The most important of these possibilities are: (1) a new orebody in the Ruth section, very promising because of the fact that the crosscuts of the Ruth are still in payable ore. (2) A large mass of copper-bearing porphyry, a portion of which has been prospected by drill-holes, showing ore averaging about 2 per cent. (3) The extensions of the Eureka, where borings with the diamond drill have disclosed considerable bodies of good ore; and probably the most important of all these possibilities is an orebody, lying adjacent to the main Eureka orebody and opened up by drill-holes. This shows a thickness of ore averaging 142 ft., a possible length of 2200 ft. and an average assay value of 2.42 per cent. copper.

"The Nevada Northern railway furnishes ample transportation facilities for cheaply taking the ore from the mines to the concentrator and smelter of the Steptoe Valley Smelting and Mining Company, in which the Nevada Consolidated Copper Company owns a half interest. One-half of this plant is for the use of the Cumberland-Ely company, and the other half for the Nevada Consolidated. There is an abundance of water brought from Duck creek by a pipeline having a capacity of 1200 cu.ft. per minute. The smelting plant will consist of roasters, reverberatories, blast furnace and converter plant. A thoroughly modern and up-to-date power plant, to furnish power for the smelter, concentrator and mines, will have a capacity of about 6500 h.p."

The Giroux Consolidated Mines Company put its concentrating mill in operation for a test run in October, and determined that a high percentage of mineral can be extracted from the milling ore. Regular operation was prevented by the caving in of the Alpha shaft, from which it was expected to obtain the water supply for the mill. Also, the financial resouces of the company had become nearly exhausted, and early in 1908 a bond issue was made. With the proceeds from that it is planned to install a pumping plant and pipe line to take water to the mine from the north fork of Cleveland creek, in Steptoe valley. According to E. W. Walter, general manager of the company, the block of milling ore developed by the workings from the Morris, Brooks and Bunker Hill shafts is 200 ft. long, 160 ft. thick, and 150 ft. wide, with ore still showing in most of the crosscuts. As it stands it contains approximately 4,000,000 tons of ore, with

an average copper content of not less than 2.25 per cent. and 20c. gold per ton. A large proportion of this orebody will average better than 3 per cent. copper.

In the workings from Giroux, Taylor and Alpha shafts there is blocked out about 350,000 tons of smelting ore, averaging 6.5 per cent. copper. The Alpha shaft is now sunk to a depth of 1100 ft. Ore was first encountered at the 1000 ft. level, the ore in the vein above that point being completely leached out. At 1100 ft. the vein is still oxidized, but it is expected that sulphide ore will be encountered at 100 to 200 ft. greater depth.

According to E. P. Jennings[1] the Yerington copper deposits in Lyon county, 40 miles southeast of the famous Comstock lode, occur in garnet rock as an impregnation throughout portions of its mass and as richer concentrations in fractures and shear zones; also as bedded veins in marbleized limestones. The effect of folding and fracturing of the strata has been to localize the deposition of the ore in the garnet rock along the fracture zones, giving the deposits the appearance of fissure veins filled with chalcopyrite disseminated in a garnet or garnet-epidote gangue. On each side of these fissures the garnet rock is more or less impregnated with chalcopyrite. The marbleized limestone which, in places, covers the garnet rock, contains bedded veins filled with oxidized copper ore in a quartz and calcite gangue. At water level, which in one case occurs at depth of 600 ft., pyrite and chalcopyrite begin to appear, replacing the oxidized ore. The mines have not been sufficiently developed to show the relation of the ore in the limestone with the ore of the garnet shear zones, but that there is some connection is probable.

The original minerals of the ore deposits are chalcopyrite and pyrite which have been oxidized and partly leached, forming in the upper enrichment zone, malachite, azurite and chrysocolla, and covellite as a coating, or chalcopyrite in the upper sulphide zone. Bornite and chalcocite occur as secondary sulphides in a few instances. The district is being developed rapidly and in two years' time it is expected to become a copper producer on a considerable scale.

New Mexico (By Charles R. Keyes). — During 1907 the principal developments were in the Burro mountains, in Grant county, where the Phelps-Dodge interests opened promising properties. In the Pyramid mountains, a few miles to the south, some important deposits have been developed. Other ranges of this region are shipping considerable quantities of good ore. Farther to the north and northwest, in the Mogollon mountains, there was more or less activity, but the lack of adequate railway facilities has been a serious drawback. In the neighborhood of Oro Grande, 75 miles north of El Paso, a large new smeltery has lately

[1] *Journ.* Can. Min. Inst., 1907, Vol. X.

been put in operation. At San Pedro, in Santa Fe county, the copper mines and smelter resumed active operations after several years of idleness. Magdalena continues to ship some copper ore in connection with its zinc ore. There are in southwestern New Mexico, besides the purer oxidized ores and pyritic ores which are chiefly mined at the present time, large deposits of complex ores containing copper, lead, zinc and iron sulphides in about equal proportions, with some gold and silver, which will no doubt come into market some day and materially increase the copper output of the region.

(By R. V. Smith.) Several of the copper smelteries in New Mexico were enlarged in 1907 and new ones were built. A number of concentrating mills were installed, and magnetic separation of copper ore from pyrites was undertaken on a large scale in one mill. Three copper leaching mills were run experimentally on oxidized ores.

North Carolina.—This State made a small output of copper in 1907, the Union mine at Gold Hill, Rowan county, being the principal producer. Its ore was shipped partly to the Tennessee Copper Company at Isabella, Tenn., and partly to the American Smelting and Refining Company at Baltimore and Perth Amboy. The ore contains 3.5 to 4 per cent. of copper, is highly silicious and is used by the smelters for converter lining. During 1907 the property was transferred from the Union Copper Mining Company to the Union Copper Mines Company.

(By Richard Eames, Jr.) Within a radius of 100 miles, Salisbury being the center of the mineral belt, I have examined more than 200 localities showing copper ore. For most part the veins are quartz in Huronian slates. The ore is pyrite, with the resulting carbonates and silicates, accompanied by gold and silver.

The Union mine has shipped in the last two years not less than 14,000 tons of ore, on which the freight and smelter charge has been approximately \$10 or \$12. This ore has probably averaged 4 per cent. copper and \$2 gold and silver. The Gold Hill copper mines have struck, on the 800-ft. level, a new vein of auriferous pyrites, which is producing paying gold ore, while the concentrates are being accumulated for shipment. In these mines there are thousands of tons of low grade copper-gold ore. The Ashboro copper mine has shipped about 200 tons of ore which ran 8 per cent. copper and \$4 in gold and silver. It has a quantity of low-grade exposed, in a strong vein, fully 8 ft. in width. Near this mine is a large belt of copper-bearing strata worthy of attention.

Ore Knob, in Ashe county, has a record of several thousand tons of 6 per cent. ore. Here, too, there is a large reserve of low-grade pyrites. Several other localities in the same county have ore worthy of development. The Copper Knob mine made one shipment of picked ore that ran 25 per cent. copper with \$150 per ton in gold and silver. The Hercules

Gold and Copper Company is blocking out ore. The Conrad Hill, in Davidson county, has a large family of veins of low-grade gold-copper ore. The Salisbury mine, in Rowan county, is a remarkable deposit of rich ore on the surface. The Virgilina district of North Carolina and Virginia has about 20 places that have produced high-grade shipping ore.

Oregon.—The Takilma Smelting Company of Takilma, and the Oregon Smelting and Refining Company, of Sumpter, made small outputs of matte in 1908, all of which was shipped to Tacoma for converting.

Tennessee.—The Ducktown district in 1907 made about the same production as in 1906. It could have been materially increased, but movement in that direction was deferred owing to the adverse condition of the metal market which developed in the second half of 1907. As in previous years, the only producers were the Tennessee Copper Company and the Ducktown Copper, Iron and Sulphur Company. The former produced pig copper, while the latter shipped its product in the form of matte, in the early part of the year to the Tennessee Copper Company and in the latter part to outside smelters. The Tennessee Copper Company erected a sulphuric acid plant to utilize the gases from its pyritic smelting furnaces. This, together with other technical developments, is described by J. Parke Channing in subsequent pages of this volume.

STATISTICS OF TENNESSEE COPPER COMPANY

Items	1906		1907	
	Per Ton Ore.	Per lb. Copper.	Per Ton Ore.	Per lb. Copper.
Mines development	$0.1067	0.343c.	$0.1318	0.407c.
Mining, hoisting, etc.	0.7817	2.512c.	0.9389	2.904c.
Crushing and sorting	0.0693	0.227c.	0.0804	0.249c.
Railway	0.1389	0.438c.	0.1329	0.411c.
Blast furnace	1.4864	4.765c.	1.6219	5.016c.
Engineering and laboratory	0.0370	0.118c.	0.0628	0.194c.
General	0.1387	0.445c.	0.1703	0.526c.
Converting	0.2733	0.876c.	0.2402	0.743c.
	$3.0320	9.724c.	$3.3792	10.450c.
Adjustment of ore account	0.0013	0.004c.	0.0045	0.014c.
Cost of fine copper in pig	$3.0307	9.720c.	$3.3747	0.463c.
Freight, insurance and selling		0.68		0.68
Taxes and all other expenses		0.51		0.67
Total cost per lb. copper		10.91c.		11.79c.

The Tennessee Copper Company in 1907 produced 383,631 tons of ore. At the end of the year the ore developed in the Polk County mine was 126,966 tons; in the Burra Burra, 1,806,960; in the London, 359,443; total 2,293,369. The probable ore in the Burra Burra mine was estimated at 1,000,000 tons, and in the London mine at 30,000 tons. The smelting works in 1907 treated 389,603 tons of ore, 8401 tons of converter slag, 23,045 tons of blast furnace products, 89,035 tons of quartz flux, 91,280 tons of first matte and custom matte, and 3571 tons of custom ore, a

total of 604,935 tons. The consumption of coke was 37,269 tons. The mine, railway, smelter and construction work employed an average of 1211 men during the year. The smelting works produced 12,599,019 lb. of copper, or 32.34 lb. per ton of ore. The cost of mining and smelting is given in the accompanying table.

The increased cost per ton of ore over the figures for 1906 is accounted for almost entirely by increase in rate of wages and increase of cost of supplies. According to the report of the treasurer, 2,230,136 lb. of the copper produced was electrolytically refined during the first half of the year; the remainder was marketed in the form of pig copper. The cost of electrolytic copper, after allowing for gold and silver, was 12.22c. per lb. The cost of producing and marketing per pound of fine copper in pig was as follows: Cost at works, 10.44c.; freight, insurance and other selling expenses 0.68c.; taxes, legal and administration expenses 0.67c., total 11.79c. The profits for the year, after deducting $70,000 for depreciation of the plant in Tennessee, were $800,635. The sulphuric acid plant was completed in December, 1907, and is now producing acid. No revenue was derived from this source during 1907. The sum of $728,067 was expended for new construction and equipment. Of this amount $504,525 was expended on the sulphuric acid plant, and about $114,000 in improving the smelting plant.

Utah.—This State showed a large increase in production in 1907, due especially to the operation of the Garfield works, which afforded smelting capacity for the ores of the Newhouse, Boston Consolidated and Utah companies. The Utah Copper Company put the first section of its new mill into operation in June and by the end of the year had eight sections (4000 tons capacity) in operation. This company alone accounts for a large part of the increased production of Utah. The Boston mill being scheduled to go into operation in January, 1908, there would normally be every reason to anticipate another great increase in Utah's production in 1908, but the closing of the United States, Bingham and Highland Boy smelters will have an effect which cannot yet be outlined. On the other hand the Garfield smelter is being doubled in capacity; the additions will doubtless be completed early in 1908. The statistics for Utah in the accompanying table of copper production in the United States are only approximately correct. The smelters received a good deal of ore from California, Idaho, Nevada, Colorado, and other States, which have been allowed for, as thoroughly as possible, but even so the figures credited to Utah are perhaps a little too high while those of certain other States are correspondingly too low.

The Utah Consolidated Mining Company in 1907 produced 271,332 tons of copper ore and 1734 tons of lead ore. The smeltery treated 272,989 tons of copper ore from the company's mine and 6653 tons of custom ore,

an average of about 748 tons per day. At the end of the year the amount of copper ore developed was 1,202,930 tons, against 1,100,000 tons at the end of the previous year. Several new ore bodies were discovered during 1907, and the outlook for the mine is regarded as promising. In January, 1908, the company made a contract with the Garfield Smelting Company for the treatment of 800 tons of ore per day, this contract being for one year with option to renew for another year. Its terms are said to be advantageous to the mining company. The latter has purchased 1600 acres of land (in and below the Pine cañon, Tooele county, about 3.7 miles north of the mine) as a site for the new smelting works. Seven miles of railway will be required to connect this site with the San Pedro railroad, and an additional seven miles to connect with the Western Pacific near Garfield. The new smelting works will be of capacity for 1500 tons of ore per day.

STATISTICS OF UTAH CONSOLIDATED.

	1904	1905	1906	1907
Ore treated tons	233,700	286,200	296,989	279,642
Copper, lb	13,553,483	17,264,474	18,533,974	13,987,551
per ton of ore	57.9	60.3	62.4	50.0
Silver, oz	268,880	374,685	457,812	390 296
per ton of ore	1.1	1.3	1.6	1.4
Gold, oz	23,374	28,290	42,601	34,554
Net profit	$1,179,412	$2,835,008	$1,887,385	$1,164,348
Mining cost	....	....	473,760	582,866
Exploration and development	....	....	84,864	107 155
Smelting cost	....	....	747,717	867,087
Custom ore purchases	....	....	274,032	131,796
Refining charges, freight, marketing, etc	....	....	267,921	227,152
Miscellaneous	....	....	70,773	70,754
Total cost	....	....	$1,919,067	$1,986,810

The Utah Copper Company's balance sheet of June 30, 1907, showed:—Assets: Cost of property, $5,702,572; improvements, $3,330,679; ore account, $212,468; outside investments at cost, $140,000; accounts receivable, $90,580; sinking fund, $23,081; storehouse supplies, $73,255; copper in transit, $425,598; bank balance, $35,802; total, $10,094,037. Liabilities: Capital stock, $5,118,000; accounts payable, $18,887; sinking fund, $16,682; A. S. & R. Co., $47,991; due to general treasury, $422,216; surplus (from sale of stock) $918,000; balance net earnings, $588,261; total, $10,094,037.

President C. M. MacNeill, in his report to the stockholders, says: "Heretofore our fiscal year has ended on June 30, but it is now proposed to change this so that it will correspond with the calendar year and consequently our next statement will embrace the 18-months period from July 1, 1907, to Dec. 31, 1908. As the new Garfield plant was not put in operation until the close of the fiscal year, results from the plant do not appear in the statement of income.

"The operation of the Garfield plant up to the present time has fully

proved the accuracy of the previous estimates, although operations cannot reach their maximum for some time to come. At this time (January, 1908) the company is producing at the rate of 3,000,000 lb. of copper per month, at a cost of less than 8.5c. per lb., average, for both plants. Of this output, approximately four-fifths is coming from the Garfield plant, at a cost under 8c. per lb. Our experience in mining by steam shovel has proved that our mining costs should not exceed 25c. per ton. The Garfield plant is showing results as above, notwithstanding the fact that the ore it is now receiving, and will continue to receive for some months to come, is partially oxidized and lower grade than the average orebody on account of the impossibility of preventing some of the surface oxidized material from becoming mixed with ore as it is mined by shovels. Therefore, in view of our performance under existing conditions, which cannot be brought to their highest degree of efficiency until the Garfield plant is in full operation, and the steam-shovel pits are somewhat further opened up, thus permitting the most economical working, it can be stated with confidence that we should be producing copper within the next 12 months at a cost not to exceed $7\frac{3}{4}$c. per lb."

General Manager D. C. Jackling says in his report: "The mine has been developed by approximately 90,000 ft. of underground workings. The ore averages about 2 per cent. copper, 0.15 oz. silver and 0.015 oz. gold per ton. The average thickness of the orebody in the developed zone, covering 27 acres, is about 310 ft., equivalent to 60,000,000 tons in the body. About 20,000,000 tons has been blocked out; 20,000,000 tons partially developed and 20,000,000 tons is classed as undeveloped. In the 72-acre area there is a zone of lower-grade ore, averaging 1.5 per cent. copper, containing about 40,000,000 tons."

At the annual meeting of the company, the stockholders voted to authorize the issue of $1,500,000 of 6 per cent. convertible bonds, to be secured by a second mortgage on all of the company's property and convertible at the holders' option into stock at $20 per share. Proceeds of the bonds sold are to be applied to increasing working capital. The stockholders voted also to increase the capital stock from $6,600,000 to $7,500,000.

The Boston Consolidated Mining Company reported, for the fiscal year ending Sept. 30, 1907, that the development of the Sulphide mine has advanced with satisfactory results, several important new orebodies having been discovered. During the year there was delivered to the Bingham Consolidated Mining and Smelting Company 11,918.65 tons of sulphide ore, containing 1,405.49 oz. gold, 9,879.44 oz. silver, and 508,862 lb. of copper, for which was received, after deducting transportation, smelting and refining charges, the sum of $95,162.51. There was also delivered to the Garfield Smelting Company 122,386.44 tons of sulphide ore, containing 11,236.96 oz. gold, 68,249.47 oz. silver, and 5,638,063

℔. copper, making a total of 134,305.09 tons of sulphide ore, containing 12,642.46 oz. gold, 78,128.91 oz. silver, and 6,146,925 ℔. copper. The cost of mining this ore, including development, was $314,584.68 (or $2.34 per ton); transportation to smelters, $67,755.78; smelting, $354,400.43; freight and refining charge upon bullion, $97,734.54; total, $834,475.43. After crediting the value of gold and silver, 12,642.46 oz. gold at $20 per oz., or $252,849.22, and 78,128.91 oz. silver at 64c. per oz. or $50,002.50, a total of $302,851.72, there remains to be applied to the cost of production of copper, $531,623.71, or a cost of 8.65c. per ℔.

The inadequate and erratic railway service afforded by the Rio Grande Western and Copper Belt Railways, due to insufficient preparation on the part of these companies to meet the increased demands of Bingham shippers, kept shipments far below the tonnage it was intended to move; the uncertain car supply served to increase the mining cost.

At the Porphyry mine at the beginning of the last fiscal year one steam shovel was in operation, and since that time three others have been added, starting operations respectively, Oct. 6, 1906, Feb. 21, 1907, and March 16, 1907. During the last year these shovels stripped and removed 2,011,733 tons of capping. There was expended for this service, including supervision, drilling, railway operation, repairs, machine shop expenses, water line expenses, the sum of $215,107.40 for labor, or a labor cost of 10.69c. per ton. There was expended for explosives, fuel and other material involved in the operation, the sum of $152,177.34, or a cost of 7.56c. per ton, making a total cost of 18.25c. per ton for mining, removing and disposing of the capping. Miscellaneous expenditures amounting to $12,290.78, not properly included in above-mentioned items, were made. The total expenditure upon this work was $379,575.52.

In former reports estimates were made that mining could be done with steam shovels at 40c. per ton. The engineers of the company estimate that the quantity of capping to be removed and disposed of is approximately one-half the amount of the ore tonnage. The steam-shovel equipment has been increased by the addition of three 90-ton Marion shovels. Upon steam-shovel equipment there has been expended $72,980.90. Seven shovel bench levels are now operated with tracks all connected to a common system. Upon railway trackage and equipment during the year there was expended $129,665.54.

An incline tramway extending from the 60-ft. steam shovel bench down the mountain side, a distance of 2050 ft., to an ore-loading station located upon the Carr Fork railway track has been constructed. This tramway includes a double road-bed, carrying four tracks laid with 60-℔. rail, 36-in. gage, and is in reality two complete and separate systems side by side. It is equipped with two Stine patent double lowering drums

at the top terminal, and at the Carr Fork receiving terminal has a 3000-ton capacity steel ore bin. The skips operating upon the tramway carry 13 tons each, and are capable of making a round trip each three minutes. The carrying capacity of this tramway is equal to 6000 tons of ore per day. There has been expended on its construction and equipment $52,504.63. In order to secure connection with Rio Grande Western Railway tracks, it became necessary to build a standard-gage track from the auxiliary yard up to Carr Fork, a distance of 2700 ft. This required the construction of 250 ft. of railway tunnel, timbered throughout, a 215-ft. railway bridge, and 1000 ft. of rock cut. On September 30 this work was about 70 per cent. complete, and there had been expended upon it the sum of $28,505.03.

The construction of the Garfield mill has rapidly advanced and with the close of the fiscal year the steel buildings were complete, the crushers, conveyers and stamps were in position, and 30 per cent. of the concentrating tables were erected and adjusted. Mill expenditure to date is $1,468,901.91.

In concluding his report, Samuel Newhouse, president of the company, said: "The company has now arrived at the point of commencing the production of copper on a very large scale from its Porphyry mine, and is in position to maintain a heavy tonnage of ore from its Sulphide mine. It has nearly finished its improvement expenditures, and while facing at the present time a depressed and unsettled copper market, I do not hesitate to say that the mines can and will earn substantial profits even at the present market price of copper. The engineers have conservatively figured 58,000,000 tons of workable and payable ores now contained in the company's mines. The mill will consume when operating at its fullest capacity but 1,080,000 tons per year of this vast quantity, which involves a period of 50 years of active operation for the conversion of ore with our present plant. In possession of one of the largest copper properties in the world, susbtantial results are limited only by the capacity provided for the treatment of the ores. The company is in a position to mine double the amount now required by the mill."

(By Lewis H. Beason.) The chief mining activity in Utah in 1907 centered at Bingham and Tintic. At Bingham, the Utah Copper Company and the Boston Consolidated Mining Company were occupied during the major part of the year in stripping the capping from their deposits of porphyry ore preparatory to milling at Garfield. The Utah Copper Company began milling at Garfield about the middle of the year, and at the end of the year had two-thirds of its capacity in operation. It is expected that the whole mill will be running by May 1, 1908, treating 6000 to 6500 tons of ore per day. Since the Utah Copper Company began production at its Copperton mill, a little over two years ago, its

mine has yielded 1,300,000 tons of ore, of which 400,000 tons have been removed by steam shovel.

The Boston Consolidated was comparatively inactive during the last quarter of 1907. It was the first of the Bingham companies to curtail operations. However, the construction of the mill at Garfield proceeded slowly, and early in 1908 the first section was put in operation.

The Utah Consolidated will build a new smeltery in Pine cañon in 1908, to which ore will be shipped from the mine by an aerial tramway about 13,000 ft. long. This plan will give the company an improved smelting works in a district where there will be but little danger from damage suits on account of smoke, and probably the transportation of ore to it will be less than the rate, 40c. per ton, which heretofore has been paid to the Rio Grande Western railway.

The litigation over the smoke question was an active subject of interest in 1906. The Utah Consolidated endeavored to secure a modification of the injunction, whereby it would be enabled to continue operations at its present smeltery until March 1, 1909. At a mass meeting of the farmers a resolution favoring this was introduced, with the proviso that the company should pay to the farmers $200,000 for the privilege. It developed, however, that it was impossible for the company and the farmers to come to an agreement, and consequently the smelting works was closed Jan. 6, 1908.

The Bingham Consolidated Company for the same reason closed its smelter Dec. 26, 1907. Early in 1908 the works of the United States Smelting Company also was closed. This left in operation only the Yampa works, which are outside of the agricultural zone, the Murray smelter, for which an arrangement had previously been made with the farmers for a pecuniary consideration, and the Garfield works, which stand in a large domain owned by the company, and therefore are out of the reach of the farmers.

The Yampa Smelting Company increased the capacity of its plant in 1907 to 700 tons of ore daily. A converter plant of two stands is now being installed. Heretofore the Yampa matte has gone to the United States Smelting Company for converting. The Yampa mines and smelter have been connected by an aerial tramway.

The Ohio Copper Company made important developments in 1907. A large supply of ore is said to have been developed. The company is building a concentrating mill at the mouth of the Mascotte tunnel, through which the ore is to be brought out. The mill is to be of 2000 tons daily capacity. The Ohio Copper Company is a Heinze interest. Mr. Heinze purchased a smelter site near Garfield last summer and soon afterward the Miners' Smelting Company was organized. Apparently, it was the

intention to engage in the general smelting business, but the project was apparently disturbed by the financial troubles of October.

Vermont.—New developments in the State are described in a special article which follows. The production increased in 1907. However, toward the end of the year the low price of copper forced the Pike Hill mine and the Smith and Higley mines, near Corinth, to shut down, and if the price for copper continues low, there is not much likelihood that operations will be resumed. The Vermont Copper Company was engaged in executing some rather elaborate plans, including the construction of a 350-ton smelter.

(By George A. Packard.) "For a period of many years the State of Vermont stood second in the list of copper producers in this country," wrote Dr. Peters in the *Engineering and Mining Journal* in 1891[1], in an article describing the Ely mine. A recent visit shows renewed activity in this State, although the Ely, at Vershire, the greatest producer of the past, is abandoned and the old buildings are being torn down.

At South Strafford, six miles south of Ely, is the Elizabeth mine, first worked in 1795 for copperas, and spasmodically since then for copper, activity or idleness depending largely on the price of the metal. In my own experience there, in 1890, 12c. copper was the dividing line, as the property was then equipped and operated. The ore was sorted and that carrying over 5 per cent. copper was hauled 10 miles to the railroad and shipped to New Jersey. The lower grade, running a little higher than 2 per cent., was roasted in heaps, smelted, and the resulting 35 per cent. matte was shipped. The furnace had a diameter of 48 in. and we put through 30 tons in 24 hours, using 12 per cent. coke. No fluxes were used, the only variation in the charge being in the use of ore high in pyrrhotite which was roasted separately and added as needed.

In those days the mine was worked through a shaft and all ore had to be hoisted. Now all this is changed. The property has passed into the hands of a New Jersey company which is preparing to operate on an up-to-date basis. A furnace of 350 tons capacity is being erected and the long flue and dust chambers, leading to the stack, are being constructed. The furnace has some new features suggested by H. F. Wierum and designed by C. F. Buck. Among these is a scheme for feeding from cars having scales attached. Heap-roasting will be abolished and pyritic smelting with subsequent treatment of the matte, somewhat after the plan of the Tennessee Copper Company, will be adopted. An adit level, 1340 ft. long, has been driven, cutting the orebody at a depth of 225 ft., while a winze sunk on the incline has proved the continuation for more than 100 ft. deeper. The width varies from 20 to 80 ft., and there is over 250,000 tons of ore exposed above the lowest level. Mr. Weed has given

[1] "The Ely Mine of Vermont." E. D. Peters, *Eng. and Min. Journ.*, LII., 6.

the average composition of this ore[1] as follows: 3.25 per cent. Cu; 35.6 per cent. Fe; 27 per cent. SiO_2; 19.18 per cent. S; 7.76 per cent. Al_2O_3; 1.07 per cent. Zn; 1.55 per cent. CaO; 0.82 per cent. MgO; from a trace to 0.02 oz. per ton gold; and 0.20 oz. per ton silver.

At the mine a new Sullivan belt-driven compressor having a capacity of 1050 cu.ft. of free air per minute is already installed. The method of mining is to be gradually changed, though the old system of underhand stoping will be continued on the upper levels. Below the adit level the the single square set system will be introduced and stopes will be carried up 100 ft. The ore from these lower levels will be handled in skips operated by an electric hoist and dumping into a pocket. From this, and from the chutes from the upper levels, the ore will be drawn into cars of five-ton capacity, which will be drawn from the mine in trains by an electric locomotive. The ore will be dumped over grizzlies, the oversize going to a 20x30 in. jaw crusher, and then be distributed by belts to an automatic sampler and bins. From these it is delivered by electrically hauled cars over a 1500 ft. trestle to the bins at the furnaces, and an automatic sampler. This part of the work is well along toward completion. A dam is being put in on the White river, eight miles away, and an electric power line will transmit a minimum of 400 h.p. to replace the old wood-burning boilers. This line is now practically completed. The work on the dam has been seriously delayed by the weather but is now progressing satisfactorily and it is hoped that the entire plant will be in operation before the close of the summer of 1908. The same company has taken over the Foster property, adjoining the Elizabeth to the south, and also the "South Mines," a mile away.

About 17 miles north are the Corinth mines. The Union, formerly the largest producer, is closed, but the Eureka which adjoins it is being operated by a New York company. This mine was originally opened by a shaft, but is now operated through an adit level connecting with the old workings at a depth of about 300 ft. Below the adit a 400-ft. incline gives an additional vertical depth of 135 ft. The ore here occurs in lenses as at Ely, the width varying from 8 to 20 ft. or more. There is no smelter at Corinth; the ore was formerly smelted at Ely. Now the products are shipped to smelters in the vicinity of New York, both freight and smelting charges being reasonable. The high-grade ore is sorted out and shipped without treatment. The low-grade, after drying and removing the pyrrhotite by a Wetherill magnetic separator, is roasted and treated by a second magnetic separating machine, the chalcopyrite concentrates being shipped. A similar process was tried at the Elizabeth[2], but was not successful, perhaps in part because the ore received a partial roast in the pre-

[1] "Notes on the Copper Mines of Vermont." Walter Harvey Weed, U. S. Geol. Survey, *Bull.* 225, 190
[2] "Magnetic Separation of Pyrrhotite and Chalcopyrite." *Eng. and Min. Journ.*, LXXX, 1212.

liminary drying, and perhaps because of the greater amount of pyrrhotite in the ore, with which the chalcopyrite is too intimately mixed. At the Eureka mine the proportion of pyrrhotite is increasing as depth is attained, and the tonnage handled by the Wetherill machine consequently decreases. As the old portion of the mine is said to contain large reserves, also high in pyrrhotite, the management is considering plans for the addition of a smelter, and it is apparently advantageous to retain the silicious ore for fluxing purposes.

Half a mile south of the Eureka, the Corinth Copper Company has a 150-ft. adit level on an orebody, crosscut perhaps 40 ft., from which some fine ore is being sorted and shipped. The development is still small, the property having been but recently opened. This ore, like that of the other properties except a portion of the Eureka, shows an excess of iron.

In addition to the properties above named prospects are being opened at several other points. One, across the Connecticut river at Lebanon, N. H., shows the most silicious ore seen in the vicinty, an ideal flux for the baser ores if all these mines could be served by a single centrally situated smelter.

The geology of this vicinity has been described by Wendt[1], Weed[2], Smyth and Smith[3], and others. Probably none of these properties will exceed 3 per cent. copper as the average of the entire orebody, with a precious metal content of one-half cent for each pound of copper. They have, however, the advantages of a situation close to market and low cost for labor and supplies. Timber is rarely necessary and there is but little water to be pumped.

Virginia.—This State produced upward of 2,000,000 lb. of copper in 1907, which came chiefly from the Virgilina district. The ore was shipped partly to the Eustis works, at Norfolk, and partly to the smelters at Baltimore and New York. The New York & Virginia Copper Company operated the Toncray mines, near Floyd, and built a small smelting furnace, but the undertaking did not result successfully.

Wyoming (By H. C. Beeler).—The principal production of copper in Wyoming in 1907 came from the Grand Encampment district in southern Carbon county, where the concentrating works and smelter of the Penn-Wyoming Copper Company at Encampment, which were destroyed by fire, were rebuilt, and produced during the latter part of the year. The total production for the State is estimated at 2,350,000 lb., only small amounts in test shipments being returned from the camps other than Encampment. Development work in the Encampment district generally was active, and there are several properties, which are expected to enter the list of producers during 1908, especially the Itmay, Portland, Doane-

[1] The Pyrites Deposits of the Alleghanies. A. F. Wendt, *School of Mines Quarterly*, VII.
[2] "Notes on the Copper Mines of Vermont," Walter Harvey Weed, U. S. Geol. Survey. *Bull.* 225,190.
[3] The Copper Deposits of Orange County, Vermont. H. L. Smyth and P. S. Smith, *Eng. and Min. Journ.*, LXXVII, 677.

Rambler, and Charter Oak. The building of the Saratoga & Encampment railroad, 44 miles, by the Penn-Wyoming Copper Company, from Wolcott (on the Union Pacific) to Encampment has changed the entire situation in this district, and there are a number of branches of this line now projected, which will afford the whole district adequate railroad transportation. The eastern portion of this district is served by the Laramie, Hahn's Peak & Pacific railroad from Laramie, and work on its North Park, Colorado, extension is in active progress.

In the Medicine Bow range at Elk Mountain, and south of Encampment, as well as in the Dillon vicinity there are a number of properties rapidly approaching the production stage. These are working on showings similar to the best properties of this section, and their proving at depth will mean much to the copper industry of Wyoming. The Strong company, east of Laramie, rebuilt its shaft plant, after a disastrous fire, and is now negotiating for a concentrating mill to be installed as early as possible in 1908.

At the Esterbrook property north of Laramie Peak, a reorganization has been effected and deep work actively resumed. This is the deepest property in this section. The opening for location of the Owl Creek range, formerly held in the Shoshone Indian Reservation, has provided a new field for prospecting in the northwestern portion of the State. It is regarded as a promising field by engineers who have examined it. Copper Mountain is the eastern extension of the Owl Creek range, and work has been actively carried on here for the last two years. The principal property is the Williams-Luman at Depass, in Fremont county, which shows both gold and copper in a crushed and fissured diorite. At the surface it is 50 or 60 ft. wide and is traced for a length of about 4500 ft., with the usual oxidized copper minerals shown in the surface developments. An adit level has been run cutting this ledge at a depth of 200 ft. and the ore shown to be over 85 ft. wide, 32 ft. of which showed over 30 per cent. of copper. The copper here is native copper, occurring in thin sheets and nuggets throughout the ore. A shaft has been started from this adit and has reached a depth of 120 ft. below its level, showing the same condition as at the adit level with the addition of gray copper in the ore. The owners of this property have contracted with the Ryan Electrolytic Copper Company to install a test plant at the mine.

Reports from the smaller camps of the State, in Uinta county, northern Big Horn county and other points all indicate an active prosecution of work.

Copper Mining in Foreign Countries.

Argentina.—Copper deposits have been worked or are known to exist at several places in this Republic. The Concordia mines yield tetrahed-

rite and chalcopyrite, containing gold and silver, and a little zinc blende and galena. They were operated from 1899 to 1903 by the Concordia Company, Ltd.; they are now owned by the Compania Minera la Concordia. This company has not yet begun to produce, but is now installing an extensive plant, including hydro-electric generator, hoist, concentrator, etc. Not far from there, at Cerro Acay in the province of Salta, are deposits of chalcopyrite, carrying 30 or 35 per cent. sulphur, suitable for the manufacture of sulphuric acid. At Las Cortaderas, to the east of Salar de l'Hombre Muerto, is a rich deposit of malachite and chalcocite which is not developed on account of the difficulty of access. The prospects for copper in Catamarca and Rioja are said to be good. The Famatina Copper Company is said to have 80,000 tons of ore developed.

Australia.—The copper production of this Commonwealth increased largely in 1907, amounting to 41,250 long tons which is the maximum on record, and shows the stimulating effect of high prices upon the industry. The large producers are the Mt. Lyell, the Wallaroo & Moonta, the Great Cobar, and the Mt. Morgan companies.

The Mt. Morgan Gold Mining Company continued to be the largest producer of copper in Queensland, its average monthly output having been about 400 tons. Another important copper mining district of this State is Mt. Perry, the yield of which in 1907 was in the neighborhood of 2000 tons. The Chillagoe Company, having improved its smelting works made an increased output. This company is rapidly pushing ahead the construction of the Etheridge railway, the completion of which should open a mineral country of great promise. The Mt. Molloy mines were compelled to suspend smelting operations when the price of copper fell.

The Electrolytic Smelting and Refining Company of Australia was engaged in the erection of works at Port Kembla, New South Wales, to treat the blister copper of the Mt. Morgan and other mines which have joined interests in the new refining company. Heretofore the Mt. Morgan blister copper has been shipped to Chrome, N. J., for refining.

The Wallaroo & Moonta Mining and Smelting Company in 1907 produced 55,129 tons of ore, containing 4419 tons of fine copper, from its Wallaroo mines; and 16,924 tons of ore, containing 1804 tons of fine copper, from its Moonta mines. The cost of production per ton of copper was £52, 13s. 9d. for mining, £19 19s. 9d. for smelting, 15s. 1d. for shipping charges, 9s. 2d. for Adelaide expenses, making a total of £73 17s. 9d. The high cost per ton of copper is attributed to the great depth of the mining operations, the increasingly difficult nature of the ground, the high rate of wages paid under the sliding scale, the increased cost of all mining supplies, and partly to the Commonwealth tariff. Some of these causes appear likely to be permanent, but the rates of wages have been to some

extent readjusted automatically by the action of the sliding scale. The cost per ton of ore in 1907 was £1 4s. 6d. for mining; 4s. 5½d. for dressing; 10.1d. for transportation to smelting works; 11s. 7½d. for smelting; 8.1d. for shipping and Adelaide office expenses; total £2 2s. 1d. The average price received for copper during 1907 was £91 15s. 1d. The smelting works treated a total of 75,666 tons of ore, matte and precipitate, and produced 8627 tons of refined copper, 223 tons of bluestone, 2009 oz. of gold and 5845 oz. of silver. The production of sulphuric acid was 5379 tons, which cost 17s. 4d. per ton.

WALLAROO & MOONTA STATISTICS.

	1904	1905	1906	1907
Ore from Wallaroo mines, tons	21,766	35,189	43,241	53,571
Ore from Moonta mines, tons	10,686	8,161	10,360	16,397
Precipitate, tons	494	930	957	833
Outside ores and matte, tons	5,381	5,151	3,520	4,865
Ore smelted, tons	38,995	49,961	58,068	75,666
Copper placed in store, tons	5,835	6,501	7,561	8,627
Silver to mint, oz	7,147	4,781	3,614	5,845
Gold to mint, oz	1,260	1,646	1,643	2,009
Sulphuric acid delivered, tons	3,433	5,312	5,112	5,379
Bluestone made, tons	181	340	328	224

The Great Cobar, Ltd., was the principal producer of copper in New South Wales in 1907. This company was engaged in making extensive additions to its plant, including the erection of a new smelting works (described with much detail in *Eng. and Min. Journ.*, May 9, 1908). The new works are expected to be in operation by the middle of 1908. The ore smelted by the Great Cobar, Ltd., contains about 2½ per cent. copper, besides a little gold and silver. The total production of the Cobar district in 1907 was upward of 6000 tons of copper. A mine of considerable promise is being developed at Cangai in the Grafton district. Work at the Lloyd mine, Burraga, was somewhat intermittent, but at the end of the year a determined effort was being made to put it on a regular basis.

In Tasmania, the Mt. Lyell company continued to be the largest producer. In its fiscal year ending Sept. 30, 1907, it produced 7886 tons of copper, 19,449 oz. of gold and 700,087 oz. of silver, paying £405,000 in dividends, bringing its aggregate of dividends up to £1,896,574. The ore treated amounted to 406,000 tons, and the average cost of mining and smelting to $3.58 per ton. The developments of the North Mt. Lyell mine are reported as being of a highly satisfactory nature, the directors stating that both in quantity and quality the ore reserves never appeared to better advantage than at the end of the last fiscal year.

(By F. S. Mance.) The output of the Great Cobar company in 1907 showed a small increase over 1906, but that of the other companies of New South Wales was on a smaller scale. Production at the Queen Bee mine was delayed for six weeks owing to labor troubles. The new

company which took over the Nymagee mine entered upon the work of replacing the old plant with a modern one, which involved suspension of production for six months. The Crowl Creek company at Shuttleton closed down in October owing to financial difficulties. The returns from the Girilambone and Mt. Hope mines show a good improvement over 1906. The Lloyd mine, Burraga, made a slightly less output than in 1906, due largely to difficulty in securing an adequate supply of fuel. The mine of the Cadia Syndicate in the Orange district yielded well, and now that a smelting plant has been provided should make a steady output. The Blayney mine was reopened during 1907 and the company has entered the custom smelting business, which has been an additional stimulus to copper mining in the western districts. The developments at the Cangai mine in the Grafton district were of a highly promising nature and the indications are that this will soon rank among the large producers of the State.

(By Sydney Fry.) A discovery of copper ore which appears to be the most promising in New Zealand, has been made at Mount Radiant, 14 miles southeast of Karamea, at the mouth of the Karamea river. The deposit occurs in a zone of granite, and porphyrite, 260 or 300 ft. wide, which has been shattered by earth movements and the resulting network of fissures has become filled with quartz and metallic sulphides, chiefly chalcopyrite and molybdenite. The sulphides are disseminated throughout the quartz veins and the adjacent granite; sometimes they occur in richly mineralized nests or pockets, or as veins of pure ore. Picked samples of the chalcopyrite, free from gangue, gave 20 per cent. metallic copper, and other samples of the quartz containing chalcopyrite, carried 0.138 oz. gold and 0.87 oz. silver per ton. Outcrops of the ore have been found for a distance of five miles along the strike, and in the sides of the gorge of the Wanganui river at a depth of 2000 ft. below the outcrops first found. It may be described briefly as an enormous low-grade deposit with probably some bonanzas; but it will take prospecting on a rather large scale, before it can be ascertained whether the grade is high enough to be payable on the average. At the time of my visit it was untouched, and nothing beyond a little surface scratching to lay bare the outcrops has been done since then. The New Zealand Mines Department is at present constructing a trail to Mount Radiant from Karamea to facilitate prospecting operations.

British East Africa.—A discovery of copper ore, about 70 km. west of Tsavo, on the Uganda railway, was reported. The ore is said to be sulphide, together with some native copper.

Bolivia.—The copper production of this Republic continues to come chiefly from Corocoro, the mines of which are considered to be rich and capable of much greater output than at present. They have been examined

in the interest of American capitalists, but the price asked for them is said to be too high.

Canada.—The smelters of the Boundary district suspended operation in the autumn, but previous to that the British Columbia company had made such a large increase that the total for the district shows only a small decrease from 1906. Rossland made a small increase. The smelters of Vancouver Island made a decrease. On the other hand the Sudbury district made a large increase. The net result is a considerable increase in the Canadian production, which was 46,356,382 ℔. in 1907, against 42,121,000 in 1906. These statistics are based on direct reports from the producers.

The smelters of the Boundary district made important improvements in 1907. The British Columbia Copper Company, at Greenwood, installed a third blast furnace, raising the capacity of the works to 2000 tons of ore per day. The Granby company produced 15,514,000 ℔. of copper, labor troubles, shortage of coke, and finally the great decline in copper, preventing the consummation of the plans to produce 25,000,000 ℔. However, the plans to increase the capacity of the works by lengthening the eight blast furnaces to 22 ft., which will give a minimum capacity for smelting 4000 tons of ore per day, are being carried out. Also, three large converters are to be added. The report of the Tyee Copper Company, Ltd., Vancouver Island, for the year ended April 30, 1907, stated that the development at the mines has not yet disclosed any new orebodies. The directors and manager are still hopeful, however, that discoveries may be made. Though the orebodies are being depleted, the custom work at the smelter is increasing in a satisfactory way, so much so that increased plant is in contemplation.

The Granby company for the fiscal year ended June 30, 1907, produced 16,410,576 ℔. of copper, 257,358 oz. of silver, and 35,083 oz. of gold. The gross earnings were $4,521,549, while the working expenses were $2,442,456 or $3.697 per ton of ore treated. The smelter treated 649,022 tons of Granby ore and 16,893 tons of custom ore. President Langeloth's report says: "The operations during the year show a very considerable falling off as compared with the previous year, in spite of the fact that the mines were prepared to furnish a much larger tonnage and the smelter fully equipped to handle it. This was due to the great shortage of fuel. In the British Columbia coalfields, there were two strikes, one last fall and the other last spring, resulting in the output of coke being crippled to such an extent that at no time could the quantities contracted for be delivered. A severe winter caused blockades of all the railroads, which, irrespective of this, were hardly able to take care of the largely increased traffic. In order to relieve the situation temporarily, contracts were made last October for about 20,000 tons of eastern coke, which entailed an extra

expenditure of nearly $100,000, but later in the season even these supplies were stopped on account of the railroads being unable to make deliveries. All these circumstances interfered seriously with the operations of the plant, and the cost of mining and especially of smelting increased considerably. The eight large furnaces could be operated only intermittently, and during May both mines and smelters had to be closed down for want of fuel.

"It was estimated at the beginning of the year that, due to the greater capacity of the smelter, the production could be increased to about 25,000,000 lb. Instead of this, only 16,403,749 lb. of copper were produced. The cost per pound of copper produced, after deducting the value of gold and silver, was 10.14c. during the past year, against 8.35c. in the preceding year. If the mines and plants are operated to their full capacity, lower costs can again be confidently expected. At the smelter the eight furnaces are now in shape to handle over one million tons of ore per year, which should produce in the neighborhood of 30,000,000 lb. of copper.

"Among the more important new work completed at the mines was the sinking of the new Victoria three-compartment shaft; a complete electric hauling system is being installed on the 400-ft. level. It is estimated to hoist and crush 2000 tons of ore daily at this shaft alone. The Gold Drop and Monarch properties, have been developed vigorously, and have proved valuable In a word, the mines are prepared to produce practically any tonnage that can be transported to the smelter, where the eight furnaces have been enlarged, and have now a maximum capacity of about 3500 tons per 24 hours.

"After mature deliberation it was decided to acquire a considerable interest in the Crow's Nest Pass Coal Company, Ltd., from which our main supply of fuel is secured. The wisdom of this step has already made itself felt, as for the last few weeks a full supply of coke has been furnished, thus overcoming the difficulties which were very costly to the company.

"During the year the stock of the company was converted into $100 shares par value, by exchanging 10 shares of $10 each into one share of $100."

(By E. Jacobs.) By far the greater proportion of copper produced in British Columbia in 1907 came from the mines of the Boundary district. About 14,000 tons of copper is the estimated output of Boundary mines, derived from 1,100,000 tons of ore. Rossland and the Coast district produced nearly 5,000,000 lb., while Nelson's production of this metal is estimated at about 400,000 lb. In the Boundary the mines of the Granby, British Columbia Copper, Dominion Copper, and Consolidated Mining and Smelting companies, were the producers; in Rossland camp the Le Roi, Centre Star-War Eagle group, and Le Roi No. 2, contributed to the total, the first making about half the copper production of this camp;

on the Coast of the Britannia's output was more than 50 per cent. of the total, with the Outsiders group on Portland canal, the Marble Bay on Texada island, the Tyee and Richard III. at Mt. Sicker, Vancouver island, and the Ikeda on one of the Queen Charlotte islands, together making up the bulk of the other half. The Queen Victoria in Nelson division was a new producer, as were also the Richard III., Lenora and Ikeda on the Coast.

Chile.—The production of copper in Chile in 1907 showed a small increase over 1906, but is still considerably below the figure which was attained in 1903. The developments of copper mining in this country appear to be retarded by labor difficulties, especially the scarcity of labor. According to Alfred A. Winslow, U. S. consul at Valparaiso, 7854 copper claims have been worked in Chile at different times, of which only 748 were worked during 1907. Many of these mines are rich in copper, but with the high price for labor and the poor transportation facilities, few can be made to pay at a low price for copper, save those equipped with up-to-date machinery. There are not many of these, and should the present conditions continue, the production of copper in Chile in 1908 is likely to decrease. The scarcity and high price of labor are factors that bear heavily on the industry. Up-to-date machinery and methods are needed to make mining profitable.

In an article in the London *Min. Journ.* (July 20, 1907) a correspondent gives some information as to conditions in copper mining in the province of Atacama. It appears that the greatest difficulty is the scarcity of labor, the natives being unwilling to work any more than enough to provide themselves with a bare living and the means of dissipation. The grade of the ore worked is over 6 per cent. copper, unless worked for flux, in which case 4 per cent. ore is utilized. The waste filling and dumps of the larger mines will yield 4 per cent. ore, and on some of the smaller prospects the grade of the waste is 5 per cent. (oxidized ore). The smelters at Lota and the Sociedad de Atacama quote only for 10 per cent. ore, though if good smelting ore, they are willing to accept 6 per cent. An unusual feature in the copper deposits is the great depth to which the oxidized zone extends. In the Dulcinea mines it varies from 350 to 400 m. from the surface, and in most mines in the province it exceeds 200 m. Coke comes from England and timber from the United States. The mining laws are liberal. A claim of one hectare in area is taxed $10 a year, and once it is surveyed by a government engineer is freehold as long as the fee is paid.

In The Mineral Industry, Vol. XV, p. 227, an error was made in referring to the works at Almendral, Coquimbo, which A. Gmehling, of Coquimbo, has been so good as to correct. There is no copper smelting works at that place, but only a small leaching establishment, extracting copper from

low-grade ores (2 to 5 per cent.) with dilute sulphuric acid and precipitating the copper with scrap iron. The precipitate (cement copper) contains on an average 60 per cent. of copper and is sold at Guayacan. The works produces monthly about 12 tons of cement copper. The copper-smelting works at Guayacan manufacture sulphuric acid as a by-product but at present its output is greater than the country can take care of. Consequently, to utilize the surplus, a plant for the leaching of copper ore with sulphuric acid is being erected in connection with the smeltery.

The Sociedad Chileno de Fundiciones, of Coquimbo, has eight reverberatory furnaces, 15x22 ft., which smelt daily about 100 tons of material, consisting of 60 tons of roasted matte (50 per cent. Cu and 5 per cent. S), 32 tons of oxidized ore (10 per cent. Cu) and eight tons of slag, about four tons of reducing coal being mixed with the charge and 50 tons of coal being burned in the fire-boxes. The product is 23½ to 24½ tons of pig copper, assaying 97.5 per cent. Cu, and 10 to 11 tons of matte with 75 per cent. Cu. The slag contains 0.5 per cent. Cu, on the average. The works also smelt 50 tons daily of self-fluxing copper ores for a product of matte, in which the fire-boxes consume 25 tons of coal. The matte contains 45 per cent. Cu and the slag 0.5 to 0.6 per cent. In 1907 coal cost $9 (gold) per ton. Good native labor costs $1@2 (gold) per day of 10 hours.

The Braden Copper Company, which owns mines about 42 miles east of Rancagua, promises to become one of the large producers of copper in Chile. These mines were described by William Braden in the *Engineering and Mining Journal*, Dec. 7, 1907. The Fortuna mine is estimated to have 4,000,000 tons of ore developed, averaging 3 to 4 per cent. copper. The occurrence of the ore is described as an extinct volcanic vent, filled with tuff, about three miles in circumference, surrounded by diorite, which is fractured and mineralized around the entire contact. The mineralization extends out for 1000 ft. in width, but the workable ore has an average width of 115 ft. The Teniente mine is opened in a breccia of tuff and porphyry, which are seamed with cuprite, carbonate of copper, native copper, and tetrahedrite, the last predominating. The Fortuna mine is opened by adit levels. The company has a concentrating mill of 250 tons daily capacity, which was built chiefly for experimental purposes and was in operation during 1907, producing sulphide concentrates, shipped to the United States. Plans for treating the concentrates on the ground are now under consideration, the purpose being to take advantage of the cheap hydraulic power, locally available. It is intended to roast the ore, make sulphuric acid, leach the copper from the roasted ore by means of dilute sulphuric acid, and precipate copper from the solution by electrolysis. Experiments that have been carried out at the Baltimore

Copper Works indicate that this process can be applied successfully. Mr. Braden estimates that in the treatment of ore containing 3.5 per cent. copper, the extraction in milling will be 70 per cent., the concentrate assaying 24.5 per cent. copper. On the basis of 1000 tons of ore per day, he estimates that copper can be produced for 6c. per pound.

China (By Thomas T. Read).—Copper has been mined in China for thousands of years, but in spite of that fact our knowledge of the industry is very vague. Yunnan and Süchuan are the chief producers. Mines are worked at three places in Kweichow, and the metal also occurs in Shantung, Kuangtung, Anhwei and Hupeh. The mines in Yunnan and Süchuan are the property of the government, and the local authorities frequently have no little trouble in persuading the miners to recognize that fact. The production, though large, is insufficient to supply enough metal for subsidiary coinage and brass and bronze ware.

A few years ago the provincial mints began the coinage of copper *t'ung-erh*, or 10-cash pieces, that were supposed to pass at the rate of 100 to the Mexican dollar. These were so popular that they commanded an even higher rate of exchange than that intended. As the profit in minting at the prices of silver and copper then obtaining was from 200 to 300 per cent. according to the purity of the pieces (zinc was used in the alloy), it was not remarkable that the provincial authorities ran their mints night and day. The copper pieces fell rapidly in price, and as the standard of living of the common people is measured in terms of these coins, the social disturbance was so serious that the national government stopped the coinage. The fall in the price of copper in 1907 again made the coinage of these pieces very profitable, and large quantities of Japanese copper are made into counterfeit coins and smuggled into China.

Congo Free State.—When the copper deposits of Katanga were first reported, the accounts were so extraordinary and indefinite that few were able to accept them seriously. However, it has gradually developed that they were substantially true as to the occurrence of immense deposits of ore extending over a large area, and it is no longer to be doubted that some day these mines will figure prominently in the copper market, although the present estimates respecting the probable cheapness of their copper are still to be taken with reserve. Obviously there is a great deal of work to be done before the prospects can be accurately determined. At present the chief matter of knowledge is the existence of very large deposits of ore that is of high grade in copper but silicious and entirely oxidized in character. The mines are remote; the ores present certain metallurgical difficulties; so far no workable coal has been found in the country. However, these conditions remind us of Arizona in the early 70's. Of course there are differences. The Southern Pacific railway across Arizona had already been projected at that time and the conditions in our West led to more

rapid developments than are probable in Africa. On the other hand, capitalists and engineers are now accustomed to consummate gigantic undertakings more rapidly than 35 years ago. Nevertheless, when we consider the many years required to bring the monzonite mines of Ely and Bingham to productiveness we can hardly escape the conclusion that it will be at least 10 years before those of Katanga will figure largely in the market.

About the end of March, 1908, word was received that the Cape-to-Cairo railway, which for two or three years, has been stationary at Broken Hill, Rhodesia—only about 225 miles from the Star of the Congo mine—is to be immediately pushed ahead to Mabaya on the Congo frontier, and thence to Ruwe in the copper belt. When we consider that for several years lead and zinc ores have been shipped in large quantities from Broken Hill to Europe, the Katanga copper ore does not appear so far away. In the meanwhile, the construction of the Benguella railway is going steadily ahead. At the end of 1907 about 100 miles had been built and the work was going ahead at the rate of one mile per day. The total distance from Lobito (the Atlantic terminal) to the Lualaba smelting site is 1000 miles, and the line is expected to cost about $30,000,000.

At the annual meeting of the Tanganyika Concessions, Ltd., at London, Dec. 20, 1907, Robert Williams who is in technical control of the mines made a report upon the developments up to date. The Tanganyika Concessions is the exploring company which received the great concession from the Congo Free State. It owns 45 per cent. of the capital stock of the operating company that has now been organized, and 90 per cent. of the Benguella railway. The substance of Mr. Williams' report is as follows:

During 1907 the Union Minière du Haut Katanga was formed in Brussels with a working capital of £400,000 to work the mines discovered up to Dec. 9, 1906. At the last meeting, I reported that, taking copper at £50 per ton, the Katanga mines had £100,000,000 opened at that date, and I put the cost of production and delivery on the market at £25 per ton. Since then the tonnage has greatly increased. The diamond drill has proved the continuity of the Kolwezi copper beds to three times the depth opened up by the shafts and drives, and developments at the Star of the Congo mine have exposed a further 800,000 tons of 12 per cent. ore, which with sorting and dressing, will be almost self-fluxing. At Kansanshi mine the ore has been cut at the 200-ft. level, and the main shaft is already down to the 300-ft. level. Test smelting operations have commenced at this mine, and at Kolweizi mine.

In view of the great wealth developed at the Star of the Congo mine, we are negotiating with the Rhodesia Railway company to construct its line direct to that mine instead of direct to Kansanshi mine, and to serve

the Kansanshi mine with a branch line. The Star of the Congo, besides having about ten times the wealth opened up that Kansanshi has, is about 60 miles nearer the Rodesian railhead. The Rhodesia railway can be at the Star of the Congo mine in a year, while our own line can tap our mines in three years.

With regard to the Benguella railway, the first 100 miles constructed over the worst section of the whole route will be opened for traffic this month at a point where it joins the main road to the interior. This line will have cost us about £2,500,000 when completed to K.320. I have our contractors' statement that a temporary line for copper can be completed for £3500 per mile—let us say £4500 per mile for 800 miles—within three years. It will, therefore, take another £3,600,000 to complete the 1000 miles at a total cost of £6,100,000 from Lobito Bay to the Lualaba smelting site in the middle of that group of mines. Of the required £6,-100,000 we have already provided nearly half the amount.

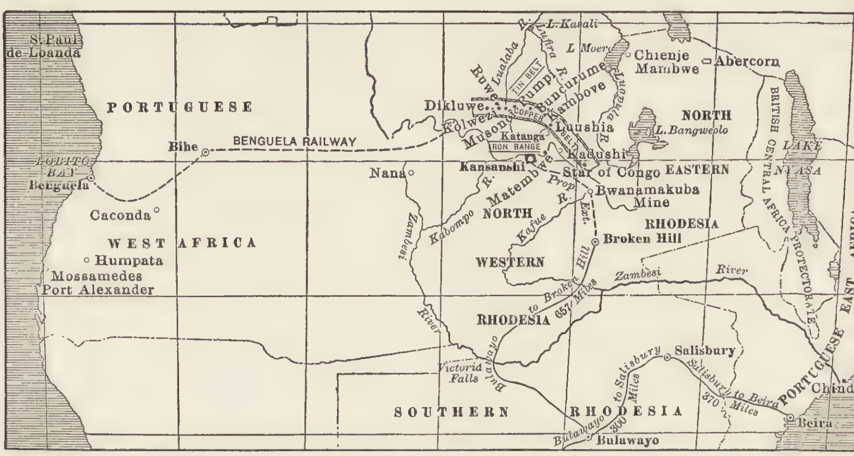
We have made solid progress during the year, and smelting is probably going on now both at Kolwezi and Kansanshi, and from all practical tests made, and from all reports received, our copper will not cost us over Mr. Farrell's orginal estimate of £12 per ton in Katanga. The average of all the mines of Kantanga may show 10 per cent. copper, with 40 per cent. free silica, taking the whole *en masse*, while the smelting production of copper would cost anything from £20 to £50 per ton. If, on the other hand, we take certain mines, and scientifically mine the ore we should be able to produce ore containing, say, 20 per cent. copper and 20 per cent. silica, with a probable cost of smelting of about £12 to £13 per ton. This is the problem we have to deal with in Katanga, which seems to me to offer facilities such as do not exist in any other country in the world.

We are now running out copper at Kansanshi and Kolwezi by test furnaces, and propose to send it to Rhodesian railhead by traction engines which are now at work on the spot. Allan Gibb, the mining engineer and metallurgist who was obtained from America, reported on Dec. 9, 1907, that the cost will not exceed £10 per ton of copper from the average produce of the Katanga mines.

(By John R. Farrell.) Katanga is the southeastern division of the Congo Free State and the portion about the headwaters of the Lualaba and Lufira rivers, the center of the northern slope of the Congo-Zambesi watershed, is known as Upper (or Haut) Katanga, whence the name of the company to which title to the mineral belts has been granted—Union Miniére du Haut Katanga. This is a Congolese limited liability company with head offices in Brussels, Belgium, organized with 100,000 dividend shares, having no described value, issued in consideration of work done, 60 per cent. to the Katanga Special Committee of Brussels, and 40 per cent. to Tanganyika Concessions, Ltd., of London; and also

100,000 capital shares of 100 francs ($20) each; 50,000 shares subscribed by the Société Generale de Belgique and 50,000 shares by Tanganyika Concessions. None of the shares are negotiable until after the company has issued its third annual financial statement and the Tanganyika Concessions' holdings, except 2000 shares reserved to commute prospectors' rights, are pledged as part of the security on its issue of $10,000,000 of 5 per cent. bonds. The grants to the company are enormous, comprising all the copper beds, without exception, in the copper belt; all the tin beds, without exception, in the tin belt; the Ruwe gold mine, the iron beds and generally all the mining beds discovered by Tanganyika Concessions before Dec. 9, 1906, and also the most complete rights to use the necessary resources of the country for the working of the mines.

Commencing at about meridian 28-deg. East, between parallels 11 and 12-deg. South, the copper belt extends for nearly 200 miles toward the northwest, and in this distance there are over 100 known outcropping copper mines, upon which natives have worked. Nothing has been done



THE COPPER DEPOSITS OF KATANGA

to open new orebodies, for the known quantity is ample for years ahead. While the mines vary in size and grade there is a marked similarity of ore occurrence throughout, and this is of an unusual character. There is no gossan, no overlay, no barren zone; the ore comes in strength to the surface as high-grade oxidized copper ore — a little chrysocolla, a little azurite, but almost entirely malachite. There is no evidence that these ores resulted from alteration of sulphides *in situ*, and with the exception of one piece of chalcopyrite picked up on an old dump at the Luushia mine in 1902, I have never seen a bit of sulphide copper ore from Katanga.

There is upon each mine a wide core of what is termed "porous quartz ore" and on each side of this is found a wide layer of "laminated sandstone

ore," these terms being used merely to designate materials. In the porous quartz ore, malachite is found in the vugs and seams while the rock itself is barren, and as a result this material lends itself well to cheap hand sorting to eliminate a large portion of its silica excess. In the laminated ore, seams and fine threads of malachite are ribboned with equally fine seams of the matrix which, generally, is itself also impregnated and does not admit of much mechanical enrichment.

Except in about 40 miles between the rivers Dikulwe and Lualaba and in the Dikurwe area at the extreme west where the quartz ore is poor, there is little, if any, difference of grade in the two varieties of materials. Although statements may have been made to the contrary, there is no evidence to my knowledge showing any fading away of value toward the edges of the layers, the ore holding up in grade clear across to the barren country rock; but different strata within the orebodies vary much in copper contents, this variation not depending upon the location of the strata with reference to the outside rock.

As to the width of the orebodies the following statement is taken from the prospectus filed with the Registrar of Joint Stock Companies in London in November, 1906, when Tanganyika Concessions made a public issue of $10,000,000 5-per cent. debentures, the whole of which were underwritten: "Copper—In the western half of the copper belt, that is in the section lying to the west of the Lufira river, 26 crosscut tunnels have been run in ten of the deposits, giving average backs of about 100 ft., which will give a sufficient mining tonnage for a great number of years. These tunnels have exposed widths of ore as follows: Kabolela area, two tunnels, 130 to 150 ft.; Kakanda area, two tunnels, 130 to 150 ft.; Fungurume area, two tunnels, 120 ft., one tunnel, 130 ft., orebody not cut through; Kwatabala area, six tunnels, 20 to 40 ft.; Pumpi area, one tunnel, 60 ft.; Musonoi area, two tunnels, 190, and 242 ft.; Dikurwe area, three tunnels, 70 ft.; Kolwezi area, one tunnel 258 ft., orebody not cut through; Likasye area, one tunnel, 3373 ft., orebody not cut through; Kambove area, five tunnels showing great widths of ore, the longest being 401 ft., with orebody not cut through.

" On each of these areas the orebodies show on the surface, being generally covered by native pits. On June 8, 1905, the official engineer of the Congo Free State, who spent over 18 months in the district during the time these developments were being made reported there was shown ore containing two million tons of copper. The engineers of the company have also examined the work and reported similar results."

The group of mines between the Lualaba and Dikulwe rivers run from 6 to 8 per cent. copper, with 50 to 60 per cent. silica excess, and are out of consideration at present. However, as they contain millions of tons of such ore which can be largely quarried and with fine running streams

right at hand, it is very certain that in the future, as occasion requires, methods will be worked out for their profitable treatment. On the remaining properties, there should be little difficulty in maintaining for years an average of from 12 to 15 per cent. Three of these have had special consideration because of their size and situation: Star of the Congo at the extreme eastern end; Kambove, the premier mine of the belt, near the center; and Kolwezi at the extreme western end where the Benguella railway will enter.

The Star of the Congo is about 40 miles from the Rhodesian frontier in a flat country where shafts will be necessary. Five prospect shafts have been sunk showing the mine to be divided into two parts, of which the northwestern is the more important. Here three crosscut tunnels give an average width of 96 ft. of high-grade ore for a length of 2000 ft., or, so long as it holds, about 1,250,000 tons for each 100 ft. in depth, and some of the shafts are deeper than 100 ft. The ore lies in four parallel adjoining longitudinal layers. Sorting tests have been made, using native labor, and it has been found that for from 10 to 20c. per ton much of the silica can be eliminated and the greater portion of the mine can be made to yield a high-grade nearly self-fluxing ore. Limestone occurs at the mine.

Kambove, near the center of the belt, shows an astonishing body of high-grade ore. It lies in a swale between rounded hills and it can be worked open-cast. The main orebody, which courses east and west with dip to the north, is exposed for a length of nearly 3000 ft., but the western 1500 ft., where the natives sank their little pits, has been the only portion opened. Here seven shafts have been sunk to the 100-ft. level and five crosscuts extended from them, no one of which at my last account had reached any limit of the orebearing ground. The most westerly crosscut shows 239 ft. of ore, of which 172 ft. is high-grade. The most easterly cut is 297 ft. long, of which 96 ft. is high-grade. Between these extremes the other three cuts show great width of beautiful ore, the longest being 401 ft. in first-class material for the full distance and both faces in solid ore. Millions of tons can be quarried at this mine, and by care with easy sorting a product averaging 15 per cent. copper and from 20 to 30 per cent. free silica can be sent out for treatment. Large bodies of limonite and limestone are close at hand.

Kolwezi is also a large and attractive property. It lies along the side of a low hill all over which the natives have worked for a length of 1500 ft., and while the ore narrows at each end of the lens, for a distance of about 800 ft. it is fully 400 ft. in width. One of the native works is a trench more than 1000 ft. long, 15 ft. wide and 10 ft. deep, all in ore. From a little shaft sunk 10 ft. deep in this trench a crosscut has been run south exposing 258 ft. of high-grade ore, and there is about as much more

lying to the north. Last season a bore hole was put down 127 ft. below this tunnel level and cut with regularity the strata outcropping with their dip to the north. The Kolwezi, as it stands, is easily a 15 per cent. mine and by a little sorting the ore can be kept under 30 per cent. free silica.

The fluxes in the country are all or nearly all barren; the ore carries precious metals only in quantities too small to be of commercial value and it is all oxidized and unsuitable for water concentration. Still, so cheap is the mining and so high-grade the ore of the mines that copper should be turned out at an exceedingly low cost. In 1902 my estimate was that, given adequate transport and a suitable central reduction plant, copper should be made at cost of £12 ($60) per ton. The specialist now in charge for the Union Minière has reported that he will be able to treat the ore at a cost not to exceed £10 ($50) per ton of copper. The ores carry no bismuth, arsenic, antimony, lead, zinc or any of the base metals.

No doubt now remains as to these ores being readily smelted to black copper in water jacket furnaces. The present idea is to smelt 3000 tons of 15 per cent. ore daily, an amount the belt can steadily supply, and it is upon that basis that operations are outlined. The great questions now are those of transport and fuel supply. Railway communication is absolutely essential for further serious work and already three lines are headed for Katanga. The nearest known source of coke supply is at Wankies on the line of the Rhodesian road between 500 and 600 miles south of the Congo border, but coal is now reported as existing about 300 miles further north and this is being examined. Further, near the Lualaba river, thin outcroppings of poor coal have been found. The coal in each case was poor, dull and shaly. It is reported, however, that there are indications of this being the edge of a coal region further west, but of this nothing is known for no one has ever been there to see. It is in the air, though, that coal will be found west of the Lualaba. Meantime, all estimates have been based on European coke which can be supplied at a price.

These great deposits of oxidized copper ores are found for 200 miles along a sandstone country, sometimes dipping with the sedimentary layers, at others cutting them almost at a right angle; here folded as an anticline, there as a syncline, but always as beautifully stratified and laminated as if they were themselves sedimentary deposits. Nature has played a most unusual but very regular and orderly prank in the Katanga copper fields. The natural conditions in Katanga are superb. It is a gently rolling and undulating country, with an elevation of from 4000 to 5000 ft., and is remarkably healthful for Africa. The climate is comparable to that of Arizona and New Mexico. The annual rainfall averages 50 in. and is confined to the summer months, October to April.

Large rivers with many branches flow through fertile valleys and the whole country, except the copper hills and glades along the streams, is wooded, the trees being acacia, mimosa and other African varieties good for some purposes, but not for lumber.

Germany.—The production of copper in this country was about the same in 1907 as in the previous year. The output of Mansfeld showed a small decline, but the other German mines more than made it good. About 85 per cent. of the German production is still furnished by Mansfeld. The present practice in mining and metallurgy at Mansfeld was described by P. A. Wagner and J. S. G. Primrose in the *Engineering and Mining Journal,* Oct. 12, 1907. The workings of the mines are naturally becoming deeper, and a new row of deep level shaft, which will intersect the copper schist at 600 to 700 m., is already partly completed. The shafts at present in operation average about 400 m. in depth. In the new mines the underground haulage will be done by electric locomotives, instead of by horses as at present. The present tendency at Mansfeld is to electrify everything. Various novelties are being introduced in the metallurgical processes, including the bessemerization of matte and the introduction of the new Günther process for electrolyzing matte directly. This process has been developed with the view of producing pure copper from a matte containing 75 per cent. The matte is cast into anodes by a secret process. Practically pure copper is deposited at the cathode. Iron, nickel and cobalt go in the solution, while silver and sulphur, together with small particles of the anodes and metallic copper, collect at the bottom of the vats.

German East Africa.—A discovery of carbonate ore was reported at Udjidji, on Lake Tanganyika.

German Southwest Africa.—The Otavi Railway Company has completed a railway of 2-ft. gage, 351 miles long, from Swakopmund on the Atlantic Ocean, to Tsumeb. This is believed to be the longest railway in the world of so narrow a gage. The ore deposits at Tsumeb are owned by the Southwest Africa Company. The ore deposits are described as elongated replacement deposits in limestone. At the outcrop the orebody is about 180 m. wide, while at the third level, 70 m. deep, it shows a greater development. The thickness varies between 10 and 17 m. The late Christopher James, who opened the mine in 1900, estimated that above the 50 in. level there was about 293,000 tons of ore, averaging 12 per cent. copper and 25.3 per cent. lead; and about 191,000 tons assaying 2.9 per cent. copper and 4.4 per cent. lead. The lead and copper cannot be separated mechanically, and it is proposed to smelt the ore for work-lead and copper matte, which will be shipped to Europe for refining. Smelting operations were expected to be under way before the end of 1907.

Great Britain.—The production of copper by the mines of this country

is insignificant, but the production from foreign ores is still large. Swansea has been a metallurgical center for over three hundred years, though its industry did not become of any special importance until the Cornish copper ores began to be produced in large quantities, about 150 years ago. Cornishmen, such as the Vivians, Williamses, and others, then came over and established works to treat their ores. Afterward English firms such as the Elkingtons and Grenfells, established works in the district. For a long time the industry flourished and Swansea practice became a standard. Copper works abounded in Swansea, and in the valley behind where the villages of Landore, Llansamlet, and Morriston, are situated. Further to the east, works were established at Briton Ferry, Neath, and Aberavon, and to the west at Llanelly and Pembrey. In those days the ores were roasted in the open, and the desolation of the district was proverbial. About 50 years ago the supply of Cornish copper ores began to decline, but about the same time large quantities of ore and matte began to come in from North and South America, South Africa, Australia, and other places, and for another 20 years the copper industry continued to be prosperous. Ever since then the supply of copper ore has been declining, and the custom smelters, which are still in existence cannot obtain as much as they want. At the present time the only custom smelters treating copper ores that have survived are Vivian & Sons, at Hafod, Williams, Foster & Co., at the Morfa works, both at the back of Swansea, and Elliotts' Metal Company, at Pembrey. There are two other copper producers there, the Cape Copper Company at Briton Ferry, which treats ores and mattes from its own mine, and from the Namaqua and Tilt Cove mines, and the Rio Tinto company, at Port Talbot, where the company's bessemer bars and precipitate are refined.

India.—Deposits of copper and lead ore are being developed by Burn & Co. and P. C. Dutt in the Jubbulpore district, Central Provinces, 1.5 miles from the Sleemanabad station of the East India Railway, and 40 miles from Jubbulpore. The minerals which haye been identified are: Chalcopyrite, tetrahedrite, galena, pyrite and barytes, with the usual oxidized gossan accompanying these minerals. The country rock consists of dolomite with occurrences of silicious slate and limestone of probably the same geological age as the Dharwar schists of the Kolar goldfields. The series forms a belt, with a maximum width of seven miles, and extends for 20 miles in a northeast-southwest direction. Within this belt are the deposits at Sleemanabad.

Italy.—The copper production of this Kingdom continues small. The Société des Mines de Montecatini, which has been developing the copper mines at Val di Nievole at a very large expense, has abandoned the undertaking.

Jamaica.—Deposits of copper ore occurring in the parish of Clarendon received some attention in 1907. Numerous attempts to work mines in this island have been made during the last 40 years, but heretofore without success.

Japan.—The copper production of this Empire increased largely in 1907, in fact attaining the maximum in the history of its industry. This happened in spite of serious labor troubles at the Besshi mines, where strikers burned mine buildings and destroyed much property.

Mexico.—The copper production of Mexico decreased materially in 1907, chiefly on account of the readjustments at Cananea which were going on in the early part of the year, and finally the complete suspension of operations there in the autumn. As a consequence of these developments, the Greene-Cananea company produced only about 34,000,000 ℔. of copper. The Teziutlan company, of Puebla, also suspended operations late in the year. This company is erecting a new smelting works, which is expected to be ready for operation during the summer of 1908. The Moctezuma Copper Company, of Nacozari, Sonora, and the Compagnie du Boleo, of Lower California, made about the same production as in the previous year. On the basis of the net imports into the United States plus the Boleo production, the production of Mexico is estimated at 126,710,000 ℔, in 1907 against 135,800,000 ℔. in 1906.

The Greene Consolidated Copper Company in 17 months ending Dec. 31, 1907, treated 1,347,054 tons (wet) of ore, yielding 60,287,121 ℔. of pig copper, containing 58,180,856 ℔. of refined copper, 766,422 oz. silver and 6100 oz. gold. Of the total bullion product, 54,574,551 ℔. were from the company's own ore and 5,712,570 ℔. were from custom ore. The recovery per ton from the company's own ore was 46.58 ℔. copper, and the value of gold and silver was $734,024 or $24.266 per ton of refined copper produced. The receipts from copper, gold and silver were $11,360,387; the total expense of production was $11,040,420. The average price received for refined copper, delivered, was 18.588c. per ℔.

According to the reports of the Cananea Consolidated Copper Company (Greene-Cananea) for 1907, the slicing system of mining was introduced in the latter part of 1906, and immediately showed a decided saving over the old method of square-setting. In January, 1907, the cost of mining was only 87 per cent. of the cost for August, 1906, and in September, 1907, it was only 63 per cent. of the cost 12 months previous. At the same time the reconstruction of the smelting works has been going on, the eight old furnaces being replaced by eight new ones of larger capacity and more modern design. These improvements will greatly reduce the cost of producing copper and enable the mines to be worked profitably.

The Moctezuma Copper Company, of Nacozari, Sonora, has been building a new concentrating mill which will materially increase its production. The Compañia Metalurgica y Refinadora del Pacifico (Douglas Copper Company) built a smeltery at Fundicion, Sonora, which was expected to go into operation early in 1908.

In 1906, the latest year for which an official report is available, the Compagnie du Boleo produced 11,000 metric tons of copper. The mines produced 304,940 tons of ore, and the smeltery treated 302,499, the average yield of copper being 3.636 per cent. The smelting works are being reconstructed, the 10 old furnaces being replaced by nine of larger capacity and more modern construction. Six of the new furnaces were put in operation during 1907. The number of men employed by the Boleo company is 2500 to 3000, of whom about 200 are Chinese. Difficulty was experienced in retaining the men, owing to the great demand for labor in the United States, and consequently wages were raised 5 to 50 per cent. The copper production of Boleo in 1907 was 10,975 long tons. Since 1893 there has been but little change in the annual production of this company. The company's holdings consist of 11 groups of mines, most important of which are the Soledad, Providencia, Purgatoria and Las Briscas. Twenty miles of railroad connect the mines with the smelter at Santa Rosalia, and a fleet of vessels belonging to the company connect that port with Guaymas, Sonora.

Norway.—The production of copper in this Kingdom showed a small increase in 1907, both on the part of the Sulitelma mine (the chief producer) and from the outside mines. An American company has been developing copper mines at Trondhjem.

Peru.—The production of copper in this country increased largely in 1907, particularly because of the increased operations of the Cerro de Pasco company. According to advices from Cerro de Pasco in October, the company then had two furnaces in operation, each smelting about 250 tons of ore per day. The combined production was nearly 50 tons of blister copper per day, or about 2,500,000 lb. per month. Consequently, the ore was yielding nearly 10 per cent. copper. These two furnaces were in steady operation during 1907, and it was expected that a third furnace would be blown in before the end of the year.

Portugal.—In this country, where the Mason & Barry company is the principal producer, there was a small increase in the production in 1907.

Mason & Barry, Ltd., in 1907 produced 227,739 tons of ore, against 218,217 in 1906. Shipments in 1907, inclusive of ore from the cementation works, amounted to 363,465 tons, against 353,273 in 1906. The quantity of ore sold and invoiced for its sulphur value in 1907 was 361,408 tons, against 350,759 tons in 1906.

Russia.—The production of copper in 1907 is estimated at 15,000 long tons, against 10,490 in 1906, this being the largest proportional increase reported for any country in 1907. This great increase is attributable chiefly to the extension of operations in Siberia, where several British companies have become interested. Not all of these companies have proved unqualified successes. Thus, the progress of work at the Julia mine, situated in Abakansk, Yenesei province, Siberia, and operated by the Yenesei Copper Company, Ltd., has been impeded by all sorts of troubles. The grade of the ore has not come up to early expectations, and scarcity of suitable fuel has made the treatment of the ore a matter of some difficulty. The ore cannot be counted on as higher than 3.3 per cent. copper. A water-jacket smelting furnace was erected early in 1907 and the ore was found to be much more silicious than was expected. This, together with the fact that charcoal has to be used as fuel, reduces the duty of the furnace from 100 tons a day to only 60 or 70 tons. Up to Oct. 15, 11,296 tons of ore had been treated, yielding 870 tons of matte, of which the copper contents are estimated at 40 per cent., giving 3.08 per cent. of copper per ton of ore smelted. The discovery is announced of coal seams in the neighborhood of the mine, so that in the near future the present difficulty in connection with fuel should be removed.

In the Caucasus, the Caucasus Copper Company, Ltd., completed its works, designed and erected by Knox & Allen, engineers, of New York, which was put in operation during the summer. This company has a large, flat-lying deposit, of extraordinarily silicious ore, containing about 3 per cent. of copper in the form of chalcopyrite, besides which pyrite is present. The ore is concentrated by crushing to 1.5 mm. size, roasting magnetically in furnaces of the McDougall type, and passing over magnetic separators of the Wetherill type. This mill is of 500 tons daily capacity. The concentrate, which is still highly silicious, is smelted in reverberatory furnaces fired with crude petroleum, which is conveyed to the works by a pipe line.

PRODUCTION OF COPPER IN RUSSIA.
(In poods. 1 pood =16.381kg. =36.114 lb.)

Years.	Ural.	Caucasus.	Siberia.	Kirghiz Steppe	Altai.	Finland.	Various.	Total.
1895	151,511	166,728	....	1,394	15,888	21,858	....	357,379
1896	167,574	149,698	....	1,868	13,239	23,640	....	356,019
1897	220,783	162,534	....	3,586	15.427	21,360	....	423,690
1898	236,863	173,993	....	2,440	16,341	15,445	....	445,082
1899	253,610	171,568	....	5,754	15,292	13,664	....	459,888
1900	241,148	227,079	....	11,273	11,322	13,354	....	504,176
1901	217,063	247,348	....	21,993	13,193	17,311	....	516,908
1902	279,135	213,273	....	25,238	7,431	13,231	....	538,308
1903	265,116	262,919	....	17,902	7,546	10,126	....	563,609
1904	265,915	296,666	....	30,513	7,344	....	....	600,438
1905	225,800	223,800	67,000	....	....	....	53,900	570.500
1906	288,600	232,300	40,500	....	....	....	74,700	636,100

There can be no doubt as to the possibility of extending the copper industry in Russia. There is an abundance of copper ore in the Kirghiz steppes—the irrigation of which and the provision of ways of communication for which is only a question of time—which is sufficient guarantee that the copper industry will flourish. The production of the three copper smelting works in the Urals in 1906 was 258,793 poods, against 223,883 poods in 1905. The increase was due to the Bogoslovsk works, the Nigni-Tagilsk and Verch-Izetsk works having made decreased outputs. A new smeltry, with capacity of 30,000 to 40,000 poods, is to be erected at the Blagodat mine, Peklevsk Kossel.

(By I. I. Rogovin.) The increase in the value of copper during the year 1906 resulted in unusual activity in the industry during 1907. Old plants were enlarged and new companies were organized, chiefly by foreigners. The total quantity of copper produced at the various smelters during 1905, 1906 and 1907 is shown in the accompanying table, the figures being in poods.

PRODUCTION OF COPPER IN RUSSIA
(In poods)

Year.	Works.				
	Ural.	Caucasus.	Altai, Siberia and Kirghiz.	Chemical and Refining.	Total.
1905	225,800	223,800	67,000	53,900	570,500
1906	288,600	232,800	40,500	74,700	636,100
1907	457,904	310,244	68,957	65,253	902,358

These figures show that the greatest increase was at the Ural works, the bulk of the production being credited to the Bogoslovsky Mining Company. The total consumption of copper during 1907 was 1,086,900 poods, the imports being 272,300 poods and the exports 87,800 poods. In 1907 there were 902,000 poods smelted into bars which, with the surplus remaining from 1906, 48,000 poods, made a total of 950,000 poods; of this quantity there were delivered to electrolytic works for refining, 68,000 poods, and 88,000 poods were exported. The consumption of copper bars was thus 794,000 poods. The consumption of electrolytic copper was as follows: Imported, 272,000 poods; refined from copper in bars, 38,000; refined from scrap, 30,000; total 340,000 poods. Thus 22 per cent. of the total consumption fell to electrolytic copper and 78 per cent. to copper in bars.

The great increase in the production of copper during 1907 caused some fear of over-production which, at the prices prevailing at the end of the year, would have been disastrous. However, this fear was removed through the recent organization of a syndicate which now controls over 80 per cent. of all the copper smelted in Russia. Some of the English concerns in the Semipalatinsk region of West Siberia, among others the

Spassky Copper Company, accepted the conditions of the syndicate without becoming parties to it; this added 10 per cent. to the quantity controlled by the syndicate. Moreover, the syndicate buys the entire output of His Majesty's Cabinet, which added 3 per cent. more. Thus the syndicate controls all but 6 or 7 per cent. of the total output.

The total capacity of the Russian works is 1,415,000 poods of copper bars. If this figure is reached during 1908 it will mean an over production unless the importation of copper into Russia ceases. In fact the consumption of this metal in 1906 reached 1,415,000 poods, which compares with 1,624,400 and 1,459,000 poods in 1902 and 1903 respectively; on the other hand, there was imported in 1906 a total of 803,000 poods of electrolytic copper, as compared with 1,042,000 poods in 1904 and 1,139,000 poods in 1905. During the first nine months of 1907 there was imported 203,000 poods. This decrease in imports can be partly explained by the fact that on Feb. 16, 1906, the duty on copper was raised from 3.75 to 5 rubles.

The year 1908 will doubtless be a difficult one for the copper producers. It can hardly be expected that imports will cease, however greatly it may be desired, for the demand for electrolytic copper can not be met by the home production. It often happens that molds are required which are not employed in Russia. The problem which the producers have before them is to prevent the over-production of copper during a period of decreasing demand.

Servia (By Walter Harvey Weed).—Copper mining in Servia has been carried on for many years, though the amount produced has been small and no deep mining has been attempted. A few years ago the government, in order to develop the mining industry, adopted a new policy, granting concessions for large areas covering the more important deposits. The result of this work is now apparent; one of the two companies produced 1,257,289 lb. of copper in the last half of 1906, while the other company has completed its preparatory work and finished a reduction plant consisting of concentrating mill, smelting plant and acid works, and will be an active producer in 1907.

The mines are situated in the eastern part of the country, in the eastern spurs of the Balkan mountains, a region which has been devoid of railroad or water transportation except that afforded by the Danube river.

The two districts of greatest importance are Maidenpek and Bor, about 80 miles southeast of Belgrade. The Maidenpek district has long been known to students of ore deposits, having been described by von Cotta and other well-known writers many years ago. The village, near which the mines are situated and from which they take their name, is 10 miles from Milanovac on the Danube river. An overhead wire tram connects the mines and the river. The Belgian company which owns the

mines (La Société des Mines du Cuivre de Maidenpek) controls the entire district, having a concession of 18,800 hectares (46,455 acres). Under the old régime the ores, which averaged about 3 per cent. copper, were treated in blast furnaces producing a black copper, which analyzed 96 per cent. copper, 2.10 iron, 0.67 sulphur, 0.05 zinc, 0.06 nickel, 0.03 cobalt, traces of antimony and arsenic, and 13.214 oz. of silver and 1.244 oz. of gold per ton.

The district is underlain by gneiss, mica schist and amphibolite, forming the southern continuation of the crystaline schist belt of the Carpathian range. Argillaceous Paleozoic slate and Mesozoic limestone and dolomite cover the country generally, but in the mining district they are cut by igneous rocks, the oldest being serpentine and olivine euphatites with later andesites, this complex being cut and overlaid by dacites and micro-granite-porphyries which are genetically related to the ore deposits.

The ores contain chalcopyrite as the dominant ore mineral, but bornite, covellite and copper glance also occur. The gangue is quartose. Pyrite masses with admixed magnetite and chalcopyrite also occur, the ores carrying from 0.3 to 1.5 per cent. copper, which will be extracted after acid has been manufactured. The new works will treat about 120 tons of 3 per cent. ore in the furnaces and 280 tons of the pyritic ore will be treated each day in the lixiviation works.

The ores are found at the contact between andesite and either mica schist or limestone; but veins in the andesite also occur. Quartzitic deposits are found in a belt four miles long, with a coördinate width and depth.

The Bor deposits are in a district about 30 miles south of Maidenpek. The district is a volcanic plateau averaging about 1500 ft. above the sea. It lies in the midst of hills of Cretaceous limestone 800 to 3000 ft. high. This plateau includes a group of little villages, Bor, Zlot, Metovnica, etc., from which the deposits take their names. Throughout the entire district andesites of varying color, texture and mineral composition prevail. They are mainly amphibole, augite, or hypersthene andesites. The ores consist mainly of pyrite and chalcopyrite, usually in a hard, compact and uniform mixture, which fills the entire vein.

The Compagnie Francaise des Mines de Bor owns the St. George concession, which covers the communes of Bor, Krivelj and Ostrelj, comprising a 50-year concession for 2400 hectares and a renewable lease on 6500 hectares more. The ores occur in fissure veins traversing andesites and volcanic tuffs and breccias. The veins are wide, ranging from 50 to 103 ft. (at Cuka-Dulkan). They have a general north-south course and extend along a belt a mile or two wide and 10 miles long. They are from 1300 to 1967 ft. apart, but are not traceable for more than a few hundred feet along their course. The veins are usually found at the contact between solid, unaltered andesite, and propylite, the altered form. This

latter rock covers large areas. The vein filling at Bor and at Krivelj consists of pyrite and of chalcopyrite, with covellite-bornite, chalcocite and enargite, as well as small amounts of galena and blende. The ores contain small but recoverable amounts of gold and silver. Rich ore also occurs in fragments and balls scattered through a mass of leached kaolinized rock.

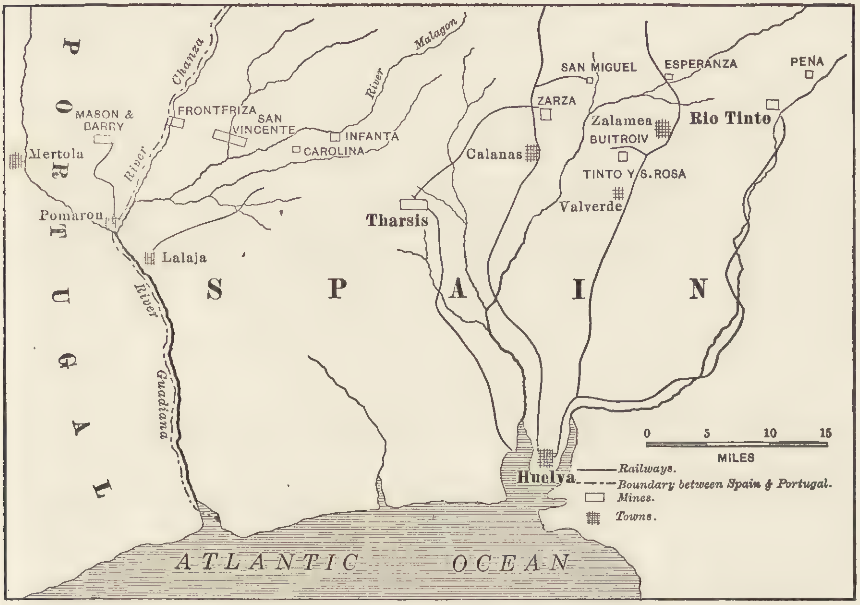
As indicating the size and richness of the deposits, D. Iovanovitch gives the results of the detailed sampling of several veins.[1] That of Cuka-Dulkan gave 6 per cent. copper with 29 to 75 per cent. silica and 11 to 23 per cent. iron for the average of all samples taken 1 m. apart across a vein 27 m. wide. At Krivelj the vein is 12 m. wide and carries an average of 5.5 per cent. copper as bornite, glance and chalcopyrite. At Crveno-Brdo there are three irregular ore chimneys and the deposit has been developed for a length of 250 m. and a depth of 70 m., showing an average width of 26 m. and a content of 7 per cent. copper. Limonite-gossan outcrops are common in almost all the ravines of the region. The new Bor reduction works contain two Fraser & Chalmers blast furnaces, four converters and 360-h.p. Babcock & Wilcox boilers. A branch of the Budapest-Constantinople Railway is under construction to the district.

Spain.—The production of copper in this Kingdom in 1907 was about the same as the previous year. The two big producers, viz., Rio Tinto and Tharsis, both showed decreased outputs, but there was an increase from the smaller mines, which was considerable in the aggregate. Several new companies were organized to operate in the Huelva district, among others the Société des Mines de Cuivre de Campanario (capital 5,000,000 francs) to exploit mines in the commune of Valverde del Camino, and the Mines de Cuivre de Nerva (capital 20,000,000 pesetas) to exploit mines in the same commune.

The Rio Tinto company in 1906 produced 34,098 tons of copper, of which 21,287 tons were extracted at the mines and 12,811 tons were contained in pyrites shipped. In 1907 the production was 21,251 tons at the mine, and 11,066 tons in pyrites shipped, a total of 32,317 tons. The decrease was chiefly the result of the three years of drought in Spain and the shortage of water at the mines. This drought has been broken during the winter just closed. The pyrites mined in 1906 was 1,923,716 tons and 1,906,948 in 1907; but the average copper contents increased from 2.411 per cent. in 1906 to 2.417 in 1907. The tendency is now for a larger proportion of the ore to be treated locally than formerly, and the smelting plant has been increased to carry out this policy. The directors say for 1907 that, except for the decline in the price of copper, they are able to report their entire satisfaction with the company's affairs in general and its prospects for the future. It is noticeable that the report for 1907 does

[1] "Or et Cuivre en Serbie orientale," Paris, 1907.

not give the average market price of copper for the year. In the 1906 report the average price was given as £87 9s. 3d. per ton. According to Merton's statistics, the average price for standard copper in London during 1907 was £86 4s. 2d. per ton. The Rio Tinto reports do not disclose the cost of producing the copper, but by deducting the dividends from the value of the copper sold, calculated on the average value of standard copper, and dividing the figure obtained by the tons of copper sold, the cost can be arrived at approximately. On this basis the Rio Tinto copper



THE COPPER MINES OF SPAIN AND PORTUGAL

cost £23 per ton, in 1906 and £31 in 1907, corresponding to 5c. and 6.74c. per ℔. respectively. According to these figures the mine is in a very comfortable position, as few, if any, mines of importance can produce copper at so low a price. It is to be borne in mind of course that the company derives a large income from the sulphur contents of its ore, which correspondingly reduces the cost of the copper.

The Tharsis Sulphur and Copper Company in 1907 obtained 81,034 tons (2240 ℔.) of ore from the Tharsis mines (against 95,409 in 1906) and 376,658 tons from the Calañas mines (against 342,348 in 1906). The total extraction of ore in 1907 was 457,692 tons, against 437,757 in 1906. The total shipments were 415,169 tons, of which pyrites (including washed mineral) amounted to 406,015 tons, and precipitate 2753 tons. The production of refined copper was 4410 tons, against 4739 in 1906. The gross

profit was £323,737, against £377,152 in 1906. The net profits were £261,023 and £315,147 in 1907 and 1906 respectively.

The San Vicente mine was floated in London, as the Anglo-Spanish Copper Company, by the instrumentality of Henry Bath & Sons, metal merchants of London and Liverpool. It is estimated that the known and probable ore so far developed at San Vincente, amounts to over 200,000 tons, with practical certainty that the bodies continue in depth. It is proposed to erect a smelting works capable of dealing with 200 tons a day, working on ore that will average 3½ per cent. copper. The lower-grade ore averaging 2 per cent. or so will be exported to alkali works, at the rate of 100 tons a day.

The Copper Markets in 1907

New York.—The enormous consumption of copper which marked 1906 continued into 1907, and until spring the market was strong and active. The outlook was promising, and manufacturers the world over, anticipating a large business and a continued short supply of the metal, bought heavily for future delivery. In the spring a severe decline in securities occurred, and sentiment throughout the United States, which had heretofore been very optimistic, became cautious and doubting. This kept buyers out of the market. The feverish activity, which had run for several years, abated, and the business which had been anticipated was not forthcoming. In consequence, manufacturers kept out of the market entirely, and consumption was reduced so suddenly and so sharply that the supplies bought early in the year lasted many months longer than had been expected, and it was not until the fall, after the price of copper had been cut in half, that the market became active again.

Meanwhile, producers had accumulated stocks variously estimated at from 200,000,000 to 250,000,000 ℔., and they decided, owing to the low price, to curtail production. By that time manufacturers everywhere had worked off their stocks and were carrying less copper than a normal supply. The cheapness of the metal was apparent, and a buying movement set in, which absorbed the bulk of the stock in the hands of producers, the market advancing to about 15c. Thereafter there was a slight reaction to about 13c., but as the mines, wisely recognizing the decrease in consumption, showed no signs of reopening, the market continued firm, and at the close of the year appeared fairly strong and advancing.

January opened with Lake copper selling at 24c. and electrolytic at 23⅝c. Transactions were small, but owing to the scarcity of the metal, prices advanced about 1c., and stocks of standard copper in warehouses abroad decreased to about 3500 tons. In February a shortage of fuel interfered with production, and the prospects for a large consumption were good. In consequence, American consumers bought heavily, covering their

expected requirements as far ahead as June, some sales being made for July delivery. This advanced the price still further, Lake copper selling at 25½c. and electrolytic at 25c. During March a large sale of Lake copper took place at 26c., and electrolytic sold at 25½c. European buyers, who had generally held aloof from the market during January and February, now came in and bought heavily. Toward the end of March, however, the market was checked by the large decline in securities. Buyers held back and sellers met the market more freely. This caused the premium which had ruled for spot copper to disappear. In sympathy with the liquidation in the share market, large quantities of standard copper were sold, which resulted in a decline to £92 at the end of the month. Meanwhile, the statistics for the first three months of 1907 showed an increase in the imports of about 6000 tons and a decrease in the exports of 9000 tons, making a total increase in the available supplies of 15,000 tons. However, up to that time the consumption here was so enormous that these additional supplies were readily absorbed.

In April the market became rather unsettled. The largest American selling interests held for 25c., but made no sales, as owing to the decline in standard copper to £92, that copper was imported to this country, refined into electrolytic copper, and sold at 24c. During May the market was completely stagnant. In London standard copper declined sharply, particularly for three months'. Prices for spot copper were maintained on account of the depletion of the stocks and the scarcity of warrants. A backwardation to £3 occurred. By this time some of the large American consumers became nervous about the outlook and resold some of the copper previously purchased. During June, although the largest sellers were still holding for 25c. for electrolytic copper, it was freely offered at 23c. without finding buyers. In London the backwardation increased to £4. There had been no buying on the part of American consumers since March, and everybody expected that their stocks would be depleted by July 1, and that they would have to come into the market for large quantities, which, however, did not prove to be the case.

On July 9 the United Metals Selling Company reduced its price for electrolytic copper to 22c., a drop of 3c. from that previously asked, thus approaching the point at which actual sales had been made by other interests. This, however, did not stimulate purchases, and the little business that offered was eagerly competed for. At the end of the month, Lake copper sold at 21c. and electrolytic at 20c. Meanwhile there had been a sharp break in the London market for standard copper, which closed weak at £86 for spot and £86 10s. for three months'. European buyers kept entirely out of the market. During August buyers continued to hold aloof and the market gradually declined to 18c. for Lake and 17½c. for electrolytic. It was confidently expected that by Sept. 1 manufacturers

generally would be compelled to buy, but sellers were again disappointed. Business had fallen off to such an extent that manufacturers had used much less copper than they had anticipated in the spring, and they were still working off the purchases then made. The little buying that took place was entirely from hand to mouth, and prices declined to 15c. for Lake and 14¾c. for electrolytic. Prices went lower from day to day, and and at the end of the month the United Metals Selling Company reduced its price to 15c. Conditions, generally, began to get worse and buyers who had been waiting for the 15c. price to place some of their orders were again content to look on and restrict their purchases to the narrowest hand to mouth basis.

Stocks had accumulated in the hands of producers and there was considerable pressure to sell, under which the market declined toward the end of October to 11⅞c. for Lake and 11⅝c. for electrolytic. In consequence of the stringency in the money market and the inability to finance further quantities of copper, many of the larger mines closed down, and European consumers, recognizing that copper was selling practically at cost, bought heavily. The Chinese also came forward as fairly large buyers. Immense transactions took place, amounting, in the aggregate, to upward of 150,000,000 lb., thus greatly reducing the accumulation. Under the circumstances, prices advanced by leaps and bounds, Lake copper selling at 15c. and electrolytic at 14⅜c. During November the market again lapsed into dullness. European buyers, having covered their requirements for the remainder of the year, held off, and domestic buyers were averse to contracting ahead on account of the disturbed financial

AVERAGE PRICE OF LAKE COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	16.33	16.08	16.55	16.94	16.55	16.00	16.16	16.58	16.69	16.64	16.80	16.88	16.52
1901	16.77	16.90	16.94	16.94	16.94	16.90	16.51	16.50	16.52	16.60	16.60	14.39	16.55
1902	11.322	12.378	12.188	11.986	12.226	12.360	11.923	11.649	11.760	11.722	11.533	11.599	11.887
1903	12.361	12.901	14.752	14.642	14.618	14.212	13.341	13.159	13.345	12.954	12.813	12.084	13.417
1904	12.533	12.245	12.551	13.120	13.000	12.399	01.505	12.468	12.620	13.118	14.456	14.849	12.990
1905	15.128	15.136	15.250	15.045	14.820	14.813	15.005	15.725	15.978	16.332	16.758	18.398	15.699
1906	18.419	18.116	18.641	18.688	18.724	18.719	18.585	18.706	19.328	21.722	22.398	23.350	19.616
1907	24.825	25.236	25.560	25.260	25.072	24.140	21.923	19.255	16.047	13.551	13.870	13.393	20.661

AVERAGE PRICE OF ELECTROLYTIC COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	15.58	15.78	16.29	16.76	16.34	15.75	15.97	16.35	16.44	16.37	16.40	16.31	16.19
1901	16.25	16.38	16.42	16.43	16.41	16.38	16.31	16.25	16.25	16.25	16.22	13.82	16.11
1902	11.053	12.173	11.882	11.618	11.851	12.110	11.771	11.404	11.480	11.449	11.288	11.430	11.626
1903	12.159	12.778	14.416	14.454	14.435	13.942	13.094	12.962	13.205	12.801	12.617	11.952	13.235
1904	12.410	12.063	12.299	12.923	12.758	12.269	12.380	12.343	12.495	12.993	14.284	14.661	12.823
1905	15.008	15.008	15.125	14.920	14.627	14.673	14.888	15.664	15.965	16.279	16.599	18.328	15.590
1906	18.310	17.869	18.361	18.375	18.457	18.442	18.190	18.380	19.033	21.203	21.833	22.885	19.278
1907	24.404	24.869	25.065	24.224	24.048	22.665	21.130	18.356	15.565	13.169	13.391	13.163	20.004

conditions, although they purchased more or less to supply their immediate requirements. In December the demand for export continued and domestic consumers bought more freely. The year closed with Lake copper at $13\frac{3}{4}$ and electrolytic copper at $13\frac{1}{2}$ cents.

London.—January opened with business restricted by the holiday season. An initial advance of 5s. was lost after publication of the fortnightly statistics showing increase of 1866 tons in the visible supply, which led to heavy realizations and bear selling. Trouble in the money market led to further realization about Jan. 9, but the market was steadied by important purchases for American account. Fully 2000 tons of unrefined copper were removed from English warehouses for shipment to America, while consignments of material from Chile, Japan and Australia were diverted to American ports. On Jan. 14 cash standard sold at £109, the highest for the month, which closed at £106 for cash warrants and £107 for three months'. February opened with an advance of £1 per ton, induced chiefly by the increase of 3470 tons in the visible supply during latter half of January, which was due to the heavy withdrawals from English warehouses for shipment to America. Heavy buying raised the price to £109 for three months' warrants on Feb. 4, after which bulls and bears contended with moderate fluctuations. Later in the month the placing of some large contracts enhanced the value of refined copper, both for prompt and future deliveries. Toward the close of the month there was evidence of producers' inability to meet the trade requirements and speculation revived. The closing prices were £$108\frac{3}{4}$ for cash warrants and £$109\frac{3}{4}$ for three months'. March opened with every prospect of improvement in trade. Statistics showed a further diminution of stocks, refined sorts being particularly scarce, and new orders coming in to an extent which caused anxiety as to the sufficiency of supplies for April and May requirements. The severe slump in American securities early in the month had little effect on copper beyond the liquidation of some bull accounts, the market being evidently oversold. But renewed trouble in American finance led to a reaction and on March 26 three months' warrants touched £97. There was a sharp recovery, but the month closed at £$97\frac{1}{2}$ for cash warrants and £$99\frac{1}{2}$ for three months'.

April opened hopefully but adverse conditions immediately prevailed and there was somewhat of a panic in which prices fell to £93 for delivery late in June and £92 for April dates. Bear covering led to a recovery, a corner being threatened, and on April 10, prices had risen to £$99\frac{5}{8}$ both for spot and forward. Then followed reports of lower prices in America, but there was a steady advance in London up to the end of the month, the closing being £$104\frac{1}{2}$ for cash warrants and £103 for three months'. The producers were very firm and the Rio Tinto disposed of its entire production for May at £114. May opened with a strong tendency, the

statistics showing a further reduction in the visible supply, and prices rose to £108 for June warrants and £107½ for May dates, which was due to sudden panic among the bears. Realizations reduced the cash price by £2½, notwithstanding a large business in refined sorts. Thereafter prices drifted downward, chiefly on account of the unfavorable conditions in America, and the month closed at £101¼ for cash warrants and £98 for three months'. June opened with business restricted by persistent depression on the stock exchange and general uneasiness. Producers held firmly, but dealers were eager to secure orders from consumers. Stale bull accounts were liquidated and bears were encouraged to sell, whereby prices were depressed to £94¾ for cash and £91¼ for three months'. Bargain hunting, bear covering, and a revival in the demand for manufactured sorts effected a sharp recovery but free offerings of American copper created uncertainty. However, the month closed under improved conditions on the Stock Exchange, with consumers very barely supplied and English smelters obliged to encroach on stocks in public warehouses in order to meet the demand for refined sorts. The premium on prompt and early warrants increased throughout the month, which closed at £98 for cash and £91¾ for three months'.

July found consumers generally bare of stocks and obliged to buy. The result was an advance in price, but on July 9 came news of reduction in price by two of the leading American producers, which previously had been standing out of the market for higher prices than were accepted by other producers. This initiated a rapid decline in the London market which brought trade to a standstill. On July 16 cash warrants fell to £89 and three months' to £85½. The market became very sensitive and trade was restricted. The Rio Tinto company after vainly awaiting an opportunity to market advantageously its June and July output now sold moderate quantities at £98. Meanwhile the Calumet & Hecla was selling Lake copper in Europe at £100. This induced a general decline to £86½ for cash warrants and £83 for three months'. The month closed with a pessimistic sentiment. During the first fortnight of August there was a steady decline in prices, American producers making further concessions, and the situation being aggravated by something like a panic in Wall Street. On August 13 three months' warrants sold at £73½. By this time it became manifest that a large number of orders had been held back and toward the end of the month there was an active business in electrolytic sorts, but the standard market continued unsettled and closed at £76¼ for cash and £75½ for three months'. During the early part of September a large amount of covering in standard copper took place around £74, but the pessimistic sentiment prevailed and the decline could not be arrested. The constant cutting of prices in America encouraged bears to push sales in London. The decline continued up to Sept. 16

when there was a general rally, and a large business was done at about £67. Important orders were booked for India and China. The requirements of consumers in many cases were urgent and the stocks available showed considerable shrinkage, but the persistent decline in the American price was a disturbing factor. The London price suffered in consequence, notwithstanding the activity, and September closed with £64 for cash warrants and £63⅞ for three months'.

October will long be remembered for the acute financial crisis which developed in America and disturbed the prices of all commodities. At the opening of the month there was a general desire to liquidate holdings of all kinds. Consumption of copper showed signs of flattening and prices dwindled almost uninterruptedly down to Oct. 23, when cash warrants closed at £55½ and three months' at £55. During this period the London price was frequently lower than the New York parity and successive reductions served only to discourage enterprise. The accumulations of copper in America were regarded as sufficient safeguard against any sudden turn of the tide and consumers bought only to cover urgent requirements. A curious feature was the scarcity of spot warrants, which commanded a premium throughout, at one time as much as £2 per ton. The lowest price actually touched was £54½ for three months' warrants. The turning point was reached when the American producers threw over large quantities of refined copper at about £55, a price which at last attracted European dealers and tempted them into buying on a considerable scale. Orders which had long been withheld were now given out freely, Indian purchases being on an important scale and Chinese still more so. The fourth week of October witnessed the remarkable turnover of 15,000 tons. The close of the month saw the market active and bouyant, notwithstanding the stringency of the money market, the final price being £66 for cash and £65¼ for three months'. November opened with a sharp rise, but quietness soon followed. However, the market appeared hopeful in view of the large disposal of American accumulations and the reduced output of the mines, but prices were very sensitive to financial influences and receded to £57¼. In spite of the troubles in America, it was manifest that the European industries were well employed, the increased supplies being readily absorbed by the trade and a considerable business being done for deliveries extending well into 1908. Electrical enterprise showed marked revival, and the Indian trade developed a welcome activity. Large quantities of copper which would otherwise have gone to Europe were diverted to China. Thus the latter part of the month was distinctly hopeful, although the closing days witnessed some depression of prices due to profit-taking and occassional bear selling. The closing prices were £62½ for cash warrants and £63 for three months'. However, December failed to fulfill the hopeful prognostications, the

causes of financial uneasiness proving too deep-seated to admit of speedy removal. Consumers restricted their purchases to immediate requirements and these were unimportant. Manufacturers continued well employed on old orders, but found new business very small. The rapid shifting of American accumulations of copper to European ports caused a plethora of refined copper at Liverpool and elsewhere, with the natural consequence that spot warrants were pressed for sale and the contango widened to 30s. During the first week prices improved irregularly, but thereafter fell away, without attracting much business anywhere. On Dec. 12, £58¾ was accepted for three months' warrants, at which point there was a rally caused by a moderate influx of buying orders. The month closed firm on Dec. 28 at £62 for spot and £62¾ for futures.

AVERAGE PRICE OF STANDARD COPPER (G. M. B.'s) IN LONDON.
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901	71.78	71.17	69.54	69.61	69.60	68.83	67.60	66.34	65.97	64.11	64.51	52.34	66.79
1902	48.43	55.16	53.39	52.79	54.03	53.93	52.89	51.96	52.68	52.18	51.03	50.95	52.46
1903	53.52	57.34	63.85	61.72	61.73	57.30	56.64	58.44	56.82	55.60	56.30	56.36	57.97
1904	57.500	56.500	57.321	58.247	57.321	56.398	57.256	59.952	57.645	60.012	65.085	66.375	58.884
1905	68.262	67.963	68.174	67.017	64.875	65.881	66.887	69.830	69.667	71.406	74.727	78.993	69.465
1906	78.869	78.147	81.111	84.793	84.867	83.994	81.167	83.864	87.831	97.269	100.270	105.226	87.282
1907	106.739	107.356	106.594	98.625	102.375	97.272	95.016	79.679	68.375	60.717	61.226	60.113	87.007

Progress in the Metallurgy of Copper.

By L. S. Austin.

The following article is a summary of the important literature of 1907:

Ore and Matte Roasting.

Cost of Roasting in Hand Reverberatories.—A long-bedded hand-stirred furnace, 16 ft. wide by 60 ft. long, will have a maximum capacity of 16 tons of pyritic ore of 36 per cent. S. in 24 hours, roasting it down to 7 per cent., or 33 lb. per sq.ft. of hearth area. This will need two men per shift, or six men in 24 hours, as well as a certain proportion of the time of chemist, foreman, weigh-master and fireman, and will require three tons, or 18 per cent. of indifferent coal. To the above charges should be added the cost of repairs, tools, renewals, lights, etc. This makes the cost per ton of raw ore $1.50 at large works in the Western States.[1]

Heap-Roasting at Mansfeld, Germany.—The copper-bearing shale (*Kupferschiefer*) is roasted in heaps 130 ft. long, 7 ft. high, 16 ft. wide across the top, and 24 ft. across the bottom. The pile is kindled with brush-wood set around the base of the pile, and combustion once started the ore, containing as it does 13 per cent. of bituminous constituent, is self-burning. The lump-ore of the heap is covered with ragging, and the

[1] Peters, "Principles of Copper Smelting," p. 101.

fine is briquetted and used to form channels for the admission of air into its interior. Burning proceeds for two weeks with some sintering of the ore, and it takes another fortnight for the heap to cool.[1]

Size of Particles for Roasting.—While the finer the ore the more quickly will it roast, there are disadvantages which offset this. These are the cost of finer grinding, the increased flue-dust made, the tendency of fine ore to sinter, and the difficulty of the penetration of air below the immediate surface. Some pyritous ores decrepitate, even up to $1\frac{1}{2}$ in. in diameter, and some are easily penetrated by the air, while others resist oxidation to the last. Experience shows that when there are few grains larger than $\frac{3}{16}$ or $\frac{1}{4}$ in. diameter the bulk of the ore will be fine enough for roasting for subsequent smelting. Coarse concentrates often exceed this limit, and are yet freely added; since, while they receive an imperfect roast, they are associated with sufficient well-roasted fines to bring down the average to a working percentage. Massive copper-bearing sulphides in lump form are advantageously roasted in heaps or stalls, and thus kept in condition for blast-furnace smelting, or they may be smelted pyritically without a preliminary roast. Matte of from 20 to 40 per cent. copper should not have its larger particles more than $\frac{1}{6}$ in. diameter for suitable roasting. For the Ziervogel process, involving a delicate roasting operation, the matte must be ground fine. Matte of over 35 per cent. copper is better treated for the removal of its sulphur and iron contents by converting, and, when much below this figure, may be brought up to it by an oxidizing blast-furnace operation.[2]

McMurty-Rogers Process (*Pot-Roasting*).[3]—This modification of the Huntington-Heberlein process consists in treating ore as in that process, but without the use of lime, wetting it down so that it will cohere easily before putting it into the pot or converter. The minimum quantity of water to produce the best result is, for the lower grades of matte, 3 to 4 per cent., for an ordinary rich matte 4 to 6 per cent., and for ore 6 to 9 per cent. In the case of ore, it was found that unless the water was in excess, ferric oxide was produced, and this forming around the particles prevented proper slagging, whereas with sufficient water a ferrous silicate would be produced. The charge which works best consists of about one-third of pieces 1 to $1\frac{1}{4}$ in. diameter and two-thirds of fine concentrate. It should contain 15 to 35 per cent. SiO_2 and 15 to 25 per cent. of sulphur.

The converters, or pots, as used at the Wallaroo works, are 8 ft. 8 in. diameter, by 4 ft. 6 in. deep, with steep sides, and have a convex wrought iron false-bottom perforated by $\frac{5}{8}$ in. holes, and set 10 in. at most above the bottom of the pot. A blast-pipe of 8 in. diameter enters below the false-bottom. Such a pot will hold eight to nine tons of charge. It is

[1] *Eng. and Min. Journ.*, LXXXIV, 673.
[2] Peters, "Principles of Copper Smelting," p. 93.
[3] *Trans. I. M. M.*, XV, 311.

mounted on trunnions, and can be tipped or inverted by means of a worm and worm-wheel attachment.

In treating a charge, the false-bottom is covered with some roasted ore of 3 to 4 in. size, after which a small fire of chips and then sawdust is started, which is urged by a light blast. The blast is shut off and ore is charged, more deeply at the sides, until the pot is half filled. A number of holes are now pricked with a half-inch rod through which the sulphur vapor and sulphur dioxide, due to the blast, come up. In about an hour, with half the full blast, a ring of fire shows up, reaching inward one-half to one-third the diameter of the pot. The full pressure of 13 oz. per sq.in. is then applied, driving through about 1000 cu.ft. per minute. When the evolution of sulphur dioxide slackens, and the charge has become red-hot throughout, the blast is turned off, the blast-pipe disconnected, and the converter inverted. It should be noted that during these blowing-up operations, the pot is covered with a hood. The block or cake of roasted ore, of six tons or so in weight, is lifted by the crane and dropped upon some cast-iron cones, which by the force of the impact breaks the block.

The time taken to treat a charge varies from eight to twelve hours. The burning should be well attended to, blow-holes should be stopped or broken up as they form, and numerous holes made to permit the escape of sulphurous fumes. The product consists of a porous sintered mass of ferrous silicate, containing shots of rich matte and free silica (ferric oxide being observed on the outside of the cake), and it forms a most suitable material for blast-furnace smelting. Most of the iron, being in ferrous form, needs no further reduction.

A large quantity of such a product showed 5.65 per cent. sulphur, while as low as 5 per cent. has been obtained from ore originally of 20 per cent. sulphur and 10 to 12 per cent. copper.

The cost of treatment at the Wallaroo works, where 400 to 500 tons are converted weekly is 84c. per ton. In a mechanical roaster the ore can be roasted for 60c. per ton.

The process is particularly suited to the treatment of matte, 15 to 25 per cent. of silicious ore being added to unite with the iron as ferrous silicate. The matte should be crushed to $\frac{1}{2}$ to $\frac{3}{4}$ in. diameter or less. The silicious ore should be well mixed with the crushed matte, and the whole wetted down as previously specified. It has been found better to take more pains in charging matte than in charging ore, the matte being added gradually where the fire shows the most until the pot is full. With the matte it is not necessary, as with ore, to prick holes in the charge. The product of the reaction is ferrous silicate with some FeS, CuS, and $CuSiO_3$, as well as a little metallic copper. Six to eight ounces of blast is sufficient at first, but this may be increased toward the end to the full extent. On

treating matte of 25 to 30 per cent. copper with one-third of its bulk of silicious sulphide-slimes, a product was obtained, which upon melting down gave a 64- to 67-per cent. matte.

Satisfactory results were obtained in the treatment of white-metal for subsequent smelting to blister copper by the "direct" process. Whereas in ordinary roasting much difficulty is encountered because of the easy fusibility of this grade of matte, in the pot the troubles of calcining are largely overcome, and a more suitable product obtained for the reverberatory furnace; and being in lump form there is no loss by dusting. The matte is crushed to $\frac{5}{8}$-in. mesh and merely wetted down, no silicious-ore addition being necessary, since there is practically no iron to be taken care of. The pots used for this treatment are conical, 5 ft. 3 in. diameter at the top, $11\frac{1}{2}$ in. at the bottom, and 3 ft. $6\frac{1}{2}$ in. deep, and hold two tons of matte each.

To work this material a small fire is built upon the false-bottom, and upon it are placed lumps of well-burned product of a former operation. When these pieces have come to a low-red heat, and the fire has subsided, the wet white-metal is dumped in to half-fill the pot, a low blast of 6 oz. per sq.in. being then turned on. When this charge has been burnt through the remainder is added. The charges take eight hours to burn, the blast-pressure being 8 to 10 oz. per sq.in. only. The final product consists of porous metallic copper mixed with copper sulphate and some undecomposed matte. The mass, when dumped out, is broken into lumps by a weight of 1600 lb. dropping 20 ft. Experiments to determine the relative elimination of nickel and bismuth from white-metal gave results as follows: Welsh roasting process, 1 per cent. Ni, 1 per cent. Bi.; direct process, 1.5 Ni, 1.12 Bi; McMurty-Rogers process, 1.87 Ni, 1.39 Bi. The gas given off in the pot roasting process was found to be of the following composition: From ore—5.4 per cent. SO_2, 8 per cent. O; from ore—3 per cent. SO_2; from matte, 5.5 per cent. SO_2, 12 per cent. O; from white metal, average, 11.1 per cent. SO_2, 5.6 per cent. O.

It has been found possible to treat successfully ore of 5 per cent. sulphur and 5 per cent iron by the process, while were it attempted to do this by ordinary roasting, especially with fine ore, external heat would be necessary to keep the roasting going. If it be attempted to blow air through a mass of dry ore, much flue-dust is made, but when water has been added to the mass it can be desulphurized to any extent required and with great rapidity. Wet mixtures conduct heat badly and so keep up the temperature at the seat of combustion and cause this zone to move upward in a fairly horizontal layer. The products of combustion leave the surface of the charge at numerous points all over the surface. It is probable that some of the water present acts in some way as a carrier of oxygen from the hot to the cooler material, the effect being to increase slightly

the oxygen at that point and so increase the intensity of the combustion. Upon attempting to desulphurize dry white-metal in the pot but little action takes place; if moistened with water, metallic copper is readily produced. This reaction takes place a little below the melting point of copper, as is shown by the porous nature of the mass, which indicates that it has not been fused. Gautier[5] has recently observed this reduction of copper from cuprous sulphide, but at a white heat. He gives the reaction as taking place thus:

$$Cu_2S + 2H_2O = 2Cu + SO_2 + 4H.$$

When air is blown in, copper sulphate and copper oxides are formed in large amount, these reacting upon the decomposed sulphide according to the well known reactions,

$$Cu_2S + 2CuO = 4Cu + SO_2,$$
$$Cu_2S + CuSO_4 = 3Cu + 2SO_2,$$

and at the same time the copper does not become molten.

If the cuprous sulphide (white-metal) contains nickel sulphide very little nickel is reduced, though both nickel sulphate and nickel oxides are formed. In ordinary roasting some nickel is reduced, and even less in the bessemerizing of copper matte containing it. It seems that the temperature for the reactions

$$NiS + 2NiO = 3Ni + SO_2,$$
$$NiS + NiSO_4 = 3Ni + 2SO_2,$$

is well above the melting point of copper. Steinhart has pointed out that nickel sulphide cannot by bessemerizing be reduced to nickel, as nickel silicate is produced.[6]

By the McMurty-Rogers process the nickel sulphate and oxide readily form nickel silicate when the white metal containing them is melted down, a nickeliferous slag of higher tenor in nickel being produced than that in the white-metal.

When a mixture of iron and copper sulphides is pot-roasted, desulphurization proceeds rapidly if both water and silica are added, otherwise slowly. Gautier gives for these sulphides the reactions

$$3FeS + 4H_2O = Fe_3O_4 + 3H_2S + 2H,$$
$$2Fe_2O_3 + 7H_2S = 4FeS + 3SO_2 + 14H.$$

When air is blown in, it is obvious that both the hydrogen and the H_2S will burn.

In a well-burned charge of matte or ore, Fe_2O_3 is to be seen at the

[5] *Journ.* Chemical Society, 1906, p. 548.
[6] *Trans. I. M. M.*, XV, 223.

sides and top of the charge, where it has been cooled by radiation, while Fe_3O_4 reacts on the FeS thus :

$$FeS+3Fe_3O_4=10FeO+SO_2.$$

This reaction is exothermic and would, at a high temperature, form ferrous silicate, which again produces heat. Indeed, in action, this formation and consequent sintering can be seen spreading as the the burning proceeds.

In ores containing pyrite or chalcopyrite, the first equivalent of sulphur is first driven off. The reactions should proceed rapidly, so that a temperature can be maintained high enough to permit the formation of ferrous silicate; otherwise ferric oxide is formed, diminishing the amount of available active oxygen near it, the temperature drops, and a patch of unsintered material is formed. If much of this Fe_2O_3 is formed, it seems difficult to reduce it. The formation of ferrous silicate aids materially in producing a product low in sulphur as well as in sintering the material. It acts probably as a carrier of oxygen to the particles of ferrous sulphide present in it.

Savelsberg Process for the Pot-Roasting of Copper Matte[1].—This consists in roasting in charges the crushed matte in pots in mixture with silicious ore and lime with air driven through the charge under pressure, giving, as in the "lime-roasting" operations, a coherent and well-roasted product for the blast-furnace. The charge is dumped on a bed of incandescent fuel, or of hot ore, in order to start the roasting.

Reverberatory Smelting

Repairing a Copper Reverberatory Furnace[2].—Defective workmanship had caused the deep corrosion of the side walls of a reverberatory furnace of 100x20 ft. hearth dimensions, belonging to the Consolidated Arizona Smelting Company, Humboldt, Arizona. The buckstaves and tie-rods held the roof in position, while these walls were removed by drilling small holes into their hot foundations, cooling these holes with sprays of water, then blasting them with small charges of dynamite. This was done from the outside of the furnace, working between the buckstaves to do so. As fast as the walls were removed, new walls were built in their place, and thus the work proceeded until the walls were entirely and successfully reconstructed.

Forced or Undergrate Draft in Reverberatory Smelting.—Peters describes[3] the method for increasing the draft, which consists in closing the ash-pit and forcing air into it. He speaks of the delay due to the operation of grating, when it is necessary to open the doors, and of course to remove

[1] Brit. patent, No. 10,329, 1906.
[2] *Eng. and Min. Journ.*, LXXXIV, 1158.
[3] "Principles of Copper Smelting," p. 192.

the undergrate pressure. Forced-blast also induces intense local action, which causes the coal to melt in large, massive clinkers, troublesome to remove.[1]

Details of Reverberatory Furnace Construction.—The roofs of such furnaces, formerly 9 in. thick, are now made 12 in., and even 15 in. thick, to lessen radiation. Such heavy roofs are heavily ironed with buck-staves of 8 to 10-in. I-beams placed nearly touching. The lower ends of the buck-staves are held by heavy masonry, or against a heavy monolith of slag, poured into a trench made just outside the walls. The longitudinal expansion of the roof is allowed for by the use of expansion-joints transverse to the furnace at 10 ft. intervals, the joints being 2½ in. wide. The bottom of the older furnaces was sustained above a vault placed there to keep this part cool, but the more recent idea is to make a solid concrete foundation, upon which the hearth is built up.

Treatment of Flue Dust and Fine Concentrate in a Reverberatory Furnace.[2]—The Detroit Copper Company, Morenci, Arizona, is experimenting with a reverberatory furnace for the treatment of the flue dust and fine concentrate. The furnace is 50 ft. long, and is heated with fuel-oil. Slag is received into the furnace in a continuous stream from the blast-furnace, while the flue dust is added by the barrow or car-load. Though rapidly melted down, it forms a superficial layer, which while liquid enough to flow is more silicious and higher in copper than the slag below. The method involving the least labor and the most perfect mixture consists in allowing a stream of flue dust to strike the jet of flame of the burning oil. There is however a corrosive effect, due to the action of the slagging flue dust upon the roof of the furnace. Experiments have been made upon fine concentrates by throwing them upon the surface of the molten slag, and liquation proceeds rapidly, the contained sulphides melting at once, and leaving infusible silicious shells, which float toward, and tend to accumulate as accretions on the side-walls of the furnace. A low-grade matte, formed from the sulphides, drops to the bottom of the furnace, and is later treated in the blast-furnace. The highest temperature at the fire-box end is 760 deg. C., and at the flue end 655 deg. C. The actual costs are not given, but are thus distributed: Fuel-oil, 73.01 per cent.; labor, 17.35; repairs, 8.85; shop expense, 0.42; water, 0.21; light, 0.16; total, 100 per cent.

Details of Reverberatory-Matting Furnace.[3]—In American practice the grate-bars are 2x2 in. set with openings between of 2 in., and capable of being moved aside when grating the fire. The accompanying table shows the trend of the practice.

---

[1] Recently the plan has been adopted of dividing the fire-box into two or three compartments by walls parallel to the axis of the furnace, each closed by its own door. Thus one compartment can be opened without disturbing the others, which keep up their active burning of the fuel. L. S. A.

[2] *Eng. and Min. Journ.*, LXXXIII, 198.

[3] Peters, "Principles of Copper Smelting."

DATA OF REVERBERATORY FURNACES.

Furnace	Length of Grate Feet	Breadth of Grate Feet	Area of Grate Square Feet	Coal daily Short tons	Coal hourly pounds	Ratio of Chimney to Grate	Ore Smelted daily Short tons	Percentage of Coal used
Argo, Colo. 1887........	5.5	4. 5	24.75	9.0	30.3	1 ·2.75	24.0	37.5
Argo, Colo. 1891	6.0	4.75	28.50	10.0	29.2	1 :3.17	28.0	35.8
Argo, Colo. 1894........	6.5	5. 0	32.50	13.5	34.6	1 :2.03	50.0	27.0
Montana, 1903..........	10.0	5. 5	55.00	36.0	54.5	1 :1.83	112.0	32.3
Washoe plant, 1906.....	16.0	7. 0	112.00	57.0	42.4		275.0	20.7

The furnaces at Argo smelt an ore-mixture containing on an average as follows: SiO_2, 33.9 per cent.; Fe, 10.8; $BaSO_4$, 15.5; Al_2O_3, 5.6; CaO, 4.8; MgO, 2.8; Zn, 4.9; Cu, 3; S, 5.1, producing a matte of 40 per cent. Cu, and containing large values in gold and silver. The coal used with this charge contains H_2O, 1.4 per cent.; fixed carbon, 54.9; volatile constituents, 32.9; ash, 10.8.

The Montana furnaces run mainly on a charge of 8 per cent. copper, consisting of roasted pyrites, concentrate and silicious ore. They produce a fairly clean slag of 42 per cent. SiO_2, and a matte of 45 to 48 per cent. copper.[1] They use a free-burning, non-caking coal containing ash 10, fixed carbon 50, and volatile matter 33 per cent.

The accompanying table shows that the coal burned per square foot of grate area increased with the ratio of grate area to stack area, i.e., the stacks of the older furnaces were made too small.

Oil-Burning Reverberatory Furnace.—Early in 1906 two large reverberatory furnaces were erected at the plant of the Consolidated Arizona Smelting Company, Humboldt, Arizona, which use California petroleum of 14 to 17 deg. B. The dimensions of the furnaces are: Length of hearth, 98 ft.; width of hearth, 19 ft. 1 in.; spacing of side-doors (seven on each side), 10 ft.; hight of roof above hearth near charging-end, 7 ft.; at front, 3 ft., 1¼ in.; thickness of arch, 12 in. The buck-staves are 8-in. I-beams with tie-rods 1¾ in. diameter. The skew-backs are formed with 12-in. 20-℔. channels. The total weight of iron-work for one furnace is 305,000 ℔.[2]

The foundation consists of a solid block of concrete 18 in. thick, 30 ft. wide and 114 ft. long. On the concrete is placed 18 in. of red brick, then a layer (2½ in.) of fire brick, and finally for the bottom 24 in. of quartz of 97 per cent. SiO_2 crushed to 12-mesh. The side-walls have a lining of 19 in. of silica brick, backed by 4½ in. of fire brick and by 9 in. of red brick, a total thickness of 32½ in. of wall. The roof is of silica brick, 12 in. thick. Each furnace has nine oil-burners, three at the back end of the furnace, three on either side. They enter the furnace at the doors, are blown by steam, and are hung by universal connections so that they may be

[1] *The Mineral Industry*, XII, 104; and, XV, 275.
[2] Peters, "Principles of Copper Smelting." p. 201; *The Mineral Industry*, XV, 278.

pointed at any angle. The three end-burners have, however, been found sufficient. The oil is fed at 80 lb. pressure by a plunger circulating-pump. The steam is obtained from the waste-heat boilers of the furnace, using 7 per cent. of what they produce. The ore is charged at the near end of the furnace (there is neither fire-bridge nor fire-box), and is carried in 20-ton hoppers from which it is dropped through four charge-openings, all within 26 ft. of the end. In multiple with the furnaces are two batteries, each of two Stirling water-tube boilers, equal in the aggregate to 1300 h.p. Connections are arranged so as to cut out either boiler, the furnace-gases being sent to the other boilers, or one reverberatory may run on any two boilers without interfering with the other two. The boilers are also equipped with oil-burners for independent firing. These furnaces are operated on custom-ores, and hence the charge constantly varies. It has been found that with oil-fuel more silicious slags can be run, and ore in pieces up to 2½ in. diameter can be smelted without retarding the furnace. This would be practically impossible with coal. Charges between the following limits have been smelted: FeO, 33 to 18 per cent.; SiO_2, 28 to 34 per cent.; S, 9 to 16 per cent.; Cu, 6 to 5 per cent.; CaO, 6 to 12 per cent.

From 11 to 19 per cent. of oil is used, and matte of 40 per cent. Cu. has been produced from a charge containing 5 per cent. Cu. and 10 to 12 per cent. S. The furnace atmosphere is evidently more oxidizing than where coal is used for fuel, and it is stated that much sulphur is burned off at the time the ore-charge is dropped.[1] The tonnage in 24 hr. is 300 to 350 tons, a charge of from 10 to 15 tons being put into the furnace every 45 minutes or so. The slag is drawn off every 2½ hours, granulated with water, and run to waste.

A small outside settler is used. The matte is tapped and transferred in ladles to the converters. The crude California oil is delivered at the smelter in tank-cars at a cost of $1.25 per bbl. of 42 gal., and 29 to 52 gal. is needed per ton of ore. Oil is very efficient in quickly bringing up the heat.

Reverberatory Smelting Plants.

Plant of the Steptoe-Valley Smelting and Mining Company.[2]—It is situated 14 miles from Ely, Nevada, and is distinctly a reverberatory plant. There are sixteen 18-ft. McDougall roasting-furnaces, which are of 60 tons daily capacity per furnace, roasting the ore down to 7.5 per cent. S. Crushed limestone is mixed with the ore before roasting, and the hot calcines are trammed directly to the hoppers of the reverberatory smelting-furnace. Moreover, a brick-and-steel storage-bin of 1500 tons capacity is interposed to regulate the supply to the latter furnaces according to

[1] Most likely by the reaction, $FeS+3\,Fe_2O_3+7\,SiO_2=7\,FeSiO_3+SO_2$. L. S. A.
[2] *Eng. and Min. Journal*, LXXXIV, 813.

their needs. The roaster-flue is double, each part of 300 sq.ft. section and 500 ft. long, and is terminated by a stack 18 ft. in diameter by 250 ft. high. There are three reverberatory smelting-furnaces, 19x112 ft., each provided with two 400-h.p. boilers in parallel and with suitable by-passes. The fire-boxes are 8x19 ft. and are 3 ft. below the top of the bridge. The ashes are sluiced to coal-washing jigs to separate the cinders from the unburned coal. For charging ore there are two hoppers, each with three discharge openings, and one coal-hopper of four openings. The slag is granulated, while the matte is tapped into ladles on cars to be conveyed to the converting department. The reverberatory-flue has a cross-section of 300 sq.ft., is 200 ft. long and doubles back through a settling-chamber of 100 sq.ft. section and 200 ft. length, to a stack of 15 ft. diameter by 300 ft. hight. There are three converter-stands 96x150 in., electrically-operated, and nine shells. The matte is poured into the converters by a launder while they are in blowing position, and silicious ore is added in the same way. The converter-flue is of 100 sq.ft. section, 200 ft. long, and terminates in a stack 10 ft. in diameter by 100 ft. high. The blister-copper is molded in an endless-chain casting-machine, from which the ingots are handled by an air-hoist. The converter-slag is cast in a slag-casting machine and returned to the blast-furnace.

There is one blast-furnace, 42 in.x20 ft. long, which has been designed so that it can be lengthened to 80 ft. if desired. It is set 13 ft. above the ground level to give room for the forehearth and ladles below the slag-spout. The distance from tuyeres to charge-floor is 13 ft. and the furnace is run with a smelting-column of 10 ft. and a blast-pressure of 24 to 30 oz. per sq.in. The charge-cars are tipped into the furnace by a compressed-air cylinder as at the Washoe smelter.[1] Each jacket has its own blast-box (supplying four tuyeres), and independent air and water connections, so that it may be quickly removed without affecting other jackets. The blast-furnace bins are made of wood and are of 6000 tons capacity.

The power-plant is equipped with eight 400-h.p. Babcock & Wilcox boilers. In it, for driving the blast-furnace, is provided one No. 10 Connersville blower (300 cu.ft. to the revolution and 100 r.p.m.), direct-connected with a compound condensing engine. There are for the converters two blowing-engines, one of 12,000 cu.ft. and the other of 6000 cu.ft. per min. capacity, both cross-compounded.

To provide power for the works, concentrator and mine, there are also two alternating-current 750-k.w. and two 1000-k.w. generating sets.

The machine shops, carpenter shop, store houses, etc., are large enough to do the work of both the mine and the smelters, and for the rolling stock of the Nevada Northern R. R. The machine shop is of steel-and-concrete construction. The administration buildings are on a liberal scale and a

[1] *The Mineral Industry*, XV, 256.

large laboratory and assay office is provided. The ore and other materials are handled from point to point in cars operated by steam locomotives running on standard gauge tracks. These tracks command the roasters, the reverberatories and the converters. The plant as now arranged has easily a capacity of 1000 tons daily and has been designed so that this may be doubled.

The Wallaroo Smelting Works, South Australia.[1]—The ore comes to the works either hand-picked and broken to 2.5 in., or as concentrate. The hand-picked ores, purchased in various parts of Australia, are oxidized (malachite, cuprite and atacamite), while the concentrates are from sulphides, the two being in the proportion of 13 to 20. The mines of the company are the Wallaroo and the Moonta. The Wallaroo concentrate contains: Cu, 13 per cent.; Fe, 26.1; S, 22.2; Pb, 0.25; Ni, 0.169; Zn, 0.73; As, 0.011; Bi, 0.0012; Te, 0.001; SiO_2, 25.2; Al_2O_3, 2.8; CaO, 3.3; MgO, 3.2; undetermined, 3.1; with 0.75 oz. Ag and 0.0515 oz. Au per ton. The concentrate from the Moonta mine carries: Cu, 19 per cent.; Fe, 25.3; S, 17.8; Pb, 0.05; Ni, 0.08; Zn, 0.04; As, 0.004; Bi, 0.001; Te, 0.001; SiO_2, 27; Al_2O_3, 2.7; CaO, 0.8; MgO, 1; undetermined, 6.2; with 0.65 oz. Ag and 0.0515 Au per ton. It is held that bismuth and tellurium are both embrittling in their action, though their effect on conductivity is less than where arsenic is present in quantities as small. The copper has a conductivity of 98 per cent. Mathiessen standard. In roasting, Hosking furnaces were used, producing 1.1 per cent. flue-dust. Putting through 160 tons daily of concentrated ore, reverberatory smelting was better than blast-furnace treatment. The size of the reverberatory smelting furnace, which at the time of installation was considered fairly large, is 23 ft. 8 in. x 14 ft. 9 in. To keep up the purity of the copper produced in these works, a fire assay is made upon purchased ores so as to produce a button of "coarse" copper, whose fracture shows whether impurities are present to the extent of including them in the impure class. If the button does not show the presence of these impurities it is subjected to careful chemical analysis to make sure of its good quality. The more nearly oxidized ores are reserved for direct smelting, and silicious oxidized ores are added in the matte treatment. The lump ore, containing 10 to 12 per cent. Cu., 19 per cent. S and 30 per cent. SiO_2, is roasted down to 5 per cent. S in muffle-furnaces, or pyrite-burners, producing a gas containing 7.5 per cent. by volume of SO_2 gas, which is utilized for making sulphuric acid. There are two sets of these burners, each having twelve muffles set back to back, with a central flue for carrying off the gases. Each muffle takes two charges of 1120 lb. in 24 hours, or an aggregate of 165 tons weekly. A man and boy working eight-hour shifts take care of these burners, there being two shifts in 24 hours, so that the furnaces

[1] Trans. I. M. M., XVI, 55; *Eng. and Min. Journ.*, LXXXIII, 324.

are left to themselves for four hours. The roasted ore is crushed through rolls to 1 in. size and trammed to the mixing yard.

The concentrate, containing on an average 15 per cent. Cu and 20 per cent. S, is calcined down to 7 per cent. S in revolving cylinders of the Oxland & Hocking type, each 40 ft. long by 6 ft. in diameter and having a 9-in. fire-brick lining. These furnaces last nine months before repairing, while the lining lasts seven years. Besides the charging bin for the calcining cylinders, there is a mixing bin, in which is put a mixture of raw ore, coarse roasted ore and reverberatory slag of 6.5 per cent. copper from the "roaster" or blister-copper furnaces. This mixture is fed upon a conveying belt to the elevator boot which takes the calcines from the cylinders. The elevator discharges to hopper cars, and these deliver their hot contents into the matting reverberatory furnace when it needs another charge. The coal used in calcining (including that needed for power) averages 7 per cent. of the ore put through. For driving each furnace 2.5 h.p. is required. A calciner, attended by one man per 12-hour shift, puts through 120 tons daily, supplying five reverberatories. These latter have hearths 23 ft. 2 in. x 14 ft. 8 in., with 4 ft. 8 in. x 6 ft. grates. The ash-pit is closed by iron doors and preheated air is drawn in through iron pipes set in the stack, thence through flues beneath the furnace to the ash-pit and to roof ports above the fire bridge. These furnaces smelt three charges of 8 tons daily, consuming 38 per cent. of coal. They are worked by two furnacemen and one helper per shift of 12 hours. The slag produced is of the following composition: SiO_2, 40 per cent.; FeO, 42.5; Al_2O_3, 6.5; CaO, 5; MgO, 3; Cu, 0.3; Pb, 0.11; Ni and Co, 0.18; Zn, 0.2; P_2O_5, 0.9; undetermined, 1.3. The matte contains 51.5 per cent. Cu, 21 per cent. Fe and 26 per cent. S. The furnace charge is tapped into sand molds and, when cold, the matte is sorted out.

The following is the composition of the furnace products: Matte charged, Cu, 51.2 per cent.; Ni, 0.25; Zn, 0.39; As, 0.006; Au, 0.25 oz. and Ag, 1.7 oz. per ton: selected matte, Cu, 80.2 per cent.; Ni, 0.20; Zn, 0.03; As, 0.003; Au, 0.012 oz. and Ag, 0.9 oz. per ton: bottoms, Cu, 98.1 per cent.; Ni, 0.5; Zn, 0.06; As, 0.010; Au, 3.80 oz. and Ag, 3.0 oz. per ton.

In refining, 12 tons of rough copper, together with some cathode copper from the electrolytic refinery, constitutes a charge, using 40 per cent. of coal. The copper is cast into ingots of 14 lb., and cakes of 40 and 100 lb. respectively. Each furnace charge is tested for conductivity and must come up to 88 per cent. If below this a sample is analyzed to discover the trouble, whether properly poled or whether there is too much nickel. If the latter is the case it is sent back to the "coarse" copper furnace to be put through, a little at a time; if the fault is due to the insufficient oxygen it is sent back for remelting and repoling. For determining the electrical conductivity a test piece is forged down to ¼ in. diameter and drawn to 0.025 in. diameter.

It is annealed and cleaned before testing and compared with a standard coil.[1]

To bring the first matte to the grade of coarse copper, the further operations are as follows: The matte is crushed by rock breakers and rolls to 4-mesh and delivered to a storage bin for feeding to a calcining cylinder already described. Each calciner roasts 16 tons daily to 12 per cent., using 6.5 per cent. coal, and takes one man per shift of 12 hours, with the assistance of a boy working one 8-hour shift. The crushing is attended to by three boys, each working an 8-hour shift daily. For crushing and conveying the matte there is needed 3.75 h.p.

The melting or "roasting" furnaces are charged first with one-fourth raw matte with a small quantity of refinery slag, flue-dust and low-grade copper precipitate, and three-fourths calcined matte dropped from the charge hopper. The raw matte is charged first since it melts more easily than the rest of the charge. The products of the process are "roaster" slag of 6.5 per cent. Cu, and coarse or rough copper of 98 per cent. Cu. These furnaces work off five charges of 8 tons each weekly, using 70 per cent. of coal. One or two furnaces are reserved for the treatment of impure ores, by treating the matte worked in them so as to produce bottoms into which go the impurities. The aim is to regulate the quantity of bottoms to contain 5 per cent. of the copper of the charge. They are made into anodes, while the purified or selected matte is sent to one of the roaster furnaces above described.

The results obtained during a week's run, where the proportion of bottoms was 7.5 per cent. of the copper in the charge, are shown in the accompanying table.

COMPOSITION OF REFINED COPPER.

	Wallaroo.	Quincy.	Mansfeld.		Wallaroo.	Quincy.	Mansfeld.
Copper	99.6648	99.7975	99.6332	Arsenic	0.0022	0.0208	0.0118
Iron	0.0053	0.0031	0.0049	Tellurium	0.0030	*Nil.*	*Nil.*
Sulphur	0.0005	0.0035	0.0039	Bismuth	0.0018	0.0001	0.0003
Oxygen	0.0997	0.0862	0.1327	Insol. residue	0.0038	0.0029	0.0037
Lead	0.0215	0.0038	0.0312	Silver	0.0063	0.0692	0.0228
Nickel	0.1764	0.0106	0.1480	Gold	0.0013	0.00002	*Nil.*
Zinc	0.0134	0.0023	0.0075	Total	100.0000	100.0000	100.0000

The electrical conductivity of the sample of Wallaroo copper was 88.63, Mathiessen's standard.

Most of the reverberatories, as well as the roasters and the refinery, are connected by an underground flue 110 ft. long by 60 sq.ft. cross-section to a stack 100 ft. high and 7 ft. 6 in. interior diameter. From these flues were recovered 620.5 tons of flue dust (36.96 per cent. Cu.), equal to 1.1 per

[1] In this respect Lake Superior practice is better. Here a "spike" ⅜ in. in diameter by 8 in. is cast; this is reheated in a muffle and rolled hot in small power rolls. It is then pointed and drawn to 0.104 in. The wire is straightened, weighed on a balance and clamped in the testing machine. The computation takes all variations into consideration and the results are taken on the hard-drawn wire. L. S. A.

cent. of the ore tonnage, the dusting being chiefly that arising from the roasting in the cylinders. The total extraction or recovery of the works is 95 per cent. of the copper in the ores. The total coal used is 0.87 per cent. of the ore smelted, containing moisture, 2.5 per cent.; volatile constituent, 34.5; fixed carbon, 52.0; ash, 11; and costing $3.25 per ton unloaded.

The coal is unloaded from vessels into square buckets each of 2.67 tons capacity. These are placed on flat cars and set on a truck at the coal yard, having a grade so that the cars can be dropped past near one end of the electric crane of 80 ft. span, commanding the yard, and capable of unloading 80 tons an hour. This crane picks up the buckets and empties them into storage. The crane can be moved along as the heap grows, thus building it up. A sand well suited for furnace-bottoms, and containing 93.3 per cent. SiO_2, 1.6 Fe_2O_3, 2.2 CaO, 1.1 organic matter and moisture, is used.

Blast-Furnace Smelting Plants.

Plant of the Consolidated Arizona Smelting Company, Humboldt, Arizona.[1]—The ore is chalcopyrite and pyrite with a gangue of quartz and fine-grained schist. After passing through the sampling mill it is screened, the undersize going by conveyor to the bins, from which it can be drawn off for the roasters, while the coarse ore is delivered by another conveying belt to the blast-furnace bins. From both these sets of bins the ore is taken by tram cars to be charged to the desired furnaces. The concentrates, produced from concentrating ore of 3 to 4 per cent. Cu, as well as the fines or undersize ore above mentioned, are received from the bins into electrically-operated hopper bottom scale cars of 17,000 ℔. capacity, and transferred to electrically-driven oil-fired Edwards roasters, of which there are four. The roasters are of the duplex type, having a double row of rabble arms and shafts.

There are two reverberatories, each of 105x20-ft. hearth dimensions, and each furnished with two 400-h.p. Stirling boilers to utilize the waste heat from the furnaces; steam thus supplied gives the power required to run the plant and for transmission to the mines. The reverberatory slag is granulated with water which comes from the blast-furnace jackets and from the condenser. The matte is tapped into ladles, picked up by a travelling crane, and poured into either of the converters. The crane also transfers the blast-furnace matte to the converters, and delivers molten slag from them to the reverberatories. These are of 84x126-in. size, and there are six shells. The material for lining is a silicious ore containing 3 per cent. Cu and 65 per cent. SiO_2, with a little silver and gold; to this is added a sufficient quantity of clay for binder. The mixing

[1] *Eng. and Min. Journ.*, LXXXIII, 901. *The Mineral Industry*, XV, 278.

is done by two electrically-driven Chilean mills, and the material is tamped in place by an Ingersoll-Sergeant tamping machine. The compressed air for this is supplied by an Ingersoll-Sergeant compressor in the power house.

The blast-furnace puts through the coarse ore above referred to (which carries 50 to 60 per cent. SiO_2 and 3 to 5 per cent. Cu), together with a limestone containing 2 per cent. Cu, 16 to 18 MgO, and 35 CaO, and an iron ore containing 6 to 8 per cent. Zn, 14 SiO_2, and 32 Fe. Both of the elements zinc and magnesia make an unsatisfactory slag, which gives great trouble in the forehearth. The matte contains 30 per cent. Cu. Side-dump cars, tilted by compressed-air cylinders, are used at the blast-furnace, and the charges are weighed on Standard track scales.

The pumping station, located at the Agua Fria river close by, contains two electrically-operated four-stage centrifugal pumps and one steam pump. These deliver the water by 8 and 10-in. mains to the works. A dam, erected across the stream at the pump house to impound the water, was recently swept away. A slag-lined pond has since been put in, connecting with the river, and from this water is raised for the works. Even Mexican labor is $3 per day.

British Columbia Copper Company Works, Greenwood, B. C.[1]—Custom ore is weighed on self-registering scales and sent to bins of 200 tons capacity. The sampling mill is of 600 tons capacity daily. The ore from it is delivered by a belt conveyer to dump-cars, which in turn discharge into the smelter bins of 12,000 tons capacity. There are also coke bins of 2000 tons capacity.

The three blast-furnaces are 46x240 in., each with a capacity of 600 to 700 tons daily. They are charged by side-dump cars hauled in trains by electric locomotives. The slag-cars are of 25 tons capacity, built by the M. H. Treadwell Co., Lebanon, Pa., and are handled by an electric locomotive. The blast-furnaces are supplied by three No. 10 Root positive-blast blowers, each of 300 cu.ft. displacement and driven by 300-h.p. motors. For the converting plant there is a Nordberg blowing engine of 5000 cu. ft. per min. capacity, driven by a 300-h.p. variable-speed motor. There is also an air compressor for driving the pneumatic tools, and air hoists for the charge cars and for raising the furnace doors.

Two motor generators of 100 and 75 k.w. capacity furnish direct current for the travelling crane in the converter house and the electric locomotive. The current for this installation comes from the power station of the Columbia Constructing and Distributing Company, at Bonnington Falls on the Columbia river, 75 miles distant. The two-stand converter plant has a 40-ton electric crane, and the converters are operated by means of a pump and a hydraulic accumulator. The water supply comes in from

[1] *Brit. Col. Min. Rec.*, Dec., 1906, p. 481.

Copper creek under ample head. The jacket water goes to a cooling pond and is pumped back by a centrifugal pump to the supply tanks which have an aggregate capacity of 160,000 gallons. The blast and converter buildings are of steel, and the plant has fully equipped machine and blacksmith shops and store houses for supplies. The works have a capacity of 50,000 to 60,000 tons monthly.

Smelting Works of the Le Roy Mining Company, Northport, Wash.—These works have six furnaces of a united capacity of 1200 tons of ore daily. This makes 200 tons of ore per furnace, or 325 tons of charge daily, the Le Roy ore constituting 55 per cent. and the lime 30 per cent. of the charge. The fuel carried is 12 per cent., and the grade of the matte 10 to 12 per cent. This is put through the furnace and afterward roasted and smelted with high-grade silicious ore. No iron flux is needed, the Le Roy ore carrying sufficient iron excess to permit the use of 500 tons monthly of silicious ore of 86 per cent. silica.[1]

Works of the Consolidated Mining and Smelting Company, Trail, B. C.—Receiving and storage bins and a sampling mill have been put in, and a gravity system with electric haulage to the blast-furnace bins installed. Another copper matting furnace has been added, thus making five in all. The older blast-furnaces have been lengthened to 20 ft., the former length having been 15 ft. The charging is done by cars operated by an electric locomotive. The additional Root blowers needed are electrically driven. This portion of the plant, devoted to the smelting of copper ores, has a a capacity of 1500 to 1800 tons daily (300 to 360 tons per furnace).[2]

Furnaces of the Mount Lyell Mining and Railway Company.—Fig. 1 is a half longitudinal elevation, half-section of these furnaces, and Fig. 2 gives the corresponding transverse views. The furnace is 210x42 in. at the tuyere level and has 48 tuyere openings, each tuyere-pipe taking two openings. Contrary to ordinary practice there are also end tuyeres. There are two tiers of jackets, and the smoke is carried off by a down-take beneath the charge floor. The effective smelting column is therefore 9 or 10 ft., while the distance from the tuyeres to charge floor is 14 ft., and between the charge and slag floors, 20 ft. The down-take has a hopper bottom with a slide by which the dust is drawn off to a car set below. The furnace has a trapped spout for continuous flow. The forehearth, 5x8 ft., is lined with 12 in. of brick and has 18 in. effective depth. There are three balanced sliding doors on each of the long sides, by which access to the furnace can be had for its entire length. The furnaces are charged by two-wheeled buggies or charge carts holding 1120 lb. each. Four buggies or 4480 lb. make a charge.[3]

Copper Smelting Section of the Plant of the United States Smelting, Refining

---

[1] *Brit. Col. Min. Rec.*, Dec., 1906, p. 481.
[2] *Brit. Col. Min. Rec.*, Dec., 1906, p. 480.
[3] *Metallurgie*, III., p. 766. *The Mineral Industry*, XII., 117.

and Mining Company.—The copper smelting division of this plant consists of six blast-furnaces and five converting stands, having a capacity of 1500 tons daily. The actual ore capacity may be reckoned at 80 per cent. of the nominal. The monthly production in the summer of 1907 was 4,000,000 lb. of copper, while the converter plant has a capacity of

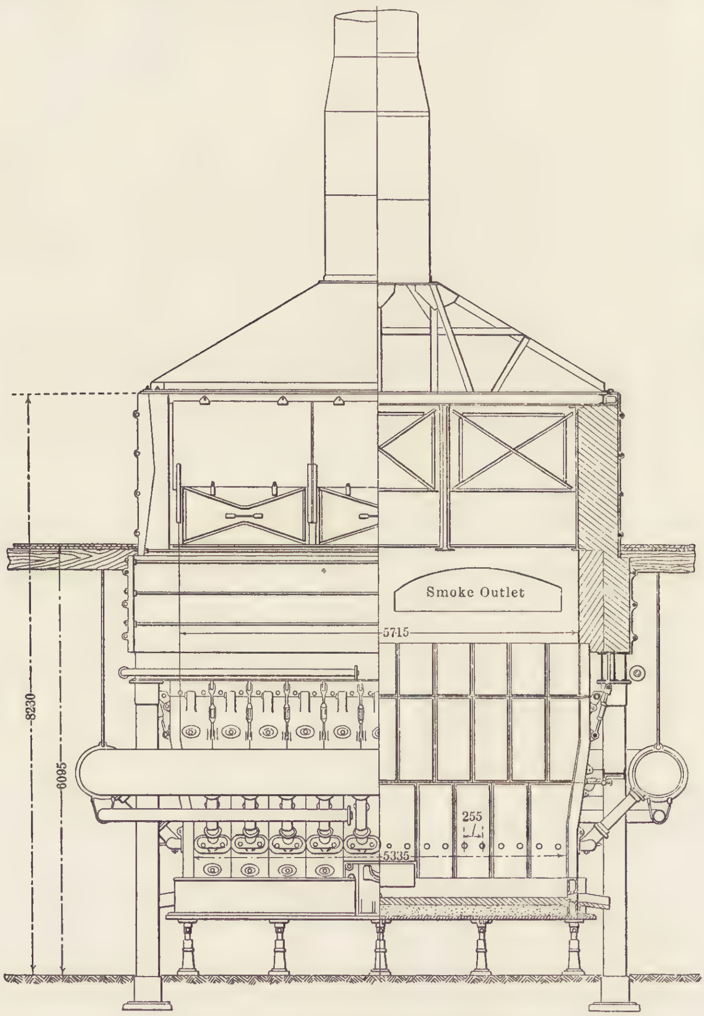


FIG. 1.—MOUNT LYELL BLAST FURNACE.

5,000,000 lb. monthly. The six furnaces are placed in line with a forehearth in front of each. In front of the forehearths is the runway or bay for the cranes. This runway is but 28 ft. wide, in place of 55 to 60 ft. allowed in modern practice. In consequence the space is cramped so that

operations are impeded. Back of the two furnaces the downtakes go to a common concrete flue which leads to the chimney. The charging is done from cars which dump sideways into the furnace. The charge consists of sulphide ores, silicious ores and limestone, and the matte runs

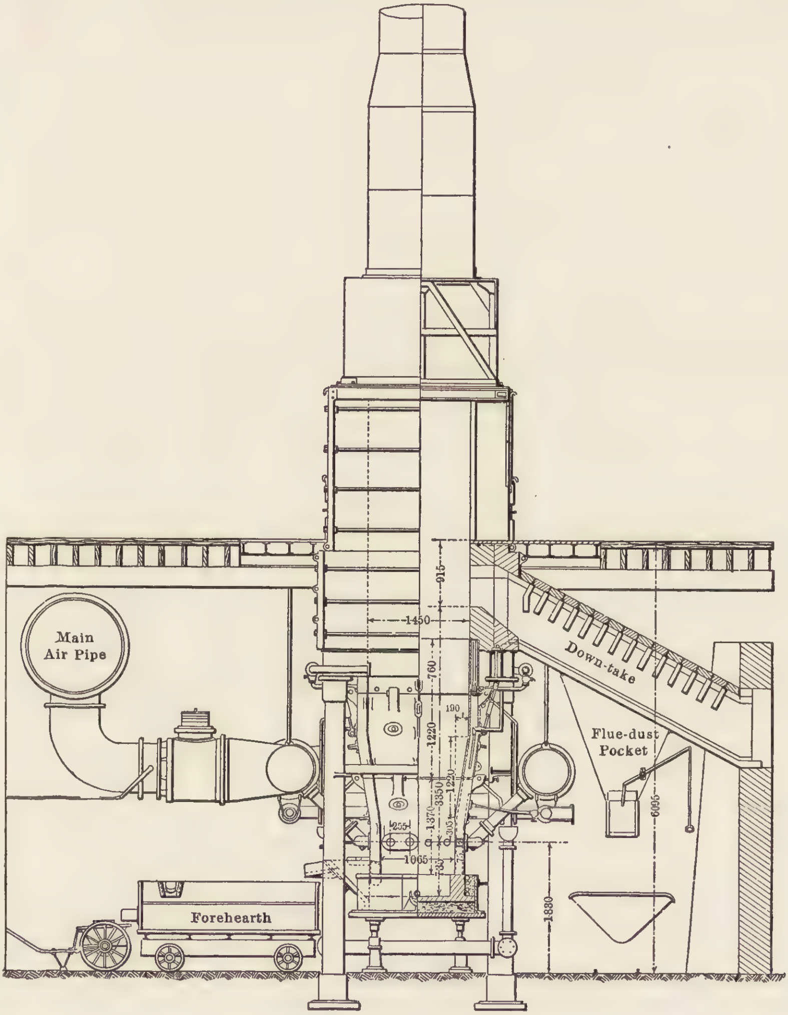


FIG. 2.—MOUNT LYELL BLAST FURNACE

45 to 48 per cent. Cu. The converters are of the barrel type, 90x120 in. in size. The plant appears to be somewhat cramped and shows the effects of six years of continuous operation.[1]

[1] *Eng. and Min. Journ.*, LXXXIV, 530.

Granby Consolidated M. S. and P. Company Works, Grand Forks, B. C.—Additions to equipment in 1906 included two large blowers, four 150-h.p. motors to run the blowers, a third converter stand, an automatic slag conveyer, and a 750,000-gal. plunger pump. The wood frames of the furnace building have been replaced by steel and new buildings have been added. There are eight furnaces, one in reserve, the capacity of the seven being 90,000 to 95,000 tons monthly.[1]

Garfield Plant of the Garfield Smelting Company (Subsidiary of the American Smelting and Refining Company), Garfield, Utah.[2]—The plant is situated near the shore of Great Salt Lake, 24 miles to the westward of Salt Lake City. It is free from liability from smoke damage. To attain this the company bought up 35,000 acres liable to be within the range of damage. The smoke either blows over the lake, or up the cañons and into the mountains, at the foot of which the plant is situated. It has good railroad facilities, having connections to the Rio Grande Western R. R., and the Los Angeles, San Pedro & Salt Lake R. R.

The enlarged plant will contain six large reverberatories and four blast-furnaces. These furnaces and the converters are under one roof, a design which has evident drawbacks, due to lack of sufficient ventilation, and of space in operating. The reverberatories are 19.5x110 ft. (too long for the present draft), and are each provided with two 350-h.p. Stirling boilers in tandem. When in fair operation they smelt 250 tons each daily, using 50 tons, or 20 per cent. of coal. There has been trouble in their operation, so that irregularities had to be corrected by the use of undergrate blast from a fan blower, but it is still impossible to run the waste-heat (Stirling) boilers on more than two out of three furnaces; otherwise the escaping gases are not sufficiently hot. The matte is of about 40 per cent. Cu; the slag 40 per cent. SiO_2. No attempt is made to save the unburned fuel of the ash.

There are two furnaces, 48x240 in. at the tuyeres, set with their long axes in line. It is proposed in the future to include the intervening space, making a single furnace 48 in. x 70 ft. The other furnaces are to be on the same center line. They carry 24 oz. blast pressure and have a 10-ft. smelting column. The ore charge contains 4 per cent. Cu on an average. One of these furnaces is now operated semi-pyritically, smelting 350 to 370 tons daily with a consumption of 8 per cent. of coke. The resultant matte of 20 to 25 per cent. Cu is added to the charge of the other furnace. This second furnace smelts 500 tons daily, using 8 per cent. of coke, and yielding 40-per cent. copper matte. The charge-hoppers, at each side of the furnace at the charge-floor level, are really troughs, of the length of the furnace, hinged at the back lower corner, and filled from the charge

[1] *Brit. Col. Min. Rec.*, Dec., 1906, p. 481.
[2] *Eng. and Min. Journ.*, LXXXIV., 576. *The Mineral Industry*, XV., 280.

car. The front edge of the trough being lowered, the charge slides into the furnace. These troughs or pans do not, as now arranged, work well. The drop from them to the surface of the charge in the furnace also seems excessive. The slag from both reverberatories and from the blast-furnace is tapped to slag cars handled by an electric locomotive. The slag carries 0.35 per cent. copper.

There are at present four stands of converters, each 38x96 in., and 12 shells. Six more stands are to be added. There are 16 McDougall roasting furnaces, each roasting 40 tons daily from 30 per cent. S down to 8 per cent. Eight more are to be added. The ore for the roasters is distributed to long bins, where it is evenly distributed by trippers automatically moving from end to end of the bins. From the beds it is drawn into cars, which are electrically trammed to the furnace hoppers. On the lowest hearths of the roasters, fine Utah Copper Company concentrate is thrown in by hand. This portion of the ore is little more than dried, but it is heated up and in better condition to enter the reverberatory and, being charged on the lower hearth of the roaster, makes but little dust.

There are 25 roasting pots for the pot-roasting of copper concentrates, each desulphurizing and sintering three charges of 7 tons each in 24 hours, making a final product of but 3 per cent. S from a partly roasted product of 12 per cent. which comes from the McDougall roasters. This is watered down in one of the pots, picked up by a travelling crane and its contents dumped upon the hot priming charge of another pot just to be started. These pots stand in a single row and are commanded by a travelling crane. Back of these is a flue to which the hoods of the pots connect by a short horizontal branch-pipe. When finished the pot is picked up by the crane, carried to the breaking floor at the end of the line and there dumped out. A crew of four men, with the aid of a 24x36-in. breaker, are able to handle and coarsely break all this material. It is removed by conveying belt and elevator to the blast-furnace bins. This breaking has the disadvantage for blast-furnace use that it makes some fines.

With the exception of four steel boiler-house stacks belonging to the four 350-h.p. Stirling boilers, the works has but one common stack 300 ft. high by 30 ft. diameter. The base of the stack is set 200 ft. above the furnaces, so that its top is 500 ft. above them. The blast-furnace, reverberatory, converter, McDougall furnace, and H. & H., or pot-roasting departments, each have their separate flues. These are combined into three main flues extending up the hill to the main stack. The length of these main flues is 3500 ft. The first portion of the blast-furnace flue is balloon shaped in cross-section, later it takes a beehive form (an inverted catenary) in section, its lower portion consisting of a series of steel hoppers for the convenient drawing off of the flue dust. Slides at the bottoms of

the balloon flue serve the same purpose. The flues from the reverberatory and McDougall furnaces are of the catenary form. Thus the dust and fines have an excellent opportunity of settling, so much so that careful tests of the smoke at the main stack show no loss of metal.

Ore, fluxes and fuel are brought in by cars upon an extensive and expensive system of high trestles, and delivered to receiving bins. From them the ore is transported by means of belt conveyers to the crushing mills. There are two of these, a sulphide mill, where the sulphide ore is fine-crushed and sampled, and a "silica" mill, where the non-roasting ore is coarsely crushed and sampled. The sampled ore now goes by another system of belt conveyers to the storage beds, ready to be drawn off by cars as already specified. In attempting to make the handling of all material about the works as automatic as possible it would appear that it has been overdone, and the system has already given a good deal of trouble in operation.

The works has an excellent foundry, a large carpenter shop and a well-equipped machine shop. The power house contains at present two cross-compound blowing engines for the converters, and two No. 10 rotary blowers (300 cu. ft. per revolution) for the blast-furnaces, each blower delivering air independently to its own furnace by a long blast main.

The plant has had troubles in starting up, such as may be expected in the operation of a new plant. It was designed upon lines well known in principle, but more or less open to criticism as affecting the efficiency of the plant; however, the defects are not of such a character as to prevent the concern running night and day successfully. The first cost of the plant is said to be very high, and to aggregate five million dollars when completed with its full quota of furnaces. Estimating the annual output at 900,000 tons this will make $5.55 per ton of output, or at 6 per cent. interest, 33c. per ton. Heretofore $3 per ton, equivalent to 18c. interest at 6 per cent., was considered a reasonable figure for a well equipped plant.

The Works of the Dominion Copper Company, Boundary Falls, B. C.—Besides two older furnaces, each of 350 tons capacity, a new furnace of the Traylor Engineering Company, New York, having a capacity of 700 tons daily, has been installed and furnished with the Giroux hot-blast top. The 50 per cent. copper matte produced is treated by the British Columbia Copper Company, at Greenwood, B. C.[1]

Plant of the Compañia Metalurgica y Refinadora del Pacifico.[2]—This is the 600-ton operating plant of the Douglass Copper Company, and is situated at Fundicion, 100 miles from Guaymas, Sonora, Mex. The principal ores, of which there are enough for years to come, are those of copper

[1] *Brit. Col. Min. Rec.*, Dec., 1906, p. 482.
[2] *Eng. and Min. Journ.*, LXXXIV, 156; *Min. and Sci. Press*, XCV, 91.

produced in the Alamos district, and are to be smelted with gold and silver-bearing ore.

Power-house.—The 150-cu. ft. Root blowers are direct connected to Bates tandem compound condensing engines. There are two generators direct-connected to Bates vertical cross-compound condensing engines. The current will be direct and in two voltages, one of 100 volts for lighting and other purposes, and one of 500 volts for power transmission to machines about the plant.

Sampling.—The ores are to be mechanically sampled, one-tenth being first cut out by means of a Vezin sampler. This is sent to the sampling works, crushed, and from this one hundredth of the original lot is reserved for a sample. The samples and assays are thus checks against each other.

Blast-furnaces.—There are two, each 44x160 in., with a 14-ft. smelting column, and having a forehearth or settler 15 ft. in diameter by 5 ft. deep. They have 6-ft. down-takes leading to a flue of balloon-shaped cross-section 11 ft. wide, and to a self-supporting steel stack 177 ft. high and 8 ft. in diameter at the top.

The materials of the charge are raised by Jeffrey steel-pan elevators to the steel charging bins, situated one at each end of the furnace, whence they can be run into the furnace as desired. The feed to the elevator boots is by means of divisional charge cars which provide for the delivery of the material in proper proportions. It is well mixed in this process of handling. The plant is served by a 36-in. gage railway to handle ore, matte and slag. The matte ladles will be of 10 tons capacity. There is to be a converter building 200 ft. from the blast-furnace building.

Blast-furnace Smelting at Mansfeld.—The roasted cupriferous shale (*Kupferschiefer*) averaging 3 per cent. Cu, is treated in a 5-tuyere circular Pilz blast-furnace for the production of a 40-per cent. matte. Air blast is furnished by Root blowers at 5 ℔. pressure. The ore is charged by bell-and-hopper, three carloads being charged to the wall by lowering the bell in the cone, and the fuel together with some slag, by raising the bell, is dropped near the center. The furnace is run with a hot top, burning off zinc and excess of sulphur. Some copper collects at the bottom of the forehearth and is tapped out at the rate of 800 ℔. daily. These bottoms contain most of the impurities, as well as much gold and silver.[1]

Blast Furnace Smelting Operations.

Magnetic Iron Oxide in Copper Blast-Furnace Smelting.—A silicious oxidized ore, containing a little copper (0.5 to 2 per cent.) and 30 to 40 oz. Ag per ton, was fluxed with magnetic iron ore and limestone to produce a silicious slag of 47 to 49 per cent. SiO^1; 15 to 19 FeO; and 30 to 32 CaO

[1] *Eng. and Min. Journ.*, LXXXIV, 673.

and MgO; matte was added to the charge to the extent of 5 per cent. to serve as a collector, and for this charge 14 per cent. of coke was found sufficient. The hight of the furnace was increased to give a smelting column of 17 ft., and was run with a cold top as in silver-lead smelting. The blast pressure was at first 16 oz., but toward the end of the run was increased to 32 oz. per sq. in. The slag was hot and fluid, the copper in it being 0.3 per cent. and the silver 0.86 oz. per ton. The resultant matte contained 30 to 35 per cent. Cu and 250 to 400 oz. Ag per ton. It went through the furnace with but little change, except for the increased amount of silver. The furnace, when it went out of blast, was found to be free from accretions.[1]

Furnace Feeding Car.—A. B. W. Hodges has designed, and is using at the Granby Works, a new and effective charge-car for mechanical feeding. It is a car of W-section with short straight sides on the wings and in the center of the letter. In the central hollow beneath the W is placed the supporting truck, which runs on floor-rails. On the top corners of the cars are lugs carrying four small car wheels. These engage a cast-iron rail built in the brickwork of the side of the furnace. As the car enters the furnace these rails engage the topwheels and take the loads a little before the truck leaves the floor-rails. There are two or three cars in the charge train. These cars are loaded and weighed at the ore bins. The outer wings of the W constitute the doors, which can be unlatched and, by a pull of the latch, the charge dropped, placing it at the center or on the sides of the furnace at will. The car remains in the furnace 30 to 45 seconds, and outside of it from 8 to 15 minutes.[2]

Cost of Roasting in the O'Harra Furnace.—When running without stoppages this furnace will roast 50 tons of Butte concentrates to 7 per cent. S, or at the rate of 61 lb. per sq.ft. of hearth area. Its defects are due to constant breakages of the propelling chains at red heat, and to the tearing of the hearth owing to catching of the rabbles.[3]

Handling Ore to Blast Furnaces.[4]—The Messiter system of handling fuel, and fluxes from the receiving bins at the smelting works, the sampling, stocking, reclaiming and feeding of the same to the blast-furnaces is shown in the typical plan, Fig. 3. There are three unloading tracks, two of them commanding a double set of 125-ton receiving hoppers into which the railroad cars are unloaded; the third is at a lower level and brings the high-grade ore alongside the sampling mill to be there unloaded direct to the mill. Troughed conveying belts, A_1 and A_2, are beneath the hoppers, and a shaker feed (which may be set under the discharge spout of any bin) regularly supplies the escaping material to the belt. This feed mechanism is mounted on a car and is astride of the belt. It is a shaking

[1] *Eng. and Min. Journ.*, LXXXIII, 817.
[2] *Min. and Sci. Press*, XCV, 594.
[3] Peters, "Principles of Copper Smelting," p. 103.
[4] *Min. and Sci. Press*, XCV, 528; *The Mineral Industry* XV, 249.

pan which is actuated by a small motor capable, when one bin is empty, of being moved to another. The stroke and inclination of the pan can be varied, as also the size of the hopper outlet, thus varying the supply of material as needed. Conveyer B receives material that is carried by A_1 and A_2, and delivers it to the crushers of the sampling-mill, or it may pass on by conveyer D. If it goes through one of the crushers, the rejected portion may go on by conveyer D, while the sample goes to the sampling

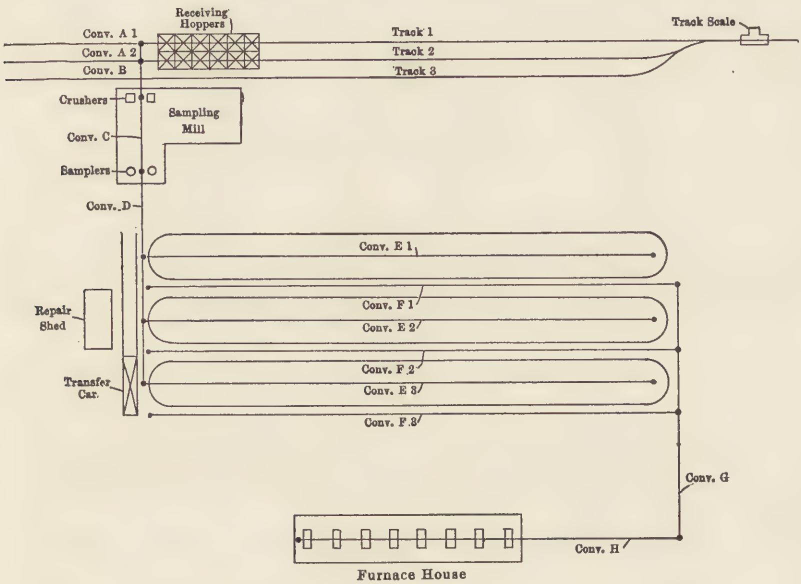


FIG. 3.—MESSITER SYSTEM OF CHARGING BLAST FURNACES AT CANANEA.

apparatus, and the rejects return to the main stream on D. A tripper on D delivers, as desired, to either of the bedding conveyers, E_1, E_2, or E_3. The storage beds are made by depositing the material in long narrow piles. A high-speed traveling tripper, depositing it, travels constantly from end to end of the bed upon an overhead skeleton structure. The tripper working as above described gives a uniform distribution, so that all portions of any cross-section of the bed are alike.

At each edge of the beds rails are laid upon the floor, and upon these the reclaiming machine advances. This is a bridge spanning the width of the bed and reaching beyond it sufficiently to take in its trench conveyer, marked conveyer F_1, F_2 and F_3, in Fig. 3. Under the forward side of the bridge is suspended a scraper conveyer of special design operating in a steel trough consisting of a nearly horizontal bottom-plate and a vertical back-plate. The conveyer flights are inclined to the axis of the conveyer and are provided at their front ends with plow-shaped extensions, which

project beyond the bottom-plate. All material within reach of these flights is swept upon the plate and carried along to its end, where it falls through a short chute to the trench conveyer above mentioned. A triangular harrow, covering the entire cross-section of the beds, is mounted at the front of the reclaiming machine so as to be adjustable in its inclination, and to be slowly but powerfully reciprocated. The teeth of the harrow dislodge or tickle the ore so that it runs down, to be taken away by the scraper conveyer. An electric motor drives both conveyer and harrow and another motor serves to advance the reclaiming machine automatically into the pile at any desired rate, or to run it at higher speed when it is being transferred from one bed to another. The movement of the machine from bed to bed is accomplished by a motor-driven transfer car. A reserve machine is kept ready in case of need. For proper storage capacity, especially where no roof is needed above the storage space, the whole cost of the bedding and reclaiming system is less per ton of ore than for any method of storing in bins from which the ore is to be drawn by gravity. The trench conveyers F_1, F_2 and F_3 deliver to a conveyer G, and this to conveyer H, which sends the complete charge into the blast-furnace building above the respective furnaces.

When ready to commence bedding, the operator at the shaking feed-car is signalled to begin delivering a certain lot of ore from one or more of the receiving hoppers. He will open a hopper gate and keep delivering ore until the lot is all gone. He then signals the millman, who changes the chute in the sampling tower to suit the new lot, and signals so that the operator at the shaking feed-car can send along that lot. Thus the work proceeds until the hoppers are empty. No other control is needed in the bedding operation. Where foul slag is to be returned to the charge, it is crushed and sent by a conveyer to conveyer D. Otherwise it may be loaded upon railroad cars and put in one of the receiving hoppers. When briquets are made, they may be piled while wet on the long edges of the bedding floor next to the trench conveyers, into which they are later gently swept by the reclaiming machine.

Let us take the case of a plant using 10,000 tons weekly and having these beds as shown in the typical plant, Fig. 3. Each bed may be regarded as being in any one of these stages. In the first stage the bed may be regarded as an overflow receptacle for materials over and above what is needed for a bed in the second stage. This second stage is where the bed becomes completed and is a reserve bed, ready for use when the one being emptied is exhausted. In any particular work the bed in the third stage will be in use supplying the furnaces. The materials received during the week will be deposited according to circumstances. If the second-stage bed is full and of the desired composition, the rest of the materials will go to the first-stage (overflow) bed. If the second-stage bed is either empty or

partly filled, the week's receipts will be first used to complete this bed and the excess, if any, will go to the overflow bed. The composition of any bed in SiO_2, FeO and CaO will be decided on and any carload, previously determined, is sent to the proper bed, being marked to that effect. When the proper quantity of any component of a bed has been delivered, any further quantity of it goes to the other bed. The rule is that any carload is to be sent to the second-stage or reserve bed, unless it is of a composition to raise any component of that bed above the assigned limit, when it must go to the first-stage bed, unless it exceeds the limit there also. Should this happen the car must be held until another bed is started. A fourth floor and bed can be added if needed, which then insures that 30,000 tons are available.

If the weekly supply drops off, the beds become smaller and can still be made up and used, but it makes no change in the routine of handling the materials. If the supply of any particular ingredient, as for example the fuel, occurs, the first-stage bed will be held open until the desired ingredient arrives. It appears that in this system: (1) The reclaiming machine will operate equally well on a low as on a high or large bed. (2) The travel of the ore-bedding tripper is adjustable, so that a shorter bed, or two or more on the same floor, can be made up. (3) As soon as a bed, even if small, has the right composition it can be used. Any such beds can be added to while being used by adjusting the tripper to discharge on the unused portion of the bed. Thus the charge can be changed at any time. (4) A small lot of ore will make but little change in the composition of the bed, over all of which it has been distributed. (5) A narrow chute placed in the path of the stream issuing from the tripper will take a true sample of the bed. This sample can be closed at any time, speedily analyzed, and the contents of the bed adjusted as desired to make it of accurate composition.

The reclaimer operator can adjust the movement of his machine to suit the speed of the furnaces, or to keep the receiving bins supplied. He will be guided in this by the feeders on the feed floor. It can be said that the total cost of handling from the unloading bins into the furnace, including labor, power, repairs and belt renewals, will be 5 to 7c. per ton. To this must be added the cost of unloading, dependent upon the kind of cars used. Extra metallurgical advantages result as follows: Extreme accuracy and uniformity in charge composition; thorough and intimate mixture of the ingredients; close control of the charging to the furnaces.

The ore-bedding tripper travels forward at the rate of say 500 ft. and returns at the rate of 250 ft. per min., or three minutes in all. With ore coming at the rate of 240 tons per hour, we have 12 tons deposited each trip, or 48 lb. per linear foot of the bed. Any variation, due to overlapping of layers of this or other lots, will therefore inappreciably affect

the bed. The machine advances into the bed at the rate of $\frac{5}{8}$ in. per min. The harrow at the front of the machine (adjustable to the needed angle even when running) agitates and rolls down the slope to the conveyer all of the loosened front of the slope of ore, and at the bottom all portions, coarse and fine and of all the components, is mixed to give an average mixture. Error on the part of the weigher (as at night) is avoided, and the bed is checked by samples taken before using.

The defects or limitations of other systems are as follows: (1) The running out of bins or small storage units; (2) variations in chemical composition and in the amount of fine and coarse in different parts of the bin; (3) difficulty of layering an equal thickness over the bed; (4) irregular caving of the working face of the bed; (5) error in weighing up small charges; (6) imperfect mixing; (7) irregularities due to careless weighing of charge, especially at the small hours of the morning; (8) difficulty of getting a pack on the furnace without undue segregation of the charge.

These defects are indicated, even with the same charge and under similar conditions, by irregularities in furnace working and by variations in slag composition, and it is these defects which are overcome under the present system.

Increasing the Hight of a Smelter Stack Without Shutting Down.[1]—A 60-ft. extension was put on the 200-ft by 25-ft. diameter steel stack of the Copper Queen smelting works, at Douglas, Arizona. A staging was erected beside and embracing the stack until its top was reached. The 60-ft. extension was built upon this, and later moved over upon the original 200-ft. stack and secured, after which the staging was removed. Two 15-in. I-beams were laid down as rails across the staging to carry the load, and upon and parallel to them was placed two other 15-in. beams on which rested the extension. Nests of rollers were interposed between the rails and the carrying beams. The new section weighed 40 tons. The actual time of moving the extension over upon the old stack was about three hours, jacks being used to push the load along; frequent stops were necessary to await the cessation of vibration. The smoke from the old stack interfered more or less during erection. When the new section had moved over so far as to half cover the old, the gases promptly adopted the new channel.

Charging of Pyritic Blast-Furnaces.[2]—On each of the long sides of such a furnace the door-sill was replaced by inclined receiving plates set at an angle of 40 deg. to the horizontal and 4 ft. 2 in. on the slope. As the ore was tilted into the furnace from the charge car some segregation took place. The coarser lumps shot to the opposite side of the furnace, while the finer material ran more slowly and reached the center of the furnace,

---

[1] *Eng. and Min. Journ.*, LXXXIII, 237.
[2] *Min. and Sci. Press*, XCVI, 593.

forming a ridge of finer material there. The process repeated at the opposite door gave the coarse at the sides, the fine in the middle. The blast consequently came up at the sides, leaving the central part dead and unpenetrated by the blast. When the surface of the charge in the furnace was raised an opposite effect resulted. Then the charge, backing up on the receiving plates, gradually worked down, the coarse material rolling to the center, while the fines clung to the sides. This produced crusts, to avoid which the furnace charge had to be carried just to the proper hight. These crusts could be easily barred off, or could be cut away by feeding an excess of sulphide ore.

The ores thus treated were self-fluxing, but carried no more sulphur than would produce the required matte-fall. To keep the slags as low as 0.3 to 0.4 per cent. in copper, and the matte at from 40 to 50 per cent., it was found that there should be 4 to 5 per cent. sulphur in the charge, and the lack of it in the ore had to be made up by treating poor pyrrhotite. To correct this and to insure the proper distribution of the charge two deflecting plates were hung longitudinally in the furnace so as to deflect straight down the coarser part of the charge. There resulted an improved distribution, and the required matte-fall was attained without using the barren pyrrhotite. The furnace also ran faster, gave a cleaner slag, and a lower-grade matte. The drawback to this method was that the plates were in the way, became warped and wore out.

Copper Mattes.

Constitution of Ferro-Cuprous Sulphides.—Fig. 4 represents the freezing

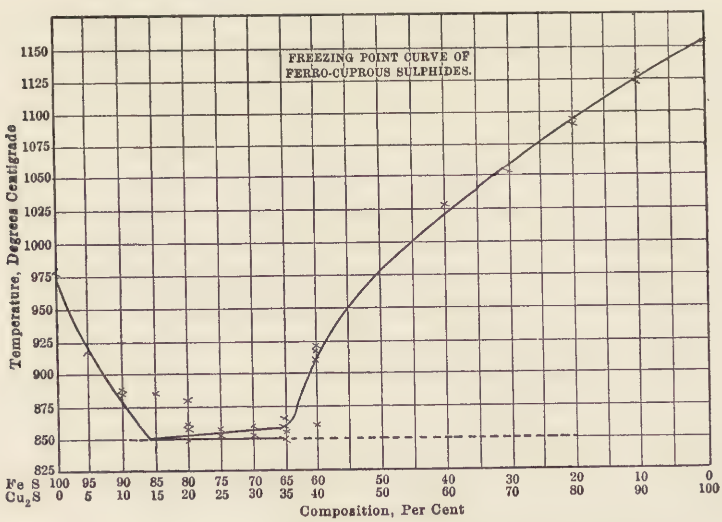


FIG. 4.—FREEZING POINT CURVE OF FERRO-CUPROUS SULPHIDES.

points of a series of mattes containing iron sulphide (FeS) and cuprous sulphide (Cu_2S), in the proportions given. The eutectic was a matte containing FeS 80 and Cu_2S 20 per cent. having a freezing point of 850 deg. C. In summarizing the results of the experiments the following conclusions were arrived at: (1) Ferrous and cuprous sulphides form no chemical, but a euctectiferous compound. (2) The structure of the eutectic of ferrous and cuprous sulphides becomes merged into pure ferrous sulphides. (3) The limited reciprocal solubility of the two sulphides diminishes along the cuprous sulphide branch of the curve, slowly at first, then more quickly. Solubility is complete beyond the alloy FeS 20 and Cu_2S 80 per cent.[1]

Treatment of Copper Matte at Mansfeld.—The 40 per cent. matte from the blast-furnace is roasted in hexagonal kilns from 25 down to 12 per cent. S, a kiln treating 5 tons daily. The sulphur fume resulting is converted by the chamber process into sulphuric acid. The roasted matte is now melted to form white metal of 75 per cent. Cu in small reverberatory furnaces. This white metal is treated: (1) By the Ziervogel method. (2) By the reaction or "direct" Welsh method, where part of the matte is roasted, then melted down with raw white metal. (3) By the new Gunther electrolytic process.[2]

Elimination of Iron, Sulphur and Arsenic in Bessemerizing Copper Matte.[3] —The experiments were performed in a 96x150-in. barrel type converter, using an average charge of 9 tons of matte. The samples were taken with the blast on, by means of a long handled conical bowl ladle, at 10-minute intervals.

Three tests were made, the results being given graphically by Figs. 5, 6, and 7. Not taking into account the "dope" (or copper-bearing material) added to the converter, and assuming the slag throughout the operation to be of the following composition: SiO_2, 35; FeO, 56 (Fe, 43.6); Al_2O_3, 4; Cu, 1.8; and other bases 3.2 per cent.; we obtain by calculation the slag in tons as shown by the shaded area. The loss of copper in the slag is 1.5 per cent. It will be noted that the FeO, as formed, takes up SiO_2 from the lining. At the time of pouring 3.4 tons of slag have been made. This is indicated as having been quite removed and in 10 minutes after beginning the second blow, 0.3 ton has again been formed. One may note that up to the end of the first blow and with 68 per cent. Cu in the matte, there is but little loss of sulphur. The copper seems reluctant to let go of it and, at least, not until the protective influence of the iron has been removed. The air is slow in starting to burn the iron, and when this occurs the reaction, $Fe+O=FeO$, is intensified by the heat of formation of the resultant slag. During the first part of the second blow sulphur is eliminated

[1] *Bimonthly Bulletin*, A. I. M. E., Jan., 1907, p. 25.
[2] *Eng. and Min. Journ.*, LXXXIV, 673.
[3] *Bimonthly Bulletin*, A. I. M. E., Jan., 1907.

according to the reaction, $Cu_2S+3O=Cu_2O+SO_2$, the Cu_2O forming with the production of much heat. The Cu_2O thus formed, acting upon

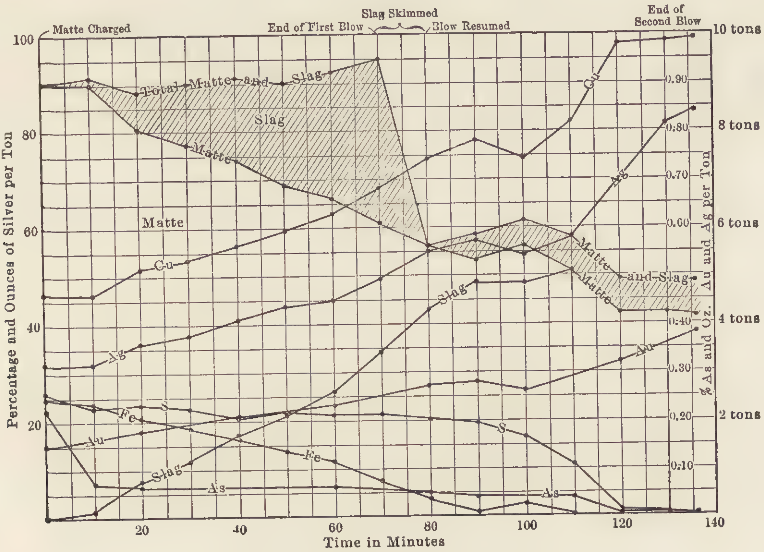


FIG. 5.—NEW CONVERTER, MAR. 7, 1906.

the cuprous sulphide ($2\ Cu_2O+Cu_2S=6\ Cu+SO_2$), causes a speedy elimination of the remaining sulphur and reduction of the copper to blister

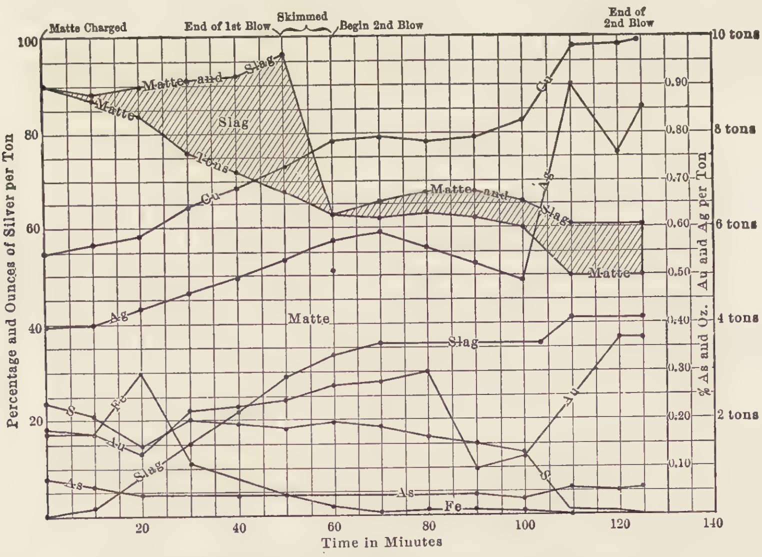


FIG. 6.—CONVERTER MADE THREE CHARGES, THEN CLEANED OUT.

up to 120 minutes, after which but little is done in the remaining 16 minutes before the final pour.

Test No. 1, Fig. 5, was made with a new converter and yet we find a plus recovery of 1.5 per cent. of the gold and of 12 per cent. of the silver. This must have been obtained from the addition of the "dope" used. Both matte and slag are plotted in tons. The line above the matte, limiting the shaded space, expresses the total weight of matte and slag present in the converter, on the assumption that all slag present has been skimmed before the second blow begins. More slag forms during the second blow. The diagrams seem to indicate that when the copper begins to reduce, it takes up arsenic from the slag.

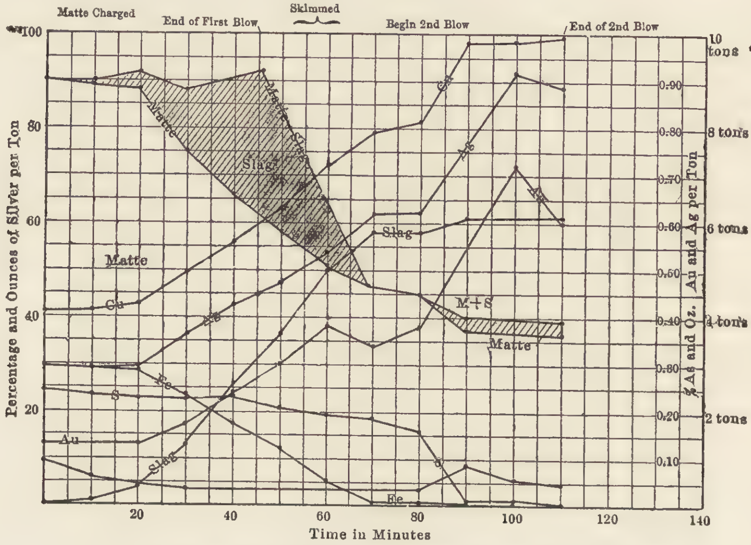


FIG. 7.—CONVERTER BLOWN TO WHITE METAL, THEN WASHED OUT BEFORE USING.

Lining and Drying Copper Converters.— H. L. Charles, superintendent of the Bingham Consolidated Mining and Smelting Company, Salt Lake City, Utah, describes his method of drying out converters. The lining is rammed around a core and the tuyere openings are plugged from the inside with clay. The converter is then poured full of molten slag and allowed to stand from one to three hours, when the slag is poured off. The slag coating which remains slowly cools, and in about 8 hours the converter shell is ready for use. The bottom should be tamped 2 in. lower than when slag is not thus used, and the core made 3 in. longer and 2 in. wider to allow for the slag-coating. It should be punched off or broken through at the tuyeres before using. The lining corrodes more evenly and lasts one to three charges longer than when the drying-out

is effected with fuel, and fewer extra converter shells are needed per stand.[1]

A Hook for Removing Converter Crowns.—When bessemerizing copper matte, a collar of matte and slag blown from the charge forms at the

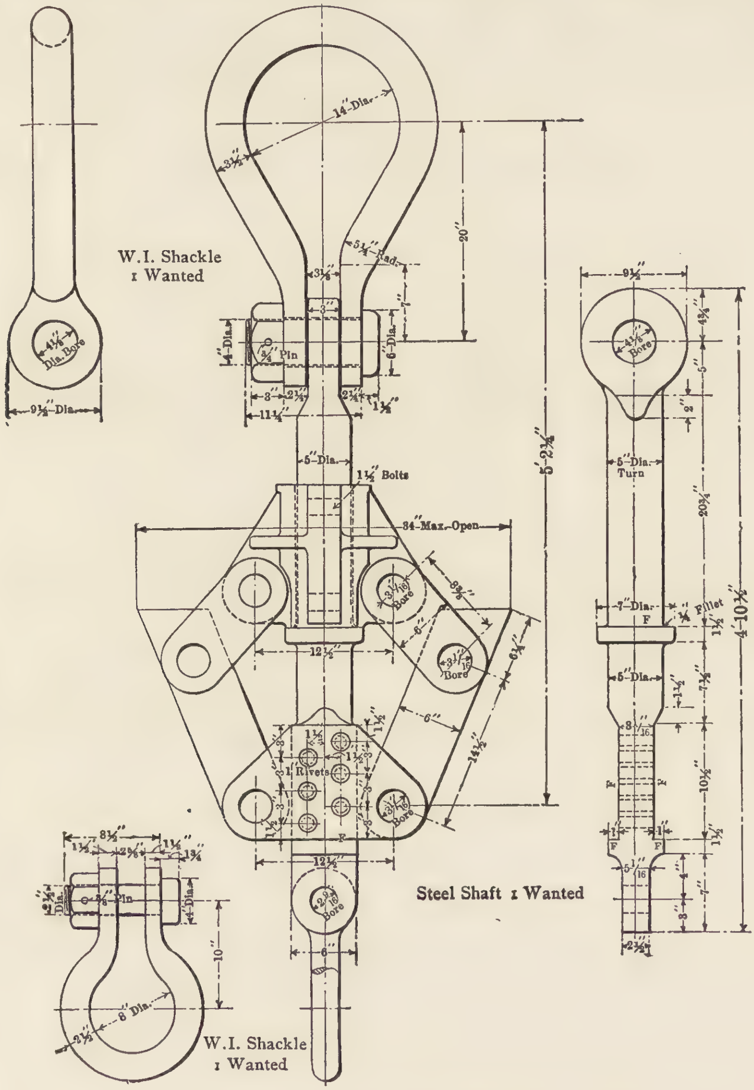


FIG. 8.—HOOK FOR LIFTING CONVERTER "CROWNS."

throat of the converter. To remove this a hook or grab, shown in Fig. 8, is used. When the hook is lowered by the crane through the throat, the

[1] *Eng. and Min. Journ.*, LXXXIII, 1046.

collapsible arms close, but on being withdrawn they open beneath the collar and the crane, acting with sufficient force, tears it away. It at times adheres with such force that the entire converter can be lifted. To take the hook out of the collar, the crane-man hooks on to the lower end of the grab.[1]

Cast-Iron Track-Rail Under Converter.—Cast-iron rails of T-section, 8 in. high with a base 6 in. wide and in lengths of 5 ft., have been successfully used to carry the converter-mold carriage under a converter. They are laid in concrete. They are solid and little affected by ordinary wear.[2]

Converters of the Mammoth Copper Company, Kennett, Cal.[3]—The equipment consists of two hydraulically operated stands and eight shells, 96 in. diameter by 150 in. long. As in the electrically operated converters of the Orford Copper Company, next to be described, the shell is so shaped above the center line as to insure a secure lining, to bring the parting joint high on the converter, and to facilitate thorough ramming of the lining material or ganister.

There are 16 Repath individual tuyeres. For turning the shell each stand is provided with a pressure cylinder of 18 in. diameter, having a stroke of 7 ft., and capable of revolving the converter 180 deg. The piston is provided with four special metallic packing rings, designed to lessen the wear, the leakage and the delays generally due to soft packing. The upper end of the cylinder is bolted to the lower flange of the supporting "A"-frame, and it is provided with independent heads. It is possible to take the piston and rod out through the top. The "A"-frame or stand is of solid cast-iron box-section with the upper part designed to receive the turning mechanism, which consists of a cast-iron rack and a cast-steel shrouded spur-gear. The rack is connected at its lower end to the piston and is guided above, opposite the center line of the shaft. The spur-gear is keyed to a hollow steel shaft, having at the driving end a universal coupling for attachment to the shell. This attachment is arranged quite as described for the Orford converters. The following data are applicable to a converter of this size: Weight of shell, 19 tons; weight of lining, 44 tons; total weight of shell newly lined, 63 tons; concrete in foundation, 1200 cu.ft.; grade of matte used (copper), 40 per cent.; first charge (new lining), $8\frac{1}{4}$ tons; average charge, 10 tons; maximum charge, 14 to 16 tons; air pressure, 12 to 15 tons.

Assuming six charges per stand per 24 hours and 26 working days per month, the capacity of each 96x150-in. converter working on 40 per cent. matte, will be 1,248,000 ℔. of copper monthly.

Electrically Operated Copper Converter.—The Allis-Chalmers Company, Milwaukee, Wis., has furnished for the Orford Copper Company, Constable

[1] *Eng. and Min. Journ.*, LXXXIV, 211.
[2] *Eng. and Min. Journ.*, LXXXIV, 499.
[3] *Min. and Sci. Press*, XCV, 375.

Hook, N. J., three converter stands and nine shells, each 48 in. in diameter by 126 in. long. The shell weighs 13 tons and the lining 28 tons. The initial charge is $5\frac{1}{4}$ tons of 40 per cent. matte and the finishing charge as much as 9 tons, while the average is $6\frac{1}{4}$ tons. A pressure of 10 to 12 ℔. is required and the capacity of a stand is 780,000 ℔. per month of 26 days.

The shell varies from the usual form in that the sides above the tuyere level are carried up straight until they meet the top, thus keeping a thick lining at this critical portion of the interior, and making the parting joint higher and farther from the corrosive action of the molten contents. The heads are of cast-steel, with the riding ring completely encircling them, and are reinforced with ribs for stiffness. There are 14 Repath individual tuyeres each fitted with a Dyblie ball valve and secured to the wind box by swing bolts. The discharge opening of the valve, which is at right angles to its inlet, projects several inches inside the shell and through a cast-steel stuffing box, bored out to suit the projection on the tuyere and arranged with an asbestos-packed joint. This projection permits the brick lining at the shell to be fitted securely around the end. Naturally, each ball valve and its seat is self contained, and the valve can thus be readily taken out and replaced. At the center of the air ends of the shell comes the end of the wind box, fitted with a ball joint flange; this connects to the stationary air nipple of the blast connection, made flexible to suit differences in line due to differences in the center line of the shell. When the shell has been placed in position the lever, shown in the perspective view, is pushed forward and thus moves a nipple into connection with the ball joint of the head.

Each stand is provided with a 30 h.p. motor geared to the main drive shaft, so that the maximum speed of the converter shell is $1\frac{1}{2}$ r.p.m. The motor shaft is direct-coupled to the worm shaft, the coupling serving as a brake wheel for a solenoid brake. The main drive shaft is hollow and is fitted at the driving end with a universal coupling, arranged to suit the alignment of the shell. On the head of the shell is a groove which matches a tongue on the coupling. When placing a shell in position, the coupling is turned till the tongue is vertical, thus allowing the groove to drop over the tongue with side clearance. This is taken up on each side by keys, which are tapped into place with a light hammer, thus ensuring rigidity of movement of the shell.

Beneath the converter a Bennetts pouring spoon is hung on a swinging steel arm secured to one of the roller frames. This permits the spoon to be adjusted to suit the pour of copper, and the position of the molds, which rest on a truck just below. The three stands are operated by one man from a pulpit.

The Laist & Tanner Movable Converter Hood.—This hood differs from the ordinary one in that it can be moved bodily back out of the way

when the converter has to be removed or returned to place. When in position over the converter it forms the front of a square flue as shown in the elevation, Fig. 9. This movement is effected by an air cylinder and,

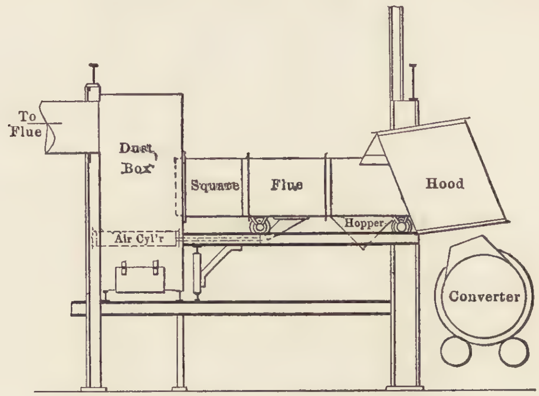


FIG. 9.—MOVABLE CONVERTER HOOK.

in the backward position, the end of the square flue closes against the back wall of the dust box, thus closing the outlet flue. Hoppers placed, one on the flue, the other on the dust box, provide means of drawing off the accumulating dust. The installation has proved effective in removing the sulphurous fume issuing from the converter, and thus in adding to the comfort of the workmen.[1]

Copper Refining.

Copper Refining by the Rio Tinto Company, Port Talbot, Wales.[2]—This works treats precipitates, the result of leaching operations on Rio Tinto ores, and converter anodes from the same source. These are unloaded directly from vessels to the works. Coal can be delivered with the same facility, and the plant adjoins the railroads. There are five refining furnaces of 30 tons (67,200 lb.) each, but they are not utilized to their full capacity, the output being 20,000 long tons of copper yearly. The furnaces are charged by overhead charge cars. The slag is skimmed at the side instead of at the front of the furnace. The charge is introduced at 6 p.m. and is ready for dipping at 10 a.m. the following morning. Bull ladles, similar to those used on Lake copper and holding upwards of 500 to 1000 lb. are here employed. They are suspended from an overhead-beam. The ladle handles are carried in a universal joint, and the bowls are counterweighted as in the Lake Superior practice. The copper is cast into molds and the ingots, on solidifying, are dumped into a bosh, lined or fitted

[1] *Min. and Sci. Press*, XCV, 400.
[2] *Eng. and Min. Journ.*, LXXXIV, 111.

with a perforated basket or cradle. When the cradle has been filled it is lifted bodily by a traveling crane and sent to the stock room. The reverberatory slag containing 4 to 6 per cent. (?) copper, is sent to a rectangular blast-furnace, the slag and copper being drawn into molds mounted on a carriage. Materials are handled electrically, current being supplied from generators driven by direct connected Westinghouse high-speed engines of 425 r.p.m.

Smelting and Refining at Swansea.[1]—Nickel, especially when accompanied by arsenic, is particularly detrimental to high-class copper, and in making the best qualities of bronze and brass. Cobalt, however, can be easily removed in refining, producing a blue-colored slag. Owing to the scarcity of low-grade pyritic ore the enrichment of the "ore metal" (matte of 18 to 20 per cent. Cu) is effected by the addition of precipitate, or cement copper, and rich slags. Furthermore, when the matte has been brought to the stage of white metal, it is not now roasted, but is melted down with a sufficient quantity of cement copper and lime to produce a pimple metal. This metal is now "dry sweated," or roasted, to give a product which, with additional white metal, will give blister copper. If arsenic, antimony, lead, nickel, etc., have not been sufficiently removed at this stage, the copper is "thrown back" before tapping, to pimple pitch, by the addition of blue or white metal. In some works basic hearths are used at the roasting stage. Refining consists in a slow melting, followed

TABLE III. — PROPERTIES OF COPPER.

		Electrolytic to Rolling Copper I	Mansfeld to A-refined II	Mansfeld to A-refined III	Mansfeld Blister to B-refined IV
Blister	O. (per cent.) Sp. gr.	0.613 8.6078	0.870 8.5226	0.854 8.4864	1.280 8.5044
Coarse fibrous	O. (per cent.) Sp. gr.	0.0352 8.7088	0.339 8.7524	0.372 8.7205	0.393 8.7188
Fine fibrous	O. (per cent.) Sp. gr.	0.292 8.7609			
Dense poling	O. (per cent.) Sp. gr.	0.039 8.6536	0.136 8.6812	0.220 8.7363	0.233 8.6806
Tough poling	O. (per cent.) Sp. gr.	0.029 8.1945	0.030 8.5242	0.040 8.5192	0.051 8.5770

by flapping or rabbling, and an addition of Chile saltpetre and lime to assist in the oxidation of impurities. Now follows the poling.

The most effective way of removing arsenic is to use, with the arsenic-bearing mineral, raw pyrite, the reaction being, $As_4O_6 + 7FeS_2 = 7FeS + 2As_2S_2 + 3SO_2$; the arsenious sulphide escapes in volatile form. The resultant slag often contains 75 to 77 per cent. silica, due to the presence of

[1] William Bettel in *South African Mines*, Dec. 22 1906.

particles and pieces of quartz floating in it, while the molten portion is rarely over 45 per cent. silica. Such slag is not, however, resmelted in the blast-furnace and if any portion of it is foul, this is sorted out and sent back to the reverberatory. Unfortunately the sulphides of the other impurities are not so easily removed, and where bad copper is produced it is electrolyzed or made into bluestone.

Variations in the Properties of Copper During Refining.—Table III gives the specific gravity and the oxygen contents of copper in progressive stages of refining and indicates what occurs at different stages in poling. The blister copper increases in specific gravity as the oxygen is eliminated until the maximum density of 8.890 is attained, when the oxygen has been reduced to 0.250 per cent., after which the density decreases from the absorption of gases SO_2, H and CO; so that at the finish the copper is in the neighborhood of 8.550. The electrical conductivity of the electrolytic copper (I) is 58 S.E. (Siemen's Einheit).[1]

Size of Charges in Copper Refining.[2]—At the Chrome plant of the U.S. Metals Refining Company, Chrome, N. J., a maximum charge of 417,000 ℔. was cast at one charge on a Monday morning, the final portion of the charge having been put in on Sunday. The casting of it into anodes of 500 ℔. each was performed in 5 hours, 15 min., or at the rate of nearly 80,000 ℔., per hour. The average size of six consecutive charges was 308,564 ℔. each, also cast at a speed of 80,000 ℔. per hour. With charges of 200,000 ℔., the limiting condition was that of charging where, with the aid of an air lift, it was 35,000 to 40,000 ℔. per hour. To perform this in practicable time with 300,000-℔. charges, it has therefore been necessary to have four charging doors, using a mechanical charging crane as described in THE MINERAL INDUSTRY, XV, 305. With this two men can charge 300,000 ℔. an hour.[3]

Hydrometallurgy of Copper.

Feasibility of the Extraction of Copper from its Ores by Wet Processes.—In the United States the local conditions, the qualities of the ores, the liberal provisions of capital, and the technical skill of the operators, have given wet methods unusual opportunity to prove their value; yet we do not know of any important copper-leaching plant, excepting one or two in Arizona which are extracting oxidized copper from jig tailings. These methods, thus far unsuccessfully tried, should be undertaken on a large scale only under the best metallurgical advice. The success of the wet

---

[1] The Siemens unit is the reciprocal of a resistance of the column of mercury 1 m. long by 1 mm. diameter, while an ohm resistance is that of a corresponding column 1.06 m. long. The resistance of pure copper being one 55.86th of of mercury will have a conductivity of 55.86x1.06 = 59.21 Siemen's units corresponding to 100 of the Mathiessen standard.

[2] *Eng. and Min. Journ.*, LXXXIV, 171.

[3] At the Michigan Smelting Works, Houghton, Mich., the largest charge cast was 397,000 lb. The ordinary charge varies between 200,000 and 325,000 lb., with an abundant supply of copper, and especially where it was a case of remelting and casting a pure grade of copper. F. I. Cairns, the manager, states that there would be no difficulty in casting a charge of 500,000 lb. The hourly casting rate on common-sized castings is 60,000 lb. L. S. A.

methods at Rio Tinto in Southern Spain cannot be cited elsewhere, since the conditions there are peculiarly favorable for success, and cannot be duplicated in North America. A drawback to these methods is the fact that when the ore contains precious metals they cannot, as in smelting, be extracted.[1]

Precipitation of Copper from Mine Waters at Butte, Mont.[2]—At the St. Lawrence and Anaconda mines the copper is precipitated and collected by either of two systems, the box or the tower system. The box system consists of an extensive series of troughs, 1 to 3 ft. wide and 1 ft. deep, into which scrap-iron is thrown, and through which the mine water passes. The copper precipitate, after being washed through many compartments, settles to the bottom of a tank which is cleaned up monthly.[3]

The tower system consists of a tower or rack, 3 ft. wide, 20 ft. high, and 30 ft. long, which carries slatted floors or stages set 18 in. apart vertically, the slats being 2x4 in. scantlings set 3 in. apart. Scrap-iron is placed upon the floors. The water is pumped to a trough at the top of the tower and is distributed by it evenly over the floors. The lighter scrap is soon eaten up, while the heavier pieces are scraped and cleaned from adhering copper and set back in place. The precipitate is washed to the bottom and then to tanks alongside of the tower. The copper precipitate is sold locally at the market price less 4c. per ℔.; the usual content of copper is from 50 to 60 per cent., although it is sometimes as high as 84 per cent. By allowing water to seep through weathered tailings a precipitate of 70 per cent. has been obtained from the resultant solution. The plant can be run the year round because the water is warm. There are some 75 concerns working in Butte by these methods. The tin cans and other scrap, some of which is shipped in, costs $10 per ton. It is estimated that it costs 8c. per ℔. to recover the copper in this way.

Electrolytic Methods.

Electrolytic Treatment of Copper Matte at Mansfeld.[4]—The new Gunther process for the electrolytic treatment of 75 per cent. matte (white metal) is as follows: The matte is cast into plates or anodes 40 in. square by 2 in. thick, which are suspended in the lead-lined vats by means of T-shaped copper straps tinned on the ends, these ends being embedded in the matte. The method of casting these anodes, upon which the success of the process depends, is a secret. The cathodes are thin copper plates. The anodes must be removed before they become so thin as to crumble away and contaminate the slime. The electrolyte is heated to 70 deg. C. by means of steam coils, and is thoroughly aerated and circulated. The

[1] Peters, "The Principles of Copper Smelting," p. 2.
[2] *Eng. and Min. Journ.*, LXXXIII, 1029.
[3] *The Mineral Industry*, XIV, 188.
[4] *Eng. and Min. Journ.*, LXXXIV, 673.

pressure is not more than 0.5 to 0.75 volt, and a current of 30 amperes per sq.ft. is used. Iron, nickel and cobalt go into solution, small particles of the anodes (copper, silver and sulphur) collect at the bottom of the vat as slime, and the pure copper is collected on the cathodes. To clean up the slime, the clear electrolyte is syphoned off, removed and filter-pressed. It is next treated with a hot solution of acetylin tetrachloride to remove the sulphur, this sulphur separating out on cooling; the residual liquid is used again. The silver-bearing mud now receives a sulphatizing roast, and is further treated by the Ziervogel method, the residue, after extraction of the silver, being used to neutralize that portion of the electrolyte which is daily withdrawn for purification.

Electrolytic Extraction of Copper from Its Ores.—The anode, consisting of an ore containing iron sulphide, is placed within a porous compartment or diaphragm separating it from the cathode; the electrolyte consists of an acid solution of copper sulphate. Air is blown into the anode compartment to oxidize the iron sulphate as completely as possible. The oxidized solution is then withdrawn and treated with sulphurous acid outside the electrolytic vat in order to reduce the ferric sulphates to ferrous sulphates, thus setting free sulphuric acid; this mixture is used for lixiviating fresh ore.[1]

Separation of Copper from Cyanide Solutions.[2]—Both gold and copper are precipitated by an electrolytic method upon lead cathodes $\frac{1}{16}$ in. thick. These cathodes, 22 in. wide and 4 ft. long, remain in the electrolytic precipitating vat 20 to 30 days, increasing in weight in that time, by 8 to 12 ℔. of a hard, firm deposit of the two metals. These cathodes are now removed to another electrolytic tank, called the acid tank, and are immersed in a 2 to 3 per cent. solution of sulphuric acid, forming anodes, while the cathodes are now lead plates of $\frac{1}{16}$ in. thickness. The newly-called anode, with its load of copper and gold, hangs within a frame having cloth sides. As the copper dissolves it passes through the cloth of the frame and precipitates on the lead cathodes, where it slimes and falls to the bottom of the tank. The gold, released from the copper, collects at the bottom of the anode frame. The lead anodes, when cleaned, are again used as cathodes in the precipitating tank first mentioned. To prevent short-circuiting in the acid tank, the liquid is circulated by a lead air injector, which raises it at one end and returns it to the opposite end of the tank. At the clean-up, the acid solution is syphoned off, the anode frames removed from the box, and the gold slimes taken from them. The copper slime is flushed through $1\frac{1}{2}$-in. holes at the side of each compartment, run outside the building through a cement launder molded in the floor, and discharged upon a filter-bottom tank. When drained,

---

[1] Brit. pat., No. 25489, 1906.
[2] *Eng. and Min. Journ.*, LXXXIII, 512.

it is dried, sampled and shipped away for treatment at a copper works.

Electrolytic Extraction of Copper in Poland.[1]—At Medzianka, Russian Poland, an experimental plant of little more than one-half ton daily capacity has been put in for the electrolytic treatment of ore containing copper glance, azurite and malachite. The ore is crushed, mixed with 5 per cent. of damp clay, and bricked. The bricks are dried and kiln-roasted, so as to convert them as far as possible into sulphate and oxide, recrushed, and then leached in lead-lined tanks with the regenerated sulphuric acid solution (containing 5 per cent. H_2SO_4) from the electrolytic vats. The solution from the leaching tank, containing 5 per cent. Cu and 1 per cent. H_2SO_4, passes through a filter press, the clear filtrate going to a storage vat. Thence it is drawn off to fill the electrolytic vats above mentioned, each of 35 cu.ft. capacity. Each vat has nine anodes, consisting of lead plates closely enveloped in cotton cloth bags, and eight copper stripping-plate cathodes. The electrolyte is agitated by reciprocating stirrers between the plates. The current-density is one ampere per sq.ft., and the power consumed is 1.04 kw., or 1.6 h.p. per hour per pound of copper deposited. The electrolyte is exhausted in 35 hours, and, now containing 1 to 1.5 per cent. Cu and 5 per cent. H_2SO_4 as above stated, is returned to the leaching vats to dissolve fresh ore. The cathodes, after having been for a month in the vats, attain a thickness of 1 to 1.25 in. and are then removed. They are of greater purity than ordinary electrolytic copper. There are four vats in series taking 900 amperes at 2.25 to 2.5 volts. A single attendant takes care of the plant.

The vital point in the success of this process consists in the employment of a closely fitting bag or envelope of thick cotton duck. This, soaked with sulphuric acid, excludes the iron salts which would be otherwise oxidized and precipitated. The copper is well precipitated and removed from the electrolyte.

Alloys and Properties of Copper.

Alloys of Copper and Aluminum.[2]—These alloys can be satisfactorily made at a single melting if thoroughly stirred and at the proper casting temperature. The linear contraction in cooling varies between 1.83 and 2.34 per cent. among the rich copper alloys, and has a uniform value of 1.66 per cent. on the aluminum alloys of 0 to 3.76 per cent. copper. The specific gravity varies regularly from 8.9 for pure copper to 7.23 for an alloy containing 13 per cent. aluminum. Fig. 10 shows graphically the results of tensile tests upon sand castings when slowly cooled, and Fig. 11 of such castings when quenched in water from 800 deg. C. Fig. 12 shows the results of tensile tests on chill castings when slowly cooled from 800

[1] *Oest. Zeit. für Berg und Hüttenwesen,* 1906, p. 387. *Revue Universelle des Mines,* XIX, 75.
[2] Eighth Report of Alloys-Research Committee, English Inst. Mech. Engrs. Jan. 8, 1907.

deg. C. and Fig. 13 on such castings when quenched in water from that temperature. Finally Figs. 14 and 15 give the results of tensile tests upon

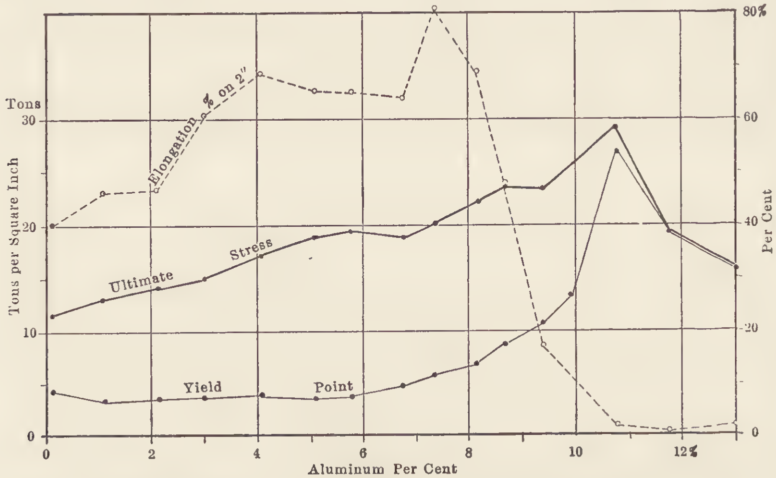


FIG. 10.—TENSILE TESTS ON SAND CASTINGS. SLOWLY COOLED FROM 800° C. (1472° F.)

the bars rolled to $\frac{13}{16}$ in. diameter when slowly cooled from 800 deg. C., and when quenched in water.

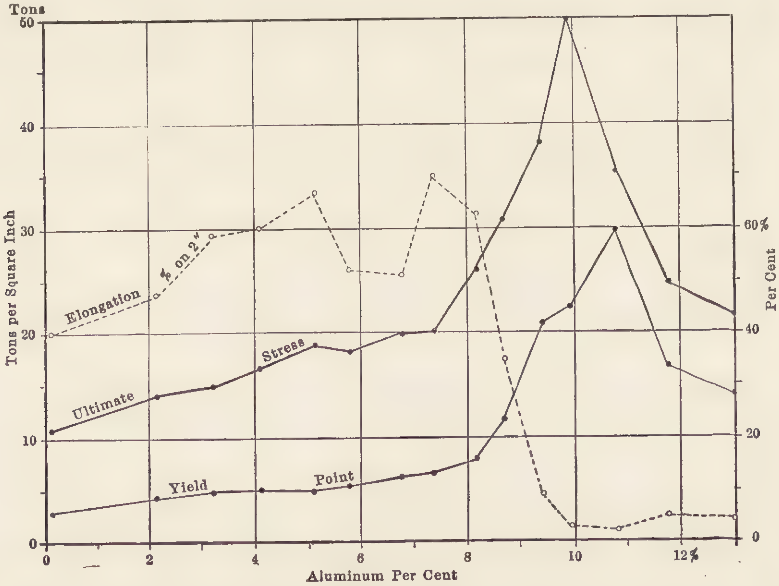


FIG. 11.—TENSILE TESTS. SAND CASTINGS. QUENCHED FROM 800° C. (1472° F.) IN WATER.

From these figures we may deduce the following results: For the indus-

trially serviceable alloys the limit may be put at 11 per cent. aluminum. Between these specified limits the alloys may be divided into two classes:

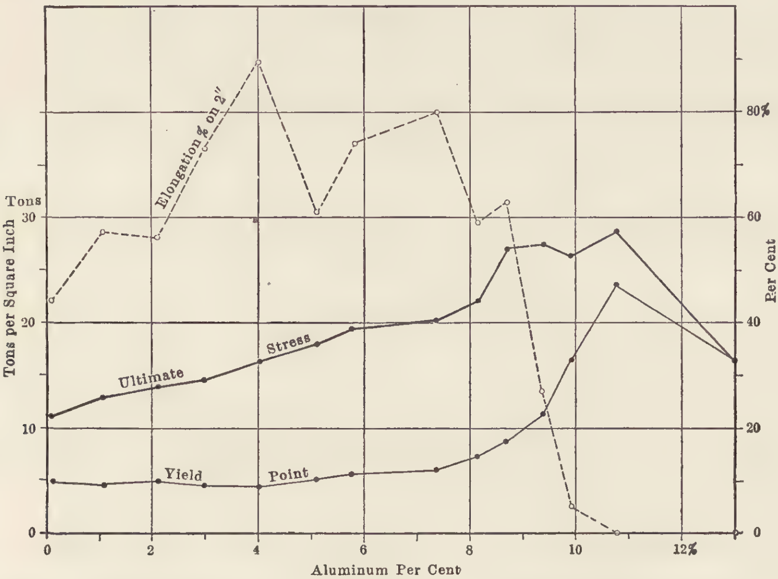


FIG. 12.—TENSILE TESTS. CHILL CASTINGS. SLOWLY COOLED FROM 800° C. (1472° F.)

(I), those containing 0 to 7.35 per cent. aluminum; (II), those containing 8 to 11 per cent. aluminum. Class I represents material of moderate

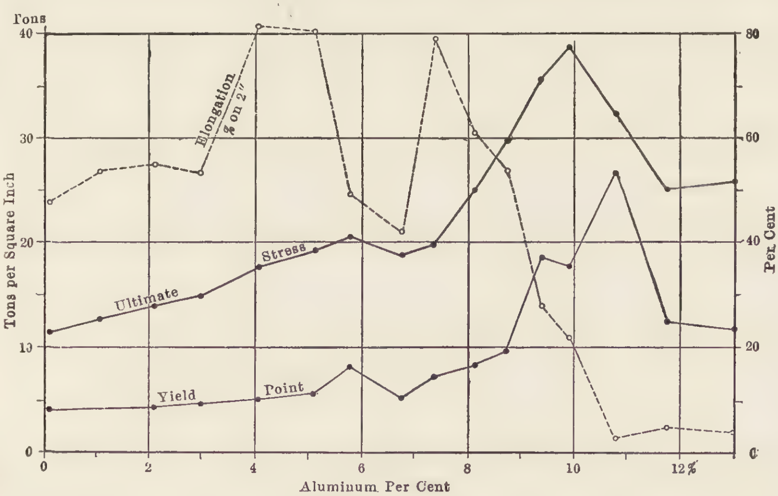


FIG. 13.—CHILL CASTINGS. QUENCHED FROM 800° C. (1472° F.) IN WATER.

ultimate stress, but of good ductility. This class is not sensitive to heat

treatment but is improved by mechanical work. They behave well under torsion tests. In class II come alloys of good ultimate stress, and of good

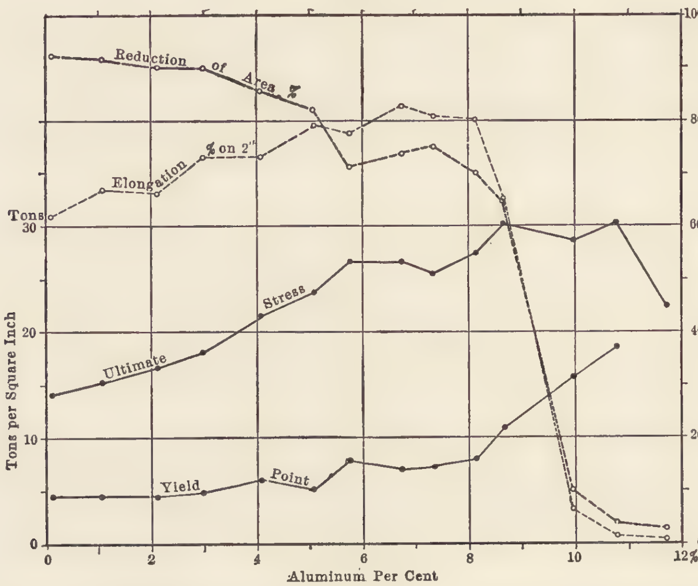


FIG. 14.—TENSILE TESTS. BARS ROLLED TO $\frac{13}{16}$ IN. SLOWLY COOLED FROM 800° C. (1472° F.)

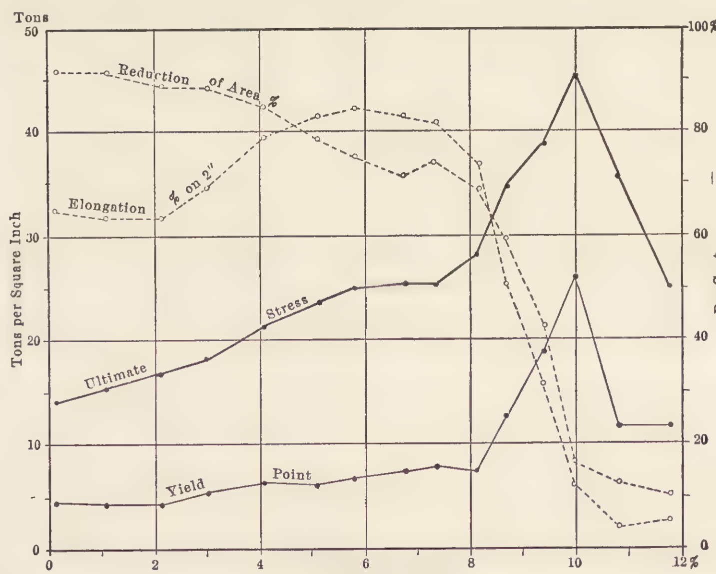


FIG. 15.—BARS ROLLED TO $\frac{13}{16}$ IN. QUENCHED FROM 800° C. (1472° F.) IN WATER.

ductility, and some of them are sensitive to heat treatment. As a whole

we find that the sand castings are inferior to those cast in a chill, and these in turn to rolled bars. Ten per cent. aluminum-bronze, when cast in a chill, is as strong and ductile as when rolled. Its properties are injuriously affected by heat treatment for temperatures over 300 deg. C., and this limits its usefulness as a valuable alloy.

In the series of corrosion tests, alloys containing 1 to 10 per cent. aluminum have shown themselves to be practically incorrodable by sea-water, either alone or when bolted to a plate of mild steel. Fresh water tests gave a different result, the alloys being corroded to some extent.

Properties of Copper-Cobalt Alloys.[1]—The electrical conductivity of these alloys rapidly varies, and keeps dropping to the point of 5 per cent. Co to 95 per cent. Cu, and more slowly to the alloy of 90 per cent. Co. Here it rises as rapidly to the conductivity of pure cobalt. It thus appears that between these two points conductivity remains low because of the presence of solid solutions in mechanical mixture. The fusibility curve of Fig. 16 indicates the same thing.

Fig. 16 represents the curve of fusibility of copper-cobalt alloys, begin-

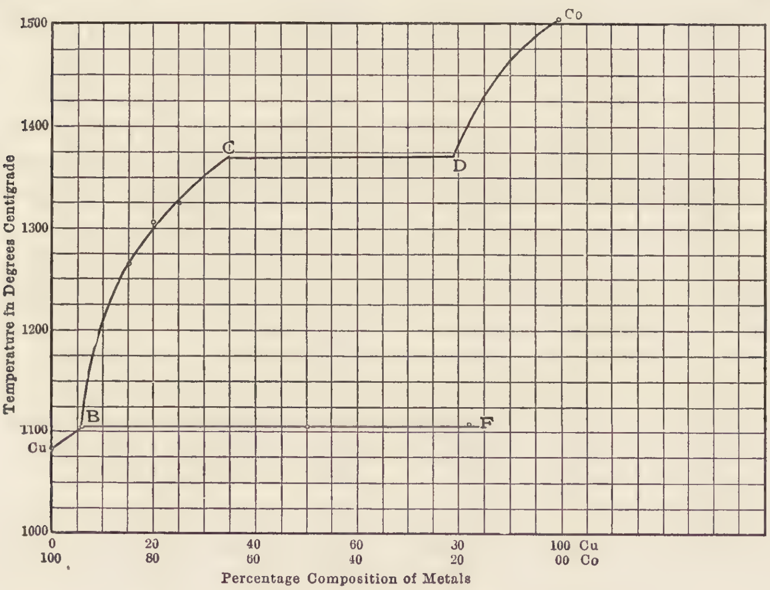


FIG. 16.—CURVE OF FUSIBILITY OF COPPER-COBALT ALLOYS.

ning at the melting point of pure copper and rising with increased cobalt contents to the melting point of cobalt at 1505 deg. C. The two branches of the curve indicate that the liquid alloy, before solidifying, separates into two layers. The concentration limit of solid solutions of cobalt in

[1] *Revue de Metallurgie*, IV, 983.

copper is 6.5 per cent., as shown at the point B of the curve. The further addition of cobalt results in a rapid rise of the point of fusion of the upper portion, while the lower layer solidifies at 1105 deg. C. When the alloy contains 30 per cent. Co, the lower layer remains as before, while the upper solidifies at 1370 deg. C.; this condition of affairs continues the same to the point of 70 per cent. Co, as shown by the branch of the curve C-D. The further addition of cobalt from this point results in a rapid rise to the melting point of pure cobalt. Formation of the layers mentioned can take place but slowly, because of the small difference in specific gravity of the two metals (Co = 8.7; Cu = 8.8). This separation is quicker and better defined in the presence of carbon than when melted in graphite crucibles with a cover of charcoal.

Copper Mining and Smelting by the Tennessee Copper Company.

By J. Parke Channing.

During the early part of 1907 the continued shortage of labor in the South prevented any material increase in the production or the tonnage treated from the mines of the Tennessee Copper Company and the depression in the price of copper in the latter part of the year of course discouraged any increase at that time. Since then the labor situation has materially improved, The present production of the Tennessee Copper Company is in the neighborhood of 12,500,000 lb. a year, though the mines, railway and smelter have been so equipped that if conditions demand the production can readily be increased to 20,000,000 lb. per year.

Mining.—At the principal mine of the company, the Burra Burra, the 2.5-ton skips have been replaced by five-ton skips and the steam pressure on the hoisting engine raised proportionately, it previously having been reduced from boiler pressure before entering the cylinders. This idea was held in view in the original equipment of the property and has worked out with remarkable success. For handling the men, man-cages similar to those used at the Quincy mine on Lake Superior have been introduced.

The shortage of labor prevented much development work during 1907, but this has now been taken up again and the shafts at the various mines will be sunk during the present winter. An order has been placed with the Bucyrus company for a special type of dipper shovel operated by compressed air, and this will be installed underground on one of the Burra Burra levels for the purpose of loading ore from the underhand stopes into the mine cars. It is hoped that this will produce a material saving in the cost of loading and will also solve to a very large extent the difficult problem of securing the common labor necessary for this very important part of mine operations. On the railway a large number of cars have

been rebuilt and one new 100,000-lb. locomotive has been added to the motive power.

Smelting.—At the smelter an immense improvement has been made in the metallurgical operations, due very largely to increased mechanical efficiency of the charging system and the slag disposal system. Heavier locomotives and larger and stronger cars have been introduced. Heavier rails have been installed so that delays from derailments have been practically overcome. It has also been found that the tuyere area of the furnaces was not too small to permit of proper punching of the crust inside the blast furnace. However, by paying particular attention to this point and keeping the tuyeres well open the volume of blast has been very materially increased with a consequent better running of the furnaces. In fact, four furnaces at the end of the year were handling as much ore as six furnaces at the beginning of the year. It is intended to enlarge very materially the tuyere openings by making out of each pair of tuyeres a long slotted tuyere and it is hoped that the tonnage of the furnaces will be still more largely increased. The increased volume of air has resulted in an increased amount of FeO in the slag and a consequent reduction in the amount of silica necessary or else an increase in the grade of the matte. During the early part of 1907 the first matte went about 10 per cent. copper. At present it seldom runs below 19 per cent. copper and at times with a new furnace and the tuyeres well open, a 30 per cent. matte has been made continuously for 48 hours. There is a possibility that in time the second or concentrating operation may be entirely eliminated, though it is still too early to predict such a radical change. Now that the problem of making sulphuric acid is of as great importance as the production of matte, it is necessary to consider very seriously the volume of air blown into the furnace. Still, however, indications point to the probability that all air within reasonable amounts blown into the furnace combines with either the carbon of the coke or the sulphur and iron of the ore.

Converting.—In the converting department the only change has been in the final determination to abandon the resmelting of converter slag and the pouring of that portion which is fluid into the settlers. This has been a much mooted question, but in Tennessee it has been definitely proved that unquestionably the commercial gains are far in excess of any possible metallurgical loss. The converter floor is kept remarkably clean by pouring the last dregs of smelted material from the converter ladles into small hand pots which are then run out and dumped. The skull in the ladle is dumped on the floor, breaks into pieces and is not cemented together by the usual small amount of slag left in the ladle under old conditions.

Sulphuric Acid.—The sulphuric-acid plant was completed just at the end of 1907 and at the present time the gases are going through it and such

minor adjustments are being made as are necessary in any plant of this size. Sufficient progress has been made to show without doubt that the process will be a success. It is found that the proper grade of gas may be obtained by an adjustment of the various dampers. It is too early to go into any of the details of the process, but it is hoped that later on a careful technical description of the results achieved may be published.

The Dwight & Lloyd Sintering Process.

By Arthur S. Dwight.

The metallurgical treatment of fine material, particularly fine sulphide concentrates, is a problem which is pressing more and more heavily upon the metallurgist. The blast furnace has an unquestioned superiority over the reverberatory when the material to be smelted can be supplied to it in reasonably coarse form, but with fine ores and concentrates it is at a disadvantage, and many metallurgists have been convinced that the return of the reverberatory into preëminence is inevitable. The development of the Montana type of reverberatory of large capacity has been indeed remarkable and reflects great credit upon the metallurgical skill of the engineers who have been responsible for its success.

The Dwight & Lloyd system of sintering fine materials is the result of a serious effort to devise a satisfactory and economical substitute for roasting and briquetting, in the treatment of fine materials such as concentrates, pulverulent ore, flue dust, etc., in preparation for blast furnaces. The metallurgical and commercial results that have already been attained in the development of this process are decidedly satisfactory and it has now been working under daily operative conditions long enough to establish its reliability and average cost factors. Indeed, it looks as though a practical and economic solution of this vexatious question has been found which may restore, to a great degree, the prestige of the blast furnace.

Blast Roasting Processes.—The process may be considered as coming within the class of those which may be designated "blast-roasting" processes i.e., those which involve the use of controlled air currents. In the same class, generally considered, are the Huntington-Heberlein, Carmichael-Bradford, and Savelsberg processes. These methods were devised originally for the treatment of lead sulphide ores, but it has been shown that copper ores are equally adaptable.

The effectiveness of an air blast in hastening roasting has long been recognized, but the very rapidity of the reaction defeated its own ends, for the sulphides quickly melted together, and impeded further oxidation. Huntington and Heberlein found that they could prevent this premature matting of the sulphides by mixing lime with the charge. The particles of lime served to keep the particles of galena separated

during the oxidizing period, thereby favoring individual-roasting of the sulphide particles, as distinguished from mass-roasting. Until this precaution was observed, blast roasting never had a chance to show what it could do, and the first time the partially roasted charge so prepared was transferred to the converter and treated with an air blast, the remarkable possibilities of roasting in that way must have become apparent. But in order to account for the astonishing results, it seemed for a time necessary to assume the existence of compounds and reactions of lime heretofore unsuspected or considered as impossible.

Principles of the Processes.—The above assumptions were widely discussed and largely accepted. Careful research, however, proved them to be fallacious. Now, it is quite satisfactorily established that the function of the lime is mechanical rather than chemical, and a brief statement of the principles involved would be simply that the charge be so made up that the sulphides are mixed with other ingredients, so chosen that during the first or roasting period they shall remain more or less inert, acting as isolaters to prevent mass-action among the burning sulphide particles, but which subsequently, during the second or agglomerating period, shall be capable of uniting with the metallic oxides produced by the roasting, and with the other ingredients of the charge to form silicates, or other compounds which will become sufficiently viscous at the temperatures developed by the reactions to cement the mass together more or less into what we call a sinter.

Defects of the Blast-pot.—It must be conceded, that even the best form of blast-pot or "converter" which has heretofore been used for these processes, is open to many serious disadvantages, chiefly mechanical, of which the following list will suffice:

(1) The process is necessarily intermittent, requiring much handling of material, and constant attention in filling the pots, stopping blow-holes, etc.

(2) On account of the agitating effect of the blast as it issues from the top of the pot there is a considerable quantity of the charge (particularly near the top) which, though partially roasted, does not have a chance to sinter. In order to reduce the percentage of these unsintered fines the capacity of the converters is made very large, usually about 10 tons to the charge. Under average conditions the quantity of fines will amount to from 10 to 30 per cent.

(3) On account of the mass-action in such a large converter, the central part of the sinter cake is apt to fuse together into a solid mass of slag, and thus lose the peculiar, cellular, coke-like structure which is such an important desideratum of the product. It also involves much extra labor in breaking up the mass after it has been discharged from the converter.

(4) It has been shown by H. O. Hofman (see paper on "Lime Roasting

a Galena Concentrate," *Trans.* A. I. M. E., Vol. XXXVIII) that the sintering action in an ordinary converter, blown from below, starts at the place of ignition at the bottom and moves slowly to the top; and the time during which a given particle is exposed to the maximum heat of sintering (1000 to 1200 deg. C.), is not more than a minute or two, the temperature curves for a given layer in the converter showing a very sudden rise to, and a very sudden fall from, the critical temperature-time of sintering. When this slowly rising plane of fire reaches the top surface of the charge in the pot the operation is completed. It must be very evident therefore, that in a converter of many tons capacity most of the space is occupied by particles which are either waiting to be sintered, or having been sintered, are waiting to have the rest of the charge finished. Hence the greatest part of the capacity of the converter is used for storing and not for actually treating its contents, and we are therefore forced to the conclusion that, considered strictly as a metallurgical furnace, the capacity-efficiency of the Huntington & Heberlein converter is very low.

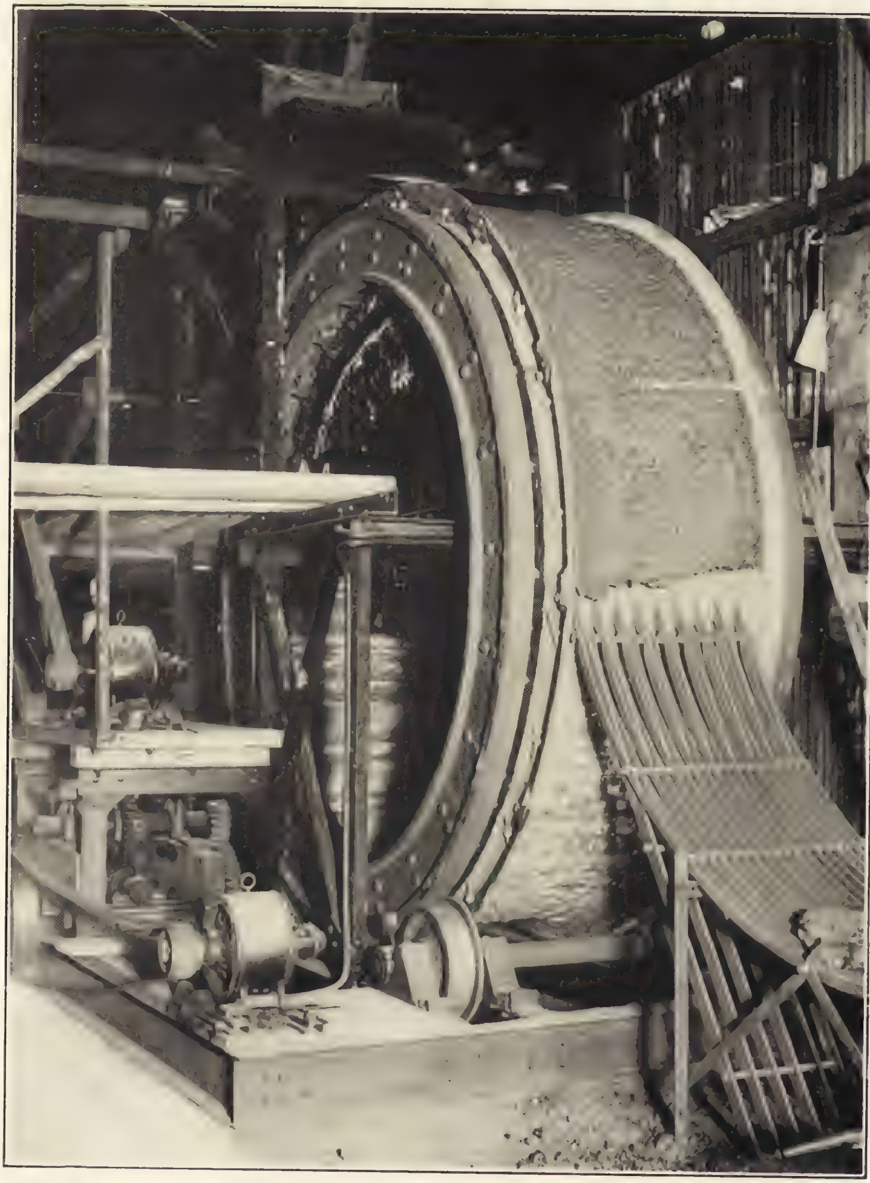
Ideal Conditions.—If the above reasoning be correct, it follows that ideal results as to character of product and economy of operation would be obtained: (*a*) If the treatment could be made continuous; (*b*) if the material could be presented and maintained in a quiescent condition; (*c*) if a thin layer, charge, or succession of charges could be employed.

After some very extensive experiments in this direction by R. L. Lloyd and myself, we found that one very important matter was to follow such a course that, from start to finish, the particles throughout the entire mass were prevented from agitation, or disturbance, and particularly those particles in that part of the mass in the region where the gases made their exit. In other words, we discovered that complete quiescence is one of the incidents of superior sintering, which apparently requires that at the instant of maximum temperature all the particles of the mass in that locality must remain in practically the same relative position and in close contact, each with its neighbors. There are many ways of accomplishing this effect, and with a great variety of apparatus, employing up-draft, down-draft, or side-draft.

By effecting the treatment of ore while it, as a mass, is undergoing movement, new lines of economy, not attainable by any of the earlier and intermittent processes, are opened up.

The Dwight & Lloyd Machine.—While it is not the purpose of this paper to attempt to describe all of the numerous matters incident to this process, and the forms of apparatus that have been devised for carrying it out, a few of the important features will be referred to. For example, a simple and convenient form of the Dwight & Lloyd machine[1], which has given very satisfactory results under actual working conditions

[1] U. S. patents, Nos. 882,517 and 882,518, March 17, 1908.



DWIGHT & LLOYD SINTERING MACHINE, DRUM TYPE, SHOWING STRIPPING GRIZZLY.

extending now over a considerable period of time, is that shown in the accompanying engraving, which machine is at present in continuous daily operation, under regular smelter conditions, treating copper concentrates and flue dust.

One lot of about 50 tons of the product of this machine, made from high-grade galena concentrates, carried more than 50 per cent. lead and less than 3.5 per cent. sulphur. The capacity of the machine on this charge was 1.3 tons of dry charge per hour, or somewhat over 30 tons per day, with a 4-in. layer and a blast pressure of 4 oz. No preliminary roasting was necessary, nor was any form of lime used in the mixture. A total power consumption of 12 h.p., as shown by meter readings, was sufficient to operate the machine and suction fan. The gases during the treatment of this lead ore were passed through a bag-house and the lead loss was proved to be a negligible quantity.

The general arrangement of the apparatus is shown in engraving. It consists of a pair of circular rims of iron carrying a set of cast-iron, herringbone grates, the whole forming a drum-shaped structure, resting on rollers like a copper converter. The drum acts essentially as an endless conveyer, and is caused to move slowly about its axis by the friction of the drive-rollers. Inside the drum and occupying the top quadrant of the circle is a stationary suction box, connected with a suction fan. The moving rims make an air-tight joint with the edges of the stationary suction box. The material to be sintered is fed in a thin layer upon the grated face of the drum from an overhead ore-hopper, immediately after which the stream of ore passes under the igniter which may be a series of gas jets, an oil flame, a charcoal brazier, hot roasted ore or even hot sinter, whereby the top surface of the ore stream is kindled uniformly across the whole width of the conveyer. The roasting action so begun is maintained and augmented by the streams of air which are sucked down through the moving layer of ore as it passes across the suction box. When a 4-in. layer of ore is used, which has been found to be a convenient thickness, about 20 minutes will be required for the sintering action to be completed and the speed of the periphery is so regulated that the layer shall be completely sintered down to the grate by the time it reaches the far end of the suction box. The discharging is done automatically by the pointed grizzly frame which strips the end of the finished sinter-cake from the face of the wheel like bark from a tree.

When the machine is sintering properly the grates do not clog. The best speed for the periphery will depend upon the character of the material under treatment; thus with the leady charge above described, it was about 5 in. per minute. Before being fed to the machine, the ingredients of the charge should be thoroughly mixed and moistened to the proper degree. This moisture not only prevents the fresh charge

from sifting through the apertures of the grate bars, but greatly promotes the activity of the sintering process.

It is evident that this same sequence of operations may be accomplished in many different mechanical ways, and a number of other types of machine have already been designed, built and operated for carrying on this process. The drum type of machine, here shown, has the minimum of moving parts to wear and get out of order and, therefore, presents some strong claims for preference.

Details and Cost.—Contrary to what would be expected, the machine does not get very hot even when continuously handling a large tonnage, and the cast-iron grate bars do not make trouble by burning out.

It will be observed that all the adjustments governing the operation of the machine are under absolute control. The character of the mixture, thickness of layer, speed of travel, ignition, and blast pressure can be set to produce the maximum output and best character of product, and when once set, can be maintained with a minimum of personal attendance. In fact, one man can as easily look out for a battery of several machines as he can for a single machine.

The factors developed from actual experience show (as might be expected) that the cost per ton is considerably less than the best pot-roasting practice up to date.

The range of composition of material from which a suitable sinter can be made, while not unlimited, has been found to be surprisingly wide; in fact much wider as to silica, iron, sulphur, etc., than the limits of composition of the blast-furnace charge of which it is to form a part, and it has also been found that for purely mechanical causes the chemical limits are wider when treating thin layers than they are with the pot-shaped converters of large capacity. This is perhaps most clearly shown by the ability of the machine to treat galena concentrates high in lead without the admixture of any form of lime, as the advocates of the older processes claim to be necessary.

Advantages and Scope.—The following suggestions are presented as to the advantages and the scope of this process:

(1) On account of the peculiar cellular structure of the product, the fine ores destined for the blast furnace can be put into a form which is admirably adapted to facilitate the reduction and smelting of the metals by the blast furnace gases. In this way increased fuel efficiency will be attained and the furnace speed will be greatly increased. This means the lowering of cost factors all along the line as any metallurgist will readily appreciate.

(2) The sintering of the fines will greatly reduce the amount of flue dust made, and what dust is made can be rehandled under most satisfactory conditions. In fact, flue dust is often a welcome addition to the sintering mixture in treating fine sulphide concentrates by this process.

(3) In cases where pyritic smelting is possible from the chemical character of the ores, but is negatived by its fine mechanical condition, the advantages of pyritic smelting may be successfully secured by making a rough separation of the fines from the coarse ore, sintering the fines to form a spongy sinter which, in combination with the coarse ore, will open up the charge in the blast furnace in the manner which seem necessary for the best pyritic work.

(4) In many isolated mining districts where concentrates are produced and shipped to distant smelting works, this process would serve a useful purpose in putting the concentrates into a convenient form for handling and permit of shipping them without the usual mechanical losses. At the same time the shipping weight would be reduced with consequent savings in the freight and smelting charges, and the product would be in a form most acceptable for the smelter.

(5) The process readily lends itself to the successful solution of many special problems in ore concentration and smelting.

(6) On account of the concentrated character of the sulphur gases that can be produced, and the correspondingly small volume, the system may be a convenient adjunct to any scheme for utilizing or rendering innocuous the sulphur in the waste gases. This is becoming an important matter in many districts where the farmers are raising objections to the smoke nuisance.

The capacity of the machine is approximately 1 to $1\frac{1}{4}$ tons per day per square foot of effective hearth area, while the cost of installation is but a small fraction of that of a mechanical roasting plant of equivvalent capacity.

Mining and Smelting at Mount Lyell, Tasmania.

By Robert C. Sticht.

Introductory.—Though the smallest of the six States comprising the commonwealth of Australia, Tasmania[1] is by no means the least significant from a mining point of view. Its mineral resources are unique in character and remarkable in variety. The number of known mineral species exceeds that of any of the other Australian State, but even in respect of the more vital interest of industrial mining, the little island may court comparison with much larger countries. Tasmania surpasses most mining localities of the world through exhibiting, within its limited area, an unusual number of commercially valuable minerals, including ores of most of the useful metals, and it is safe to say that no other mining region can duplicate the existence of similar diverse enterprises, of equal magni-

[1] Tasmania, island south of Victoria, Australia, in lat. 42° S., long. 146° E, separated from the mainland by Bass Strait, about 300 miles wide. Area, 26,375 square miles; population, 180,000; capital, Hobart. Chief ports, Hobart, in the South; Launceston, Devonport and Burnie, in the North; Strahan, on the West Coast. Principal industries, mining, agriculture, ruit-growing (apples), timber, sheep-raising, etc.

tude and importance, within an equally narrow territorial compass. The celebrated Mount Bischoff tin mine, the famous Tasmania gold mine, the magnificent Blythe iron deposit, the well-known Mount Lyell copper mines, all lie within easy reaches of from 14 to 35 miles of each other. These luminaries are supported by a plurality of sound, productive undertakings of less importance, such as the alluvial tin mines of the North East coast (Pioneer, Briseis and others), the interesting lead-silver deposits of the West Coast (Zeehan, Rosebery, Mount Farrell, etc.), the coal mines of the East Coast, and others, which, but for the geographical isolation of the island, and its consequent seclusion from the eyes of the capitalist, would long ago have been supplemented by many more of equal, or perhaps, greater importance. At least one-third of the island is well mineralized and is fit for no other purpose than mining, and, though much of this ore-bearing area may be of low grade, still, the economic minerals are closely distributed, and the generality of ore-deposits is valuable enough to become amenable to perfected processes of treatment, which can be operated profitably, when conducted on a large scale.

As usual under British law, all minerals belong to the Crown, but the right to mine is secured under lease for a term of years, renewable upon expiry, the Crown practically ceding all proprietary rights for the payment of initial application and survey fees, an annual rental, and the annual compliance with certain labor covenants. Mining lands are divided into square sections, usually adjusted to the cardinal points of the compass, the boundary planes vertical, so that extra-lateral troubles are impossible. Terms for gold leases: limit of size, 40 acres; annual rental, £1 per acre; tenure, 21 years; labor covenant, and expenditure, in labor, machinery, or improvements, of at least £10 per acre per annum. Terms for mineral leases (for metals other than gold, and not for coal, freestone, etc.): limit of size, 80 acres; annual rental, 5s. per acre; tenure, 21 years; labor covenant, £2 per acre. Amalgamation of groups of leases is permitted, and larger enterprises avail themselves of the privilege of special bills, or acts of Parliament, in order more securely to define, or to modify, the conditions of the tenure. Water rights: term, 21 years; rental, $1 per acre per sluice-head (24 cu.ft. per minute). Machinery sites, 10 acres: term, 21 years; rental, 5 s. per acre per annum. Prospecting licenses cover 20 acres for gold, 40 acres for mineral; rental, 10s. per annum. Mining easements, i.e., surface rights for trams, buildings, tunnels, waste-dumps, etc., 5s. per acre per annum.

The legal provisions are liberal, and the industry, including prospecting, is protected and encouraged, to which end the mining acts and regulations are constantly being revised and perfected. Their administration is in the hands of a special state department, and litigation on their account is unknown.

In addition, the mining industry of the state of Tasmania is interesting to an unusual degree from the technical standpoint alone. The larger concerns are all well managed, and have originated methods for the solution of their individual problems, which command attention. Local difficulties throughout are very great, yet the work done may be classed as belonging to the most efficient and most economical of its kind performed anywhere.

The object of this article is to lay before the readers of this publication some information regarding past and present aspects of mining and smelting operations at Mount Lyell. As nothing authoritative has appeared in these volumes on this head, and past references, culled from current sources, have frequently been inaccurate, the data may be of interest on the other side of the world.

Location.—The Mount Lyell mining field is situated in about the center of the West Coast of Tasmania, 18 miles inland from Macquarie Harbour (see Fig. 1, locality plan). This expanse of sheltered water is about 20 miles long, by four to five miles wide, and connects with the Southern Ocean by means of an extremely narrow, navigable entrance. Its utility is at present hampered by a depth of only 14 ft. of water (formerly only 9 ft.) on a bar outside of the entrance, and by a shallow, tortuous channel within, so that the reach to Strahan is only navigable for vessels of less than 1200 tons. Three railways run to the shores of the harbor, viz.: the Government railway from Zeehan to Strahan, connecting at the former place with the Emu Bay railway, running northward to the deep-sea port of Burnie; the Mount Lyell railway, connecting Regatta Point with Queenstown (population 4000); and the North Mount Lyell railway, connecting Kelly Basin with Gormanston, or Linda Valley (population together 3000).

The country surrounding the harbor is exceedingly inhospitable and repellent in character, being ruggedly mountainous and covered with extensive forests, almost devoid of animal life, which are rendered impenetrable by the wall-like entanglement of an exceptionally heavy undergrowth. The climate is wet and rough, the West Coast in general differing very markedly in this respect from the rest of Tasmania, with its singularly uniform, sunny, but genial weather. The coast has no redeeming natural resources beyond its mineral wealth, except the prospective utilization of abundant water-power for industrial purposes.

The Mount Lyell mineral field itself is not large, at present actually consisting only of the neighborhood of the Mount Lyell mine, but the country southward to Mount Sorell, being mineralized, is properly tributary to the field, and so is the ore-bearing, but scarcely explored, country immediately to the north. The following remarks, however, apply to the Mount Lyell field in the restricted sense.

FIG. 1.—LOCALITY PLAN OF THE MOUNT LYELL DISTRICT, TASMANIA, (After Government Sketch Map).

Historical.—Tasmania was settled in 1803, and not long after was turned into a penal settlement for the deportation of criminals from England, as was New South Wales before. Convict stations were established on the East Coast, but the plethora of such human consignments from home induced the authorities to select Macquarie harbor, on the remote West Coast, for use as a great natural prison, suitable for the incarceration of the more refractory subjects. The natural features referred to doubtless

were the deciding point, as it was considered that there was no escape, either by sea or overland, to the settlements on the other side of the island. Accordingly, a convict establishment was founded in 1822, on Sarah island, near the head of the harbor, and conducted on the lines of discipline customary at that day. However, the brutalities committed on both sides, no doubt aggravated by the gloomy influence of the locality and its climate, were so shocking, that the place outlived its usefulness and was abandoned in 1833. This period has left melancholy historical traces, but, apart from a little timber-selling and some ship-building, the authorities did not further utilize, or even recognize, the true natural resources of the locality.

In 1840 the country to the north of Mount Lyell was entered for some distance by J. C. Calder, and in 1842 Sir John and Lady Franklin, the the former then governor of the colony, crossed the island from Hobart, the capital on the southern point, through to Macquarie harbor. It is doubtful if either of these expeditions noted the presence of valuable minerals along the route, such, for instance as the occurrence of gold in the creeks. The mineral resources of Australia in general were not actually appreciated, or investigated, until after the discovery of gold in larger quantities on the mainland, in the fifties. There the existence of that metal was known to geologists at the beginning of the decade, but the knowledge was suppressed by the government, on account of the unsettlement of the population which was feared, and which, in fact, did eventually take place, as is well known. The gold-fever, however, did not extend to the West Coast of Tasmania so early. It was not until 1860 and 1862, when the geologist, Charles Gould, on exploring expeditions into the western interior, found that gold did occur here, that the district attracted attention. The difficulties of access from the coast, and the forbidding character of the country itself, however, overawed adventurous spirits seeking an entrance from the sea, while the formidable obstacles opposed to inland progress by the vegetation, militated against incursions from the north. Gould himself, in the year first mentioned, came within a quarter of a mile of the Mount Lyell pyrites deposit, without observing its existence, though it was marked by a very prominent geographical landmark, in the shape of a huge hematite cliff. Thus the extensive mineralization of the district remained unknown for about 20 years more. By this time the famous Mt. Bischoff tin deposit at Waratah, which was discovered in 1871, caused the Coast to assume certain potentialities in a mining sense, so that, in their southward marches from there, prospectors in 1877 located the stanniferous, but now languishing district of Heemskirk. Penetrating the bush still further south, they discovered the lead- and silver-bearing lodes of Zeehan in 1882, but, even prior to this, in 1881, they had come across gold-bearing alluvial within four miles of

the Mount Lyell mine, by working their way up the King and Queen rivers. Allured by the occurrence of payable gold, they pursued the small tributaries of the latter stream to their sources, and finally, crossing a saddle between Mounts Owen and Lyell and descending into Linda valley, they discovered the outcrop of the Mount Lyell mine. This occurred in 1883, while seeking for the source of the metal present in an auriferous drift, lying at the foot of a rocky spur bearing the landmark above referred to, the so-called "Blow," a bold mass of iron ore jutting out of the hillside. They were successful in establishing the existence of gold-bearing limonite at the foot of the Blow, which was so rich as to start an excitement of some duration. By 1886 this became strong enough to induce a fairly extensive line of "lode" to be taken up, in consequence of the existence of further large iron-ore outcrops in a direction north and south, the sequence of which gave rise to the belief that a tremendous gold-reef, or lode, existed, that, it was supposed, followed the line of strike of the strata for a distance of some 1½ miles. Notwithstanding the fact that, in 1884, pyrites was observed to bottom the hematite and limonite, this material remained unheeded, especially as assays of it, made in the Government laboratory at Hobart, returned the verdict that it was of no commercial value. On the strength of the auriferous limonite a 10-stamp gold-battery was erected on the spot in 1887, the plant for which was painfully carted and packed a distance of 28 miles from Strahan, over a barbarous, narrow, largely only corduroyed mountain road, and a mule pack-track, to the site on the intervening saddle. This enterprise, however, soon came to a standstill, for the gold petered out, and in consequence the entire undertaking lay fallow, although the existence of the immense pyritic "lode" by that time attracted some attention. It was not before 1891, however, that the true character of this ore occurrence, as a magnificent copper deposit, was recognized, the appreciation being due to H. H. Schlapp, then at Broken Hill. This gave the incentive to the formation of a syndicate which took in hand the further prospecting of the body, and exploitation was at once initiated on systematic lines, chiefly at the instance of Bowes Kelly, from then until now chairman of the companies working the mine. By 1893 four prospecting tunnels afforded a good idea of the size and importance of the ore deposit. No. 4 tunnel in this year fortunately led to the discovery of a very important secondary enrichment on the footwall of the body, in a drive along the latter, to the south. The bonanza thus discovered was small, but so rich that the sum of £105,000 was netted, in 1893-94, from the sale of only 850 tons[1] of ore.

In 1893 also, Dr. E. D. Peters, well known to the readers of this publication, and T. A. Allan, formerly general manager of the Tharsis

[1] Throughout this article the ton of 2240 lb. is used.

GENERAL VIEW OF MT. LYELL OPEN-CUT FROM THE NORTH, SHOWING EXCAVATION AND TIPS. MOUNT OWEN IN THE DISTANCE.

mines in Spain, reported on the property. Their statements, supported by the phenomenal find referred to, essentially assisted in getting together capital for energetic operations, commensurate with the apparent size and value of the deposit. The usual railway bill was put through parliament at once. The act provided that the railway should connect with Strahan, on Macquarie harbor, a condition which bound the route to the western fall of the range. Construction work on this line was taken in hand as early as 1893, after three different routes, some of 30 miles long each, had been investigated, the trial surveys alone costing the sum of £10,000, a fact mentioned only to give an indication of the difficulties attending the penetration of new country of this nature, the sight lines having to be chopped out with the axe for miles. To gain time, the railway, however, was only completed, from the proposed locality of the smelting plant on the Queen river, for a distance of 14 miles, to a point on the King river (Teepookana), corresponding to the head of navigation in that stream. This portion of the line was finished in July, 1896. Simultaneously, systematic development of the pyritic deposit by means of the opencut system, and on lines originated by the company's officers, together with the erection of a suitable smelting plant, were also started, in 1895. The smeltery was at once arranged for the use of pyrite smelting, recommended by me, notwithstanding the fact that the process had never before been tested on a large scale on regular copper ores, and was not, at that time, generally understood. Two shaft-furnaces and two hot-blast stoves were erected, and completed before the arrival of the railway, the machinery and supplies for them being delivered over the road at the rate of a maximum of 100 tons a month. Smelting operations were started in June, 1896, about a month before the advent of the first train, as soon as a stock of only about 150 tons of coke had been amassed. However, the subsequent success justified the risks then taken. In 1899 complete rail connection with Strahan, as well as with the entire railway system of the island, was completed.

The pyritic ore indications on the strike of the strata, particularly in the locality of the above-mentioned line of hematite deposits, caused a large number of copper mining claims to be taken up on the continuation of the supposed Mount Lyell "lode," and for several years after 1896 a pronounced boom prevailed. With a few exceptions, nearly all of these enterprises have failed, for lack of the necessary mineral foundation, the more promising of them passing into the hands of the Mount Lyell company.

The most important outside property, which time and the trend of circumstances has thus brought into the possession of the Mount Lyell company, is the one formerly belonging to the North Mount Lyell Copper

Company. This has been amalgamated with the Mount Lyell company since 1903. Its principal mine is essentially different from the Mount Lyell ore-deposit, both in point of quality of ore and type of occurrence. The North Mount Lyell company was, however, first founded on the strength of a very much less important showing (the so-called Eastern ore-body), and, notwithstanding extensive tunnelling, remained unsuccessful until, in 1897, the construction of a government cart-road to the North Mount Lyell sections accidentally exposed the outcrop of a rich ore deposit. This has since become, and still remains, the chief metal producer of the field. The support which the former company received from this find led, among other things, to the construction of a 30 mile railway, likewise running to the shores of Macquarie harbor, but on the other (eastern) side of the range, and to the erection of a smelting plant 11 miles from the mine. The latter was at first provided with reverberatory furnaces, in which the ore was treated with limestone and ironstone flux. Later, four small blast furnaces, of a capacity of 50 tons a day each, were substituted for the reverberatories, and a dressing-plant was also finally added. The ore, however, is a very silicious bornite ore, and serious economic difficulties, inherent in its treatment by these means, may be discerned by a glance at its analysis. The operations of the North Mount Lyell company were not successful, and resulted in a financial crisis by 1902, after about 140,000 tons of 12 per cent. ore had been extracted, and partly treated at Crotty, in the King valley, and partly sold, some 80,000 tons being taken by the Mount Lyell company. Finally the North Mount Lyell company opened negotiations for a combination of the two concerns, a union which, in the best interests of the metallurgical and economical treatment of the two ores, might have been effected long before. The amalgamation took place in August, 1903, the new company retaining the name and the administration of the former Mount Lyell company.

The following are the most important other properties which have been taken over by the Mount Lyell company in the course of time, and which possessed more or less productive areas, viz.: Lyell Tharsis, South Tharsis, Royal Tharsis, South Mount Lyell, King Lyell, Prince Lyell, North Crown Lyell, Central Lyell, etc. Of these, the first-mentioned started on a promising career, working a channel of bornite ore, but in a couple of years it apparently exhausted its orebody. Subsequently the mine was purchased by the Mount Lyell company, and has since then worked as an annex to the Mount Lyell mine. As long as it existed, the Lyell Tharsis company sold its ore to the Mount Lyell company. The second and third mentioned enterprises did not last long, owing to the comparative poverty of their ore, and were soon acquired by the parent company. The second company tried to dress

its pyritized schist ores by means of wet concentration, but with signal lack of success, the copper going off in the tailings and the barren pyrites into the heads. The deposits of the other properties enumerated, with the exception of the South Mount Lyell pyritic body, which perhaps stands in some geological relation to the Mount Lyell mass, are not of wider interest. The King Lyell is small, and low-grade, occurrences of cupriferous clay, at the foot of the Mount Lyell mine.

The Mount Lyell district is thus a characteristic "one mine (or one company) camp." The only other companies of prominence, independently carrying on mining in the district, are the Mount Lyell Blocks company, which is treating a large copper-clay occurrence at the foot of the Mount Lyell mine, and the Tasman and Crown Lyell company prospecting a galena and bornite deposit on the other side of Mount Lyell.

The lands held by the Mount Lyell company in this district aggregate 1785 acres, a figure which has been larger in the past, and now represents the retention of only those mineral leases which seem to be of value, the rest having been given up.

Topographical and Geological.—The average annual rainfall at Mount Lyell is not less than 110 in., and is probably twice as great in the higher mountain plateaux. The climate is salubrious and mild, but cool, the temperature averaging about 55 deg. F., seldom going down to the freezing point in the winter (middle of the year), or rising higher than 80 deg. F., in the summer (at Christmas). Frequent and tempestuous winds, however, give it an aspect of severity, which is accentuated by the perennial dampness, while the sky is almost always overcast. The rainfall is responsible for the dense "scrub" and impenetrable "brush" (forest), which grows on heavy layers of peaty humus, forming practically the only covering of the rocky surface. It also causes the formation of difficult bogs and marshes ("button-grass plains") in regions free from forest. The latter, however, is the most serious obstacle in the way of a proper reconnaissance of the country. The vegetation completely obscures the ground, leaving only the mountain streams open for inspection and for natural passage-ways, and the axe is constantly in requisition to open tracks and clear the surface. Bush-fires, notwithstanding their terrors, are hailed (or provoked) as aids to prospecting.

The general trend of the West Coast is SSE by NNW and is paralleled by that of the inland mountain series, the so-called West Coast range, some 20 to 25 miles from the sea. It is not a true range, the arrangement of the mountains not being linear, but in groups. The individual elevations are not connected with each other, as a rule, except perhaps by very low saddles, but are separated by transverse valleys. Single successions of elevations, however, can be picked out, which have an approximately

straight alignment, such as the Geologists range, named by Gould, which runs through the Mount Lyell district. This consists of a row of mountain bosses of from 2000 to 4000 ft. in hight, bearing the names from south to north, of Sorell, Darwin, Jukes, Huxley, Owen, Lyell, Sedgwick, Geikie, Tyndall, and Murchison. They are, in the main, composed of quartz conglomerate, though there is some eruptive rock, presumed to be portion of an extensive plateau of such, still covering the interior of the island. This whole range is about 30 miles long, and the Mount Lyell field occupies a central position. To the west of the southern half of this series of elevations lies a strongly dissected pene-plane skirting the Mount Lyell district, the surface of which has an elevation of from 500 to 800 ft. above the sea. Its edge falls precipitously toward the latter, but is partly encircled by a band of sand dunes and sandhills, some three to five miles wide, bearing vegetation. To the east of the Geologists' range and its northward continuation lies the great central plateau of Tasmania, with its outliers between.

The most important rock of the district, from a mining point of view, is the Mount Lyell schist, which encloses the ore-deposits of the district. It occurs in a comparatively narrow belt, of a northerly and southerly course, and is of a peculiar character. Its geological age is estimated as Lower Silurian, on stratigraphic evidence only. The sedimentary rocks lying to the west of the schist belt, and which consist of sandstones, quartzites and clay-slates, carry interbedded banks of limestone, identified as in part upper, and in part lower Silurian. If, as it would seem, the schists are older than these rocks, then they belong to the lowest horizon of the Palæozoic, if not to the Archaean division.

Petrographically, it is a hydro-mica schist, and belongs to a general category which would formerly have been designated as talcose, but it is devoid of magnesia. Chemically the schist is chiefly an alumina silicate, combined with free silica, and the "talcose" habitus is derived from an obscure mica variety related to muscovite (margarodite, damourite, or paragonite), though apparently unique. The typical color is light-green to bluish-green (due to chlorite, etc.), locally altered to gray and white, particularly in the vicinity of the orebodies. The structure is massive or blocky, schistose, and nodular, but usually finely laminated near the deposits, frequently becoming fissile.

The strike of the Lyell schists is fairly regular and about parallel to the general direction of the coast, varying between 50 and 60 deg. W. by 50 and 60 deg. E., and their dip is 60 to 80 deg. to the SW. Only in the neighborhood of the ore deposits is the foliation disturbed to any great extent, and then not by folding so much as by very numerous faults, i.e., crush-zones. The direction of the ore-channels, as a rule, conforms closely to the strike of the schists.

Directly over the schist rocks lies a mighty layer of quartz-conglomerate, portion of a general covering which must once have covered the entire north-western portion of the island. It is solely composed of quartz pebbles, white, or iron-stained, from light pink to dark red, running in size from the grains of a dense sandstone to boulders of a foot or two in diameter. The cementing material is identical with that of the pebbles. On account of its position on top of the schists, this rock is held to be Devonian, the palaeontological proofs themselves being obscure. The rock is not ore-bearing, as a rule. In the neighborhood of the ore deposits it carries occasional impregnations of iron pyrites, and even of bornite and copper glance, the latter minerals occurring in the lowest division, where the rock merges into a plain, hard quartzite, although in a few instances the pebbly structure is recognizable. But these mineralizations are purely local, and restricted to a very small area, chiefly portions of the North Mount Lyell mine. The dip of the beds, as a rule, is very flat, roughly about 10 to 15 deg. to the west. As in the case of the schists, the foliation is not very much disturbed by folding, although its regularity is interfered with by fault fractures. The strike, as a rule, is north and south, like that of the schists, but varies a good deal locally.

Igneous rocks are not uncommon on the West Coast, and those near the Mount Lyell mines occur in rather extensive masses, approaching to within 1½ miles of the principal ore deposits, but occasionally coming much closer in the form of dikes, though they never come into immediate contact with any of the orebodies. The larger masses on the Queen river are probably pre-Silurian, and consist of diabase porphyrites, on the margin of which quartz porphyrites occur. The intrusive igneous rocks are of uncertain age. They are not abundant, and, as far as investigated, are diabasic in character.

The tectonic structure of the field is comparatively simple. There are no violent contortions of the strata. There is, however, a very complicated system of faults, which has greatly disturbed the regularity of the stratification, or foliation, near the orebodies.

The special geology of the field has been but little studied by competent specialists. A very excellent monograph, embracing the chief features of the district, however, was written in 1903 by Prof. J. W. Gregory, then of Melbourne University and Director of the Victorian Geological Survey[1], applying advanced modern ideas to the interpretation of the field. In this valuable memoir the derivation of the schists from a felspathic base is established, and, in this connection, it is significant that they seem to form an interruption in a long belt of felsitic rocks accompanying the mountain range. Professor Gregory also adduces strong

[1] Now at Glasgow University. The paper was printed in *Trans., Aust. Inst. Min. Eng.*, Vol. X, No. 135, June 1904, but is also separately published under the title: "The Mount Lyell Mining Field, Tasmania, with some Account of the Geology of other Pyritic Ore Bodies," Melbourne, 1905.

microscopic proofs in support of the ascription of a metasomatic origin to the pyritic deposits in the schists.

The orebodies occur on both sides of a nearly straight mountain ridge, or divide, running NNW by SSE, of an average hight of 1500 ft. above the sea, and from 200 ft. to a quarter of a mile wide on top, which extends from Mount Owen, on the south, to Mount Lyell, on the north, and is about two miles long. The western fall of this connecting saddle joins fairly steeply on to the hills and ridges of the strongly dissected country forming the coastal plane. The eastern declivity, however, falls steeply down to the U-shaped glacial valley between Mount Lyell and Mount Owen, Linda valley, the upper margin of which is at an elevation of about 1000 ft. above the sea. The saddle itself serves as a watershed, the waters falling to the west running into the Queen river, while those of the east unite in Linda creek, which, for its part, runs 2½ miles through Linda valley into the King river. The latter, after following the eastern edge of the range, suddenly breaks through same, in a deep gorge, and is there joined by the Queen river, thus draining both sides of the range into Macquarie harbor. These two rivers formed the natural entrance into the Mount Lyell district, and guided the early prospectors to the field. The railway, following the track of these pioneers, now takes the same course. For the purpose of circumventing the gorge, and to shorten the distance, it surmounts an elevation of 750 ft. above the sea, lying between the rivers, which is effected by means of an Abt rack railway.

Finally, it may be remarked that the district, in common with other portions of the West Coast, shows pronounced evidences of glaciation, moraines and other signs abounding immediately at the mines.

Railways.—The Mount Lyell railway is 22 miles long, but freight rates are reckoned on a 31 mile basis, by parliamentary sanction, on account of the 4½ miles of Abt rack section referred to above. The Abt system has splendidly demonstrated its suitability, and has never given any trouble, which is equally true of the permanent-way and the special locomotives. Compared with any other entry into the district it has the undoubted advantage of bunching all the grades in one place. A 1 in 16 gradient is against the (smaller) outgoing traffic, and a 1 in 20 against the (greater) incoming loads. There are four Abt locomotives, four-wheel coupled, with four cylinders; six three-wheel coupled Baldwin adhesive engines are also used over the line. The former haul a gross load of about 80 tons over the hill, and the latter of about 40 tons. There are 135 vehicles. At Queenstown there is a well-appointed railway workshop.

The North Lyell railway has a length of 28 miles, 11 miles thereof on a 1 in 40 gradient against the incoming load. There are three Avonside

six-wheel coupled tender locomotives, one saddle-tank engine and 63 vehicles.

Both lines are of the Tasmanian government gage, 3 ft., 6 in. The Mount Lyell line is actively occupied, but the other runs only on two or three days a week, at present. Both lines follow picturesque routes, the Mount Lyell railway especially being one of the principal scenic lines in Australia. At the sea-port termini each line is provided with extensive wharf facilities, including cranes, etc.

The Mines.

Situation, etc.—The Mount Lyell mine is situated on the northern face of a broad, knoll-like spur off the eastern flank of the connecting saddle, about half way up the latter, and 1½ miles north of its junction with Mount Owen. Its position is in a bay formed by the line of contact of the conglomerate and the schists, on the Linda valley side of the saddle.

The North Mount Lyell mine is situated on the same fall, at the other end of the divide, at its junction with Mount Lyell, at the head of a steep gully, intervening between the flank of that mountain and a long spur off the saddle. Immediately south of it is the Lyell Tharsis mine, practically on the same ore-channel.

Between the two terminal properties, i.e., the Mount Lyell and the North Mount Lyell, no mineralization of special value exists on the eastern fall, low-grade fahlbands, or channels of pyritized schist, only occurring. These are paralleled by similar deposits on the western side of the ridge, with the exception of a small pyritic channel or two, enriched by fahlore to 6 per cent. copper. As a whole, however, the western mineralization is poor, compared with the eastern, the Mount Lyell company only finding profitable sources of mineral there in the South Tharsis and Royal Tharsis ore-channels. These are situated, just below the top of the saddle, about three-fourths of a mile north of the Mount Lyell deposit, and a quarter of a mile south of the North Mount Lyell group. In the depressions between the eastern spurs, especially below the two principal mines, there are more or less considerable deposits of ferruginous clay, carrying low copper, some of which is payable.

The main comb of the divide is formed of the typical Lyell schists for its whole length. On the west they constitute the exclusive rock, and extend until they meet the Silurian strata and the eruptive rocks coming in from the north-west. In contact with the latter their surface width is narrowed down, along the northern half of the saddle, to three-quarters of a mile, and, against the extreme western buttress of Mount Lyell, to three-eighths of a mile. Along the southern half, the schist belt is 1½ miles wide, and more.

On the eastern fall, looking down into Linda valley, the exposures

of Lyell schists only extend about half way down the slope, including the spurs, the eastern ends of which are conglomerate. The apparent contact of the two rocks thus practically runs parallel to the alignment of the saddle.

Linda valley is underlain by the conglomerate, but northward of Mount Owen, for a distance of about one mile, this is covered by glacial deposits, and at the edge of these by alluvial formations. The valley itself is a triangular basin, with a base of about one mile and a length of $1\frac{1}{2}$ miles, the apex corresponding to the narrow outlet into the flat lands of the King river.

In addition to the main body of the conglomerate there are several immense individual blocks of this rock observable, whose isolation from the main body is probably due to fault action. Such a conglomerate crag is situated in the immediate vicinity of the Mount Lyell mine, and a still larger mass, boldly jutting out of the schists, which surround it on all sides, lies west of the North Mount Lyell and Lyell Tharsis ore-occurrences. At this particular spot the schist belt is contracted to a width of only 200 ft., and it is near this narrow area, though not quite in it, that the principal bornite ores of the district are deposited.

As intimated, the economic, permanent mineralization of the Mount Lyell Field is distinctive of the eastern fall of the saddle, and this embraces the occurrences of secondary bornite ores as well the large masses of low-grade pyritic ore. Both are restricted to the immediate vicinity of the contact of the two rocks, i.e., "contact" interpreted in the mechanical, and not in the usual geological sense. One of the interesting peculiarities of the field is the circumstance that the ore-deposition is similar to that which is observable in other parts of the world, where true "contacts" occur, i.e., the junctures of eruptive rocks with sedimentary rocks. In the present instance, the contact, so-called, is that merely of a sedimentary, non-metal-bearing rock and a metamorphic, mineralized rock. Remote from this line of contact there are no ore-deposits to be found, except pyritized channels of schist, or the fahlbands, which are very subordinate in value.

The stratigraphic relation of the orebodies to the country-rock at first is puzzling, for it is apparent that the conglomerate forms the foot-wall rock and the schist the hanging-wall rock, in all deposits. This makes it appear as if the schists were more recent than the conglomerates. Closer examination, however, reveals the presence of a gigantic fault phenomenon. The general rule throughout the adjacent districts is that the schists underlie the conglomerates, and there is evidence of this in the field itself, remote from the mines. In the neighborhood of these, however, there is a tremendous overthrust fault, dislocating the two rocks, traceable the whole way from Mount Owen to Mount Lyell, and beyond, the course

of which happens to coincide with that of the divide, and by which the conglomerates have locally been brought quite under the schists.

This very important, complicated fault-system cannot be discussed in detail. It is only roughly indicated on the geological sketch map (see

FIG. 2—GEOLOGICAL SKETCH MAP OF MOUNT LYELL MINING DISTRICT.
(After Prof. J. H. Gregory.)

Fig. 2). Its continuity is much disturbed by cross-faulting, and it is in the bays and angles thus caused that the ore-depositions have taken place. The deposits all lie wholly within the schists, although the conglomerate locally comes close enough to the Mount Lyell pyritic body

to be quite tangent to it, on the one hand, though there is no actual contact, while, on the other, almost the same thing may be said about the North Mount Lyell deposits. In a few instances in the latter, however, the contiguity is still closer, for occasionally bornite is found practically frozen on to the conglomerate wall, if not actually behind it.

In addition to the regular orebodies, it is geologically correct to state that the contact of the two rocks is marked by a further series of deposits, viz., the hematite masses of more or less size, which follow its line on the surface. These are fairly large and prominent, and were originally regarded as "iron hats," or cappings, particularly as one of them, the "Blow" of the Mount Lyell mine, abutted so closely upon the pyrites as to lead to an apparent confirmation of this well-known phenomenon. As a matter of fact, however, the hematite stood in no connection with the pyrites. Its position (strike and dip) altogether differed from that which an iron hat should have exhibited in relation to the ore, and the fallacy involved became disappointingly evident in the case of the other hematite outcrops, when they were tunnelled into for subjacent pyritic bodies. It was found, in all instances, that the hematite died out into the conglomerate, without a sign of pyrites coming in.

The Mount Lyell Mine.—The Mount Lyell pyrites consists of a very pure, massive, homogeneous, unstratified, cupriferous pyrite, with exceedingly little gangue, principally quartz and heavy-spar, both very evenly distributed throughout the mass, and rarely visible to the unaided eye. It probably differs from the pyrites of most similar deposits by being, not absolutely, but relatively, higher in silver and gold, compared with the copper content. In the same way it is also relatively far richer in precious metals than the bornite ores of the field, or the pyritized schist ores. The ore usually is jointy, but also extremely compact and dense, in places. The texture is finely granular, and visible crystals of iron pyrites are extremely rare; so also are vugs, containing minerals of any kind. The entire orebody is wonderfully free from deleterious elements, containing as it does only about 0.25 per cent. arsenic, less than 0.17 per cent. antimony, no bismuth, and traces only of selenium and tellurium. Galena and zinc blende are now, however, not uncommon in the lowest grades, in certain places in the depths of the open cut the zinc sometimes rising to 1.5 to 3 per cent. in large parcels of ore.

TYPICAL ANALYSES OF MOUNT LYELL ORE.

	SiO_2 Per Cent.	Fe Per Cent.	Al_2O_3 Per Cent.	$BaSO_4$ Per Cent.	S Per Cent.	Pb Per Cent.	Zn Per Cent.	As Per Cent.	Cu Per Cent.	Ag Oz.	Au Oz.
Formerly	4.42	40.30	2.04	2.50	46.50	......	......	......	2.35	2.00	0.07
Now	2.12	42.02	1.97	tr.	48.77	0.65	1.30	0.30	0.73	1.50	0.07
Now	1.40	44.76	0.60	tr.	52.00	0.70	1.50	0.23	0.75	1.12	0.08
Now	3.76	39.88	2.62	1.8	46.60	0.89	2.10	......	0.60	1.74	0.06

A VIEW OF NORTH MT. LYELL MINE FROM THE EAST.

The Mount Lyell orebody (see Fig. 3) is not truly a lenticular mass but rather pipe, or chimney-like, vertically considered, while the horizontal contours are irregularly rounding. The general configuration is that of a much corrugated, tapering, horn-shaped body, gently curving downward and having a cork-screw twist. This mass is obliquely stuck, point first, into the swelling side of the knoll-like spur on which the mine is situated, and which is crossed by the contact of the two rocks. The regularity of its downward trend, however, is modified by a considerable bulge, in depth, on the southern hanging-wall, beginning in the region of No. 6 level. The periphery of the body is wholly surrounded by schist, though this casing is frequently only a few inches thick near the contact, the body at the southern extremity being practically tangent to the contact, or fault line. In consequence of the overthrust, which locally brings the conglomerate under the schist, the conglomerate, as remarked, forms the footwall and the schist the hanging-wall. The upper, highest surface practically came to daylight, the superficial layer of the detrital and gossany matter being very slight in thickness, and the usual covering of humus also very thin, while the spot was free from all but the smallest varieties of scrub, with the pyrites exposed in the surface water rills.

Development and exploration work have established the dimensions shown in the accompanying table. It happened that No. 4 tunnel, on the principal early exploration level, in addition to striking the bonanza, also cut the body in its greatest area.

DIMENSIONS OF MOUNT LYELL OREBODY.

Level.	Greatest Width. ft.	Length. ft.	Strike. deg.
No. 3	280	450	N. 50 deg. W.
No. 4	270	660	N. 43 deg. W.
No. 5	210	510	N. 45 deg. W.
No. 6	270	510	N. 64 deg. W.
No. 8	150	280	N. 88 deg. W.

There is consequently a contraction of area, or "waist," between Nos. 4 and 6 levels, also a helical northward turn of 40 deg., in the major axis, from surface to bottom. The round-off lowest surface, 65 ft. below No. 8 level, corresponds to 690 ft. above the sea. The highest point of the body was 1375 ft. above the sea, thus giving a known vertical depth of pyrites of 685 ft. The general horizontal shape is irregularly oval, with jutting points here and there, and fin-like out-runners of pyrites, interlaced with schist, at the northern end. It is not unlikely that the thinning out of the body at the bottom is only local, and that, in due course, it may open out again beyond the assumed apex. The adjacent South Lyell body, only 300 ft. distant, which does not come to surface, and the downward extent of which has not been established, is possibly a dislocated continua-

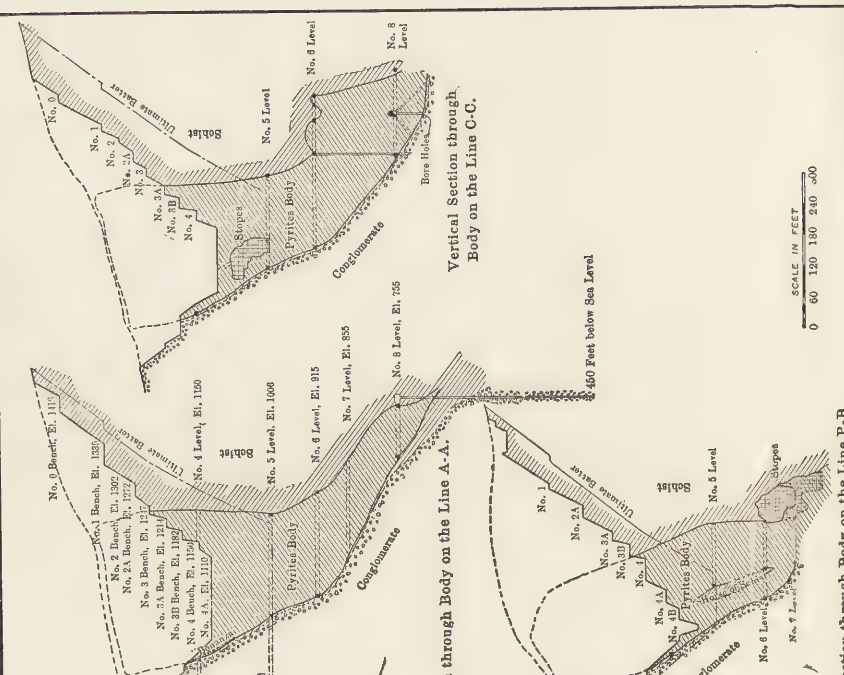


FIG. 3.—SECTIONS THROUGH OREBODY OF MOUNT LYELL MINE.

tion of the Mount Lyell body, but it is more probably a distinct individual occurrence, certain theories as to dislocation by thrust-planes being, so far, only conjectures.

The distribution of the metallic contents was irregular, and rapidly diminished with depth. As a whole, the body is low grade, say 0.6 to 0.75 per cent. copper, plus small amounts of the precious metals. From the beginning this was known to be the value of about one-third of the thickness, counting from the hanging-wall inward, and bounded on one side by the hanging-wall itself, on the other by a diagonal line drawn from the top to the footwall below No. 4 level. This value also applied to the entire northern half of the body to the right of No 4 tunnel. Then again, exclusive of the enriched crusts and shoots on the southern region, it characterized the whole of the body below No. 4 level. The metallic contents of the wedge-shaped piece thus left above and to the side of these regions were marked by an enrichment from the surface down to the foot-wall, a little below No. 4 level, the whole of this wedge-shaped mass having from 2 to 2½ per cent. copper. Over this part the body culminated in a peak, which came to daylight, and from which the surface of the body fell quite rapidly toward the northern extremity, and was buried under the schists.

The entire southern half, from top to below No. 4 level, was, however, in a general way, distinguished, on the foot-wall side, by a crust, or casing, richer than the ordinary pyrites. The increased value was due to copper pyrites, and the thickness of this shell, or crust, varied from 3 to 50 ft., averaging about 20 ft., the copper running from 3 to 8 per cent. in large parcels, and shading off rapidly into the low grade mass of the heart of the body. Since the first exploration work happened to be along this portion of the periphery, an impression was created that the payable part of the orebody averaged about 4½ per cent. copper.

The Mount Lyell orebody possessed further interesting and important enrichments, of a more pronounced character. In the first place, the major portion of the exterior underground surface of the ordinary pyrites is practically everywhere coated with a very thin layer of fahlore, generally not exceeding the thickness of paper. Then the poorer portions, both on the hanging-wall and foot-wall, in various places, but always near the periphery, showed several fairly large fahlore enrichments, following the outline of the body, and running up to 3 to 6 per cent. copper. Considerably more extensive and better areas, however, marked the corners of the southern extremity. Here a continuous, shoot-like, enriched incrustation extended all the way down from the surface to the lowest level, resting, with variable width and thickness, more or less immediately against the pyrites, but in depth occasionally departing from same to the extent of touching the conglomerate. In the higher levels, on the

rounding surface, at shallow depths below the soil, these rich excrescences assayed 35 to 50 per cent. copper, largely owing to the presence of glance. Lower down, below No. 4 level, they consisted of a mixture of bornite, iron pyrites and quartz, with heavy spar, the bornite giving way to chalcopyrite with depth, and finally being wholly replaced by same. This encrusting enrichment nowhere penetrated into the mass of pyrites, but the latter, immediately adjacent to it, throughout only displayed the grade distinctive of its horizon, without a gradual transition between the two. All these enrichments were as free from impurities as the original pyrites itself.

Finally, there was a very important improvement, of extraordinary richness, both in copper and silver, i.e., the unique rich shoot, the discovery of which financially assisted the enterprise at an opportune moment in its early history. It is singular that there has been no repetition of this lucky strike since, notwithstanding diligent search. The enrichment occurred about half-way down the foot-wall of the body, in a surface corrugation, practically in contact with the conglomerate, on the one side, and lying directly against the pyrites, without shading off, on the other. The composition of this bonanza was that of a mixture of bornite, copper glance, and silver glance, with some pyrites and gangue, the value being very variable. As a whole it averaged 21 per cent. copper, 1010 oz. of silver per ton, and 0.1 oz. gold per ton.

The enrichments so far considered are all surface or peripheral phenomena. In the heart of the orebody there was only one similar occurrence. This was a small columnar, or pipe-like zone, enriched beyond the average of the pyrites by chalcopyrite, which was cut on the foot-wall, while driving along the northern half in No. 5 level, and which extended vertically upward from this level to a little above No. 4 level. Neither quartz nor bornite were associated with it, and it ran from 3 to 6 per cent. copper.

The body is traversed by fracture and movement planes in many directions, none of the heads, however, extending into the adjacent rocks. Although on the whole, massive and devoid of layers, the lamination is occasionally quite pronounced. Slickensides are common. An interesting and significant, but exceptional, phenomenon was that on the northern portion of the foot-wall, in No. 3 level, where a decided false, curved bedding of the pyrites displayed itself, coinciding accurately with the foliation of the enclosing schists at that spot.

Needless to say, the low average grade of the pyrites made it necessary to watch variations in value very closely during extraction, and also to expedite the determination of the size, and the localization of the assay values of the body, with all possible despatch. For these purposes the body was riddled with drives, crosscuts and bore-holes, and, making no

allowance for further possible continuations in depth, the total contents, in round figures, ran up to 6,500,000 tons (7.28 millions short tons) for the mass in its original size. Of this quantity about 2,750,000 tons have been extracted and smelted, leaving about 3,750,000 tons in reserve. Out of this tonnage roughly 1,050,000 tons are available for open-cutting down to No. 5 level, and the remainder of 2,700,000 tons remains for underground mining.

The smelting results exhibit the flow of values clearly. In connection with them it must be remarked that no selection was possible, except in the earlier days, when it was considered necessary to avoid the poor hanging-wall and the northern half of the body, both of which it was then thought would probably never be treated.

DECLINE IN GRADE OF ORE.

During treatment of	Decline in copper from	With an average of		
		Copper. Per Cent.	Silver. Oz.	Gold. Oz.
323,000 tons dry.....	5 to 3¼ per cent.	4.0	4.17	0.159
432,000 tons dry.....	3¼ to 2½ per cent.	2.6	2.45	0.086
711,000 tons dry.....	2.4 to 2.2 per cent.	2.35	2.12	0.071
1,466,000 tons dry		2.85	2.67	0.095

Then followed the amalgamation of the two companies and excavation on wider lines, the grade decreasing, within 500,000 tons, to the vicinity of copper 1.0 per cent., silver 2 oz., gold 0.07 oz., with the deepening of the open-cut and its encroachment on the hanging-wall and the northern half. Since the beginning of 1905 the value has ranged at about copper 0.9 per cent., silver 1.8 oz., gold 0.07 oz., with temporary improvements to about 1.0 per cent., owing to the excavation of regions close to the stoped-out enrichment crusts, etc. The prospective value, however, is likely to be, permanently, from 0.75 per cent. down to a fixity of 0.6 per cent. copper, together with silver 1.92 oz. and gold 0.044 oz., the precious metals evidently holding their own stronger than the copper. The total money-value of this low-grade ore is such that it remains distinctly payable, for a fairly low copper quotation, with the present treatment practice. Needless to say, one explanation of this is that the amalgamation of the two companies has considerably cheapened treatment costs, the North Mount Lyell ore, besides supplying copper, also replacing barren silica flux formerly used, etc.

Method of Mining.—The orebody was very favorably situated for the open-cast work, the apex being exposed for a considerable length of the strike, and the outcrop traceable for 300 ft. wide, up and down the hillside. After the first breach was made the excavation of the orebody

down to a suitable depth, however, has required the simultaneous removal of a considerable amount of the overlying rock, to free the pyrites in depth. The excavation is done in benches, in a funnel-shaped cavity, with pre-established ultimate side and bottom limits to work to. These were originally decided upon by fixing the size of the bottom of the open-cut as the contour of the body at No. 4 level. Recently, however, the extent of the excavation has been enlarged to embrace the area of No. 5 level, as the ultimate lowest plane. In each case the batter-lines were inclined outward from the respective contours at the steepest angles guaranteeing the safe standing of the walls of the excavation, and running from ½ (horizontal) in 1 (vertical) to 1 in 1. The amount of overburden to be removed is, fortunately, much curtailed by the external slope of the hill.

The open-cut bench levels correspond to the old underground levels, with interpolated floors, so that, reckoned from the top of the knoll, there are now 12 overburden and ore benches, averaging about 30 ft. in hight. The vertical hight from the edge of the highest bench down to the present lowest is 360 ft., corresponding to a full vertical depth of open-cut of 457 ft. down to No. 5 level. The size of the excavation, measured at the upper margins, is now 900 ft. by 700 ft.

The bench system gives the workings an amphitheatrical appearance, the terrace being crescent-shaped, owing to the hillside location. This position obviated the necessity of a spiral or helical arrangement of the benches, for the material won on each bench is readily removed outward on level lines. Until a few years ago all spoil was tipped upon the valley flat-land to the south and east of the knoll, but at present the overburden is almost entirely run northward across a narrow gully facing the mine, and deposited on steeply sidling ground. The leads are not conveniently long, and neither steam nor electric haulage, nor steam shovels, are applicable. Each bench is operated by itself. If in ore, the product is constantly sampled in the face, as well as in the trucks, for immediate information, and the daily grade averaged from the results, from which, again, the set monthly grade is controlled and maintained, the final value assigned to the mine deliveries, however, being that obtained by the systematic sampling at the reduction works. The explosives used are gelignite in the ore, and black powder and cheddite in the rock, the total explosives consumed annually being 60 to 70 tons. Electric blasting is also used. Blasting is done twice a day, regularly, at the noon hour and at the close of the day shift, but occasionally later also, though the work is only carried on in the daylight hours. The ordinary percussion rock-drills, hand hammer-drills, etc., are employed. The handling of ore is effected in one-cubic-yard, side-tipping trucks, 2-ft. gage, run on movable tracks, filled at the face, and braked in long trains on slight gradients, through headings in the footwall, on to discharging platforms bridging

two self-acting, 2-ft. gage rope-inclines. The latter run down the slope of the knoll to storage bins situated in the gully, just below the mouth of No. 4 tunnel, which forms the upper terminus of ropeway and haulage line to the furnaces. Counter-balanced, 5-ton, self-discharging trucks on the inclines limit the attendant cost of lowering to a small fraction of a penny per ton. The overburden is removed in ordinary contractors' style, by means of 3 ft., 6 in. gage, 3.5 cu.yd. capacity, fiddlestick wagons, loaded through headings driven into the face, and dumped head first over a cradle on edge of tip, at the end of a short steeper run. Overburden broken below No. 4 level is thrown down a pass into trucks in No. 5 tunnel, the lowest adit, and trammed out to the valley flat. Pyrites below No. 4 level is raised to haulage-line bin-level by means of an inclined surface hoist, installed on the foot-wall side of the open cut, and electrically operated from the reduction works.

The daily rate of excavation is from 850 to 1200 tons of pyrites, and from 375 to 1150 cu.yd. of overburden. The work is not given in contract at the Mount Lyell mine, the wages system being found preferable. The ordinary shift is eight hours, from 8 a.m. to 4.30 p.m., 30 minutes allowed for crib (lunch) being counted out of the working time. At least one-quarter of the entire time, however, is lost, on account of tempestuous, rainy weather, and the whole work must be put down as having an unique character, quite different from surface operations in other climates, for ordinary all-day showers do not interfere with operations. On the other hand, there is but very seldom any snow to stop them. The average rate of the wages, embracing all classes of labor, is 8s. 9d. per shift, the lowest, or "rouse-about" rate being 8s., and the highest being 10s. 6d. for drillsmen.

The number of tons of pyrites broken per man, counting all hands engaged at the Mount Lyell mine, including supervision, is from 800 to not quite 1000 tons per annum, i.e., for 365 days. More specifically, the output is 9.25 tons per man per shift for all men at the face, 10.5 tons for the shift for each man drilling and shooting, and 17.5 tons per man per shift on the hands engaged in spalling, picking, and shoveling. The ground is all hard, of course.

The original quantities available for open-cutting, not counting the wedge-shaped pieces of pyrites below the respective horizontal planes, contained within the batter lines, down to No. 4 level, were 693,000 cu.yd. of pyrites together with 1,500,000 cu.yd. of overburden; or, down to the flat of No. 5 level, 823,000 cu.yd. of pyrites and 2,000,000 cu.yd. of overburden. Each cubic yard of overburden liberates about $1\frac{1}{2}$ tons of pyrites. The cost of excavation of the pyrites has averaged 2s. 6d. per ton over the whole time since the beginning, but of late years has been only 1s. 6d. per ton. The average cost of shifting a yard of overburden

similarly has been 3s. 1d. since the beginning, but latterly 2s. 9d. Taking into account the relative quantities of each, a levy of 2s. for removal of overburden (including certain unrelated, subordinate surface operations) was placed on every ton of pyrites under the old scheme, down to No. 4 level. As this brought the account into a fairly heavy credit, in course of time, through the cheapening of the work, the charge was brought down to 1s. a ton toward the end. The initiation of the wider scheme down to No. 5 level, however, has caused a return to the 2s. charge, which will again leave an ample margin. The capacity per man per shift in overburden is 4.5 cu.yd. for all hands at the faces, or 4.75 cu.yd. for drillsmen, powder-monkeys and helpers, and 8 cu.yd. on the crew spalling, picking and shoveling.

Underground the Mount Lyell body was opened up in eight levels. The tunnels and drives of the upper three levels are now wholly cut out by the progress of the open-cut excavation. No. 4 tunnel, at an elevation of 1150 ft., was run through the foot-wall rock for 320 ft., and still remains the principal working level, for reasons of convenience connected with the transportation of the ore to the reduction works. No. 5 level, elevation 1008 ft., and opened up by No. 5 tunnel, 1450 ft. long to the pyrites, is the lowest adit level possible and drains the mine, the lower levels being unwatered by pumping. The density of the pyrites and the adjoining rocks guarantees comparative freedom from water, so that the underground workings were never "wet". In nearly all of the levels the body was fully contoured. Frequent winzes and prospecting shafts connect the levels, sunk on enrichments on the periphery. A blind vertical shaft, with an air-driven winding engine, was sunk in the hanging-wall, 260 ft. for access to the workings below No. 5. The ore won out of the stopes in the southern enrichments, raised in this shaft, was trammed to a subsidiary shaft on the line of the No. 5 adit tunnel, from which it was hoisted to the yard level of the haulage line, at the bottom of the storage-bins in front of No. 4 tunnel mouth.

The underground extraction of the richer ores was conducted by the usual overhead stoping methods. The local practice is to push up rises in advance of the floors, and, if possible, thus to provide passes for filling from the surface. Where this was not possible, square-set timbering was resorted to, using round local timber (eucalyptus), not, however, for the purpose of supporting the ground, which would have stood in any size of opening, but as a scaffolding to work from. Waste from neighboring exploratory workings was stowed between the timbers, where available. With the deepening of the open cut the latter were subsequently withdrawn here and there, the cavity then being filled with mullock from the open cut.

South Mount Lyell Mine.—The South Mount Lyell pyritic body, which lies to the southwest of the Mount Lyell body, is similar to it in character,

VIEW OF SMELTER SIDE OF HAULAGE LINE.

but very much smaller. It was discovered by the sinking of a shaft, near the common boundary line, by the former owners, in the hope of striking the larger deposit. At 524 ft. of depth pyrites was encountered, but left again 124 ft. deeper. At this point it is 35 ft. lower than the No. 8 level of the Mount Lyell mine, so that the lower surfaces of both bodies attained at present are in practically the same horizon. As already mentioned, the shortest distance between the two deposits is about 300 ft. The workings of both are now connected. The South Mount Lyell body has been investigated by contour drives and stoping in one level, and found to be about 160 ft. long, by 80 ft. wide. Its dip is steeply to the SW, and the strike of the major axis NW by SE, the extent in depth being unknown, while its upward extension can not be much higher than the point of entry of the shaft into the pyrites. The body has not so far been of economical interest, as it appears devoid of enrichments, and has an assay of copper 0.40 per cent., silver 0.25 oz. per ton, and gold 0.02 oz. per ton.

North Mount Lyell Mine.—Though the area of its mineral leases only comprises 30 acres of ground, this mine is at present the richest property in the field, and the largest copper producer. While the consumption of Mount Lyell pyrites at the reduction works is two to three times as great as that of the North Mount Lyell ore, yet the value of the latter in copper is eight times that of the former. It is, however, rather lower in gold and silver.

The ore typical of the bornite deposits of the field consists of a gray, or whitish schist, in which iron or copper pyrites are subordinate, and bornite is prominent. It is made up very largely of "eyes," or small lenticular masses and flat nodules, of denser, silicious material, enclosed in the soft, fissile laminae of the schist, the nodules measuring from a few inches to feet. The copper mineralization is irregularly deposited in the laminae of the surrounding schist, in grains or shots of small size, or in stringers, etc., or in larger blotches, but it also occurs frequently in particles down to microscopic fineness, in the silicious matrix itself. A peculiarity is that the ore breaks in such a manner that the pieces at first sight do not externally reveal their mineral contents, but appear like valueless lumps of soapy schist. When broken transversely, however, they at once show the mineral sprinkling, sometime revealing astonishing richness. The North Mount Lyell ore is of this character, but richer than the ordinary run. As the presence of bornite seems to stand in some relation to that of uncombined silica, it is also relatively more silicious. Bornite (and glance) thus associated with quartz predominates to such an extent that the ore generally has a more blocky or lumpy, less schisty appearance, although there is often no difference. The occurrence of solid, or truly massive bornite is, however, comparatively rare, even the densest looking faces of this kind really being fairly silicious, except for sporadic

bunches of the pure material, running up to 45 and 50 per cent. in copper. The ordinary richer varieties of the ore (12 to 20 per cent. copper) consist of a dark gray, quartzitic mass, in which bornite and glance appear in thin veinlets, grains or splashes. The medium-grades (6 to 10 per cent. copper) show a similar, but lighter gray quartz-matrix with more schist, and with the same minerals, but less glance. Lower grades (3 to 6 per cent. copper) are as above described, having still less bornite and no glance, or else derive their copper chiefly from chalcopyrite. There are also larger masses of firm, flintlike quartzite, carrying an impregnation of bornite of extreme fineness (3 per cent. copper). The characteristic, predominating mineral is practically always bornite. Accessory non-cupriferous minerals are scarce. The rarer copper (or silver) species cannot be recognized by the eye. Barytes is insignificant in amount, and sulphides of lead, zinc, etc., are not found. Native gold is an interesting occasional constituent, occurring in places in association with white cellular quartz, usually accompanied by bornite, and even growing directly upon same. Enargite and fahlore are likewise scarce. Typical compositions are given in the accompanying table, the figures representing large averages only, as the different varieties of the ore are thrown together in the smelting lots, or parcels.

COMPOSITION OF NORTH MOUNT LYELL ORE.

	SiO_2 Per Cent.	Fe Per Cent.	Al_2O_3 Per Cent.	Cu Per Cent.	$BaSO_4$ Per Cent.	Ag. Oz.	Au. Oz.
Older....	63.70	6.22	10.10	7.42	trace	3.08	0.01
Recent..	66.60	6.80	7.50	6.47	1.00	1.80	trace
Present..	63.92	8.40	11.30	5.57	0.24	1.21	trace

The great overthrust fault runs southward through the property and is locally complicated by an extreme jumble of subordinate disturbances, much more pronounced here than at any other point along its line (see Fig. 4). The locality constitutes an area of disturbance caused by a sudden deviation in strike of the main fault-line from its NS alignment to an EW one. The latter has its course along the apparently non-mineralized southern flank of Mount Lyell (although this portion of the contact has not yet been examined). The continuation of the fault-line further north, along the western buttress of Mount Lyell, has been investigated to some extent, and shows a bornite mineralization. Within the mineralized area covered by the North Mount Lyell leases, the dip of the schist and conglomerate to the SW, which they show conformably to each other, suddenly changes to the SE, beginning at a depth of about 500 ft. It is significant that at this point the very hopeful mineralization constituted by the so-called "New Development Orebody" referred to below, and

its accessory orebodies, takes its beginning. Along the fault-line the conglomerate rock exhibits a textural change, by showing less of the well-bedded pudding-stone character, and becoming more dense and compact, in fact, turning quite into an iron-stained quartzite, with little

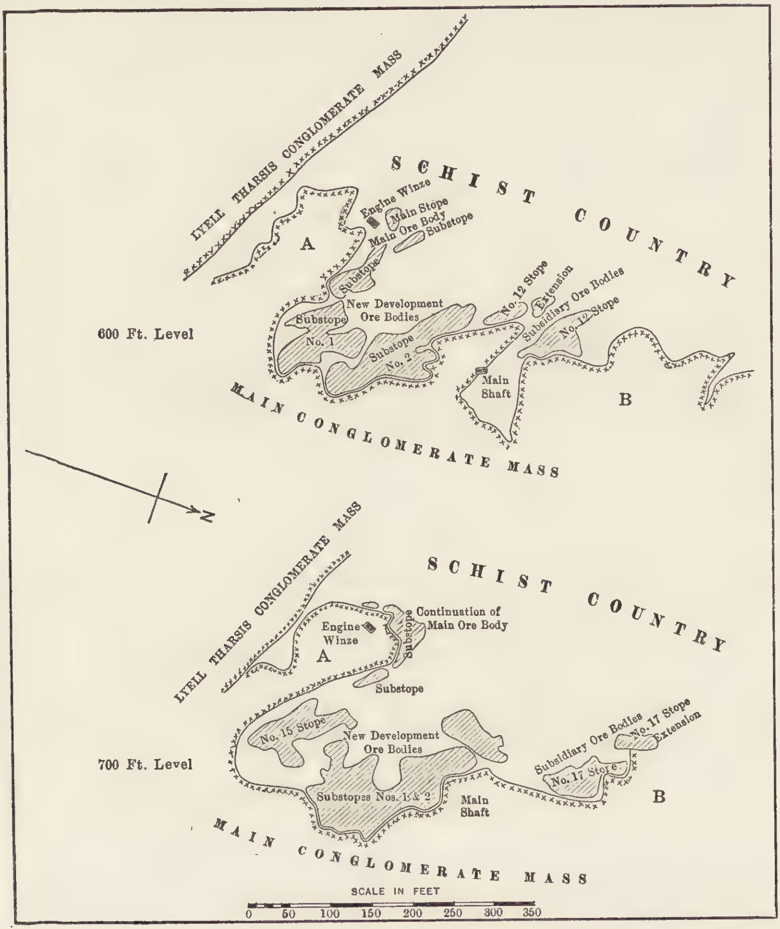


FIG. 4—PLAN SHOWING MAIN LINE OF CONTACT AND ORE OCCURRENCES, NORTH MT. LYELL MINE, AT 600- AND 700-FT. LEVELS.

or no signs of bedding. A similar transition also takes place, alongside of the conglomerate, in the schist rock itself, this being locally replaced by a quartzite sometimes difficult to distinguish from the other, though this modification is less frequently noticeable than in the case of the conglomerate. The actual contact is generally marked by a more or less pronounced occurrence of hematite in seams.

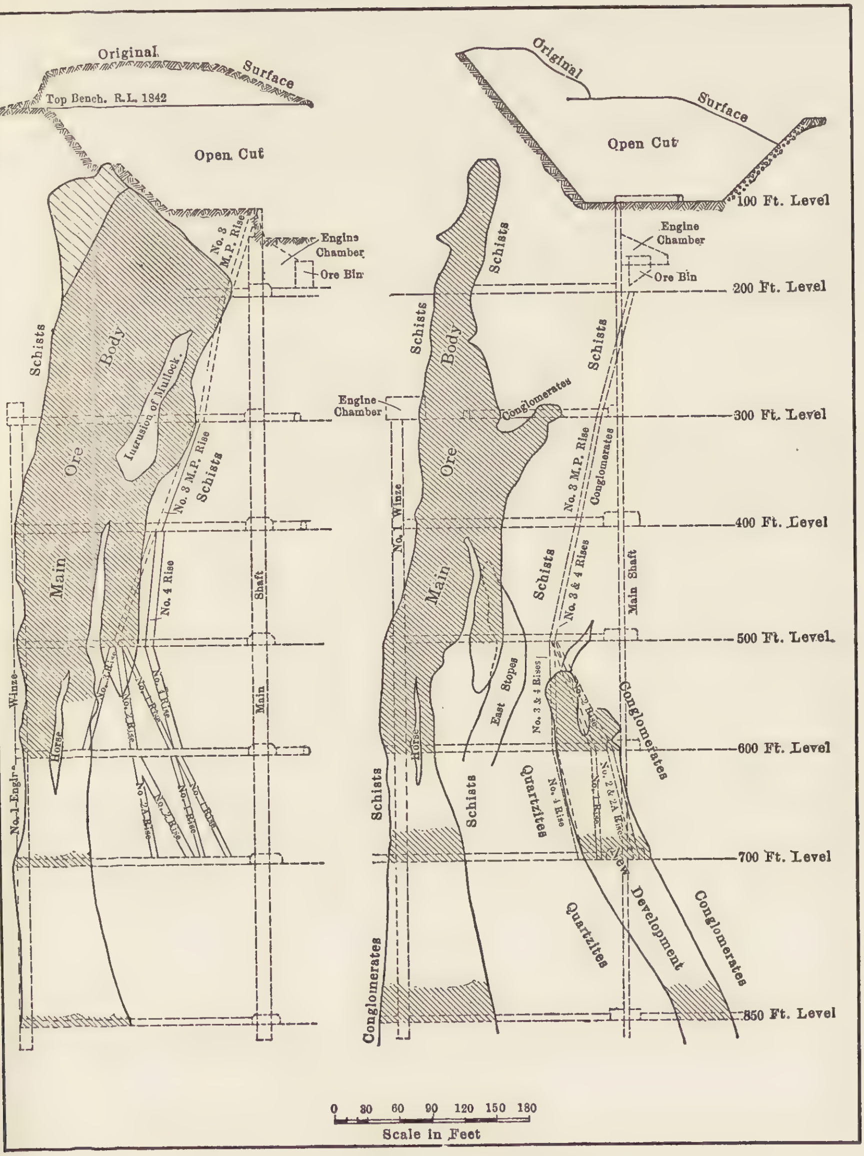
The designation of the ore-courses of this mineralogical type as "masses,"

or "bodies," takes certain liberties with the proper meaning of these terms. The conception of what, in local application, constitutes a "body" is, in fact, purely economic, i.e., defined by payableness, and the bodies are not true "masses," like the Mount Lyell deposit. Structurally, the entire ore-occurrence is merely a pronounced example of the dissemination of secondary ores throughout a much disturbed region of schist rock, greatly silicified, but otherwise not differing essentially in petrographic character, or departing very much tectonically, from the general run of the schist in contact with the quartzites belonging to the lower region of the conglomerate. The distinction between this part of the field and all the rest lies chiefly in the violence of the crushing to which the schist rock has been subjected. The copper minerals are deposited within the foliation planes of the schist in a confused, irregular manner, in the way described above. The planes are occasionally wholly obscured by blocks, of more or less size, of solid, quartzitic material, which is almost flintlike or jasperoid, in nature. At the same time, even these cores, as intimated, often carry payable dissemination, or "impregnations." There are no observable surfaces of demarcation between the ore-bearing rock and the non-mineralized country, comparable to true walls. As far as the erasure of all persistent, continuous structural lines admits, the foliation of the rock of the barren country, and of the orebodies, is identical. As there is some misconception about the nature of the deposits, it may also be stated that there is no saddlereef-like configuration of any orebody and the containing country, nor is the foliation of the country coincident with the contour of the ore-bearing nuclei, in the sense of curving around external bays and angles of the latter, such as in the corners, or re-entrant angles, of apparent "legs," or prongs, which the mineralized mass seems to throw out. The foliation is consistent throughout, and cuts through main bodies and legs, or prongs, alike. Locally, horses of poor ore, or of barren quartzitic rock, have been left standing within the stoped-out regions, on account of unpayableness, but the rock mass of the ore was the same as that of the horses. In other places the body, as mined, seems to have thrown out a so-called "leg," but this is only due to an analogous cause, i.e., the lateral intrusion of barren regions.

There appear to be over a dozen distinct orebodies, of more or less value, all of which are being worked. The oldest, and principal one of them, the so-called original Main orebody, exceeded the others above the 500 ft. level very much in size, but its lateral underground offshoot, the New Development orebody, more than rivals it in importance.

The Main orebody (see Fig. 5) came to surface at 1750 ft. above sea level. It had an extremely irregular shape, marked by deep corrugations and indentations, but on the whole its structure was pipe-like, or columnar. In fact, in its most continuous dip-line, on the south, it presented a

vertical edge. This edge, situated in the southwest corner of the Main orebody, was marked by a singular feature, consisting of a remarkably



FIGS. 5 AND 6.—VERTICAL SECTIONS THROUGH MAIN OREBODY IN N. N. E. AND E. N. E. DIRECTIONS, NORTH MT. LYELL MINE, SHOWING NEW DEVELOPMENT OREBODY.

rich shoot of ore, which extended uninterruptedly practically all the way down from the surface to the 600 ft. level, where it became dispersed.

Below that level, however, similar ore again comes in strong, with undiminished richness (see Fig. 6). The whole body may be said to have an irregular, much indented carrot-shape, down to the 600 ft. level, and widens out again below the 200 ft. level, where it is measured 200x100 ft. Other smaller bodies in the higher levels, more or less connected with it, seem to branch off and come to individual existence. At about the 400 ft. level the New Development orebody already mentioned, a second extensive body, but not so straight, and not vertical, but underlying to the S.E., joins it, by means of two intermediate bodies, acting as connecting links (East stope). The New Development body itself exceeds the Main body in horizontal area, at the 700 ft. level. It has again been met with in the 850 ft. level, where it is now explored to a size of 250 ft.x60 ft., without full delimitation, and more recently it has also been struck in the 1000 ft. level. In the 850 ft. level a further orebody (No. 20 stope) has been met with between the New Development body and the downward extension of the Main orebody, which is not only one of the largest bodies in the mine but also one of the best in grade. An interesting fact is that while the bornite holds its own in these depths, the accession of glance seems to increase.

The other Mount Lyell deposits, so far found, lie to the north of the above, and are irregularly dispersed, with varying strikes and dips, along the sinuous line of contact. None of them comes to the surface. The original ones do not go below the 400 ft. or 500 ft. level. There have, however, recently been some very important, promising accessions, from similar masses, in the 700 ft. level, starting in so healthy a manner that there is little doubt that they will also attain a fair size with depth. The horizontal dimensions of these subsidiary orebodies vary from 25x30 ft. to 150x50 ft. In part they consist merely of chalcopyrite impregnations in quartzitic schist, but most of them have the characteristics of the ordinary North Mount Lyell ore, i.e., the bornite predominates.

In addition to these directly payable orebodies, there is a huge zone of more or less mineralized quartzitic material, below the 500 ft. level, between the two shafts, which occasionally shows a heavy impregnation of pyrite and chalcopyrite, with some bornite.

Method of Mining.—In the case of the Main orebody the surface portion is being excavated by an open-cut, measuring 400x200 ft. on the upper rim, the bottom coinciding with the 100 ft. level. This open-cut has supplied some extremely high grade bornite ore, which was dispersed around poorer nuclei, or lenses, of dense quartzite.

For the underground workings there are two adit tunnels. The first, the 200 ft. tunnel, elevation 1561 ft., and 850 ft. long, was driven by the old company for the exploration of the neighborhood of the Eastern orebody, some 500 ft. to the north of the Main orebody. On the discovery

GENERAL VIEW OF REDUCTION WORKS OF MOUNT LYELL COMPANY FROM THE WEST.

of the latter on the surface, as above related, driving was deflected in its direction, which explains the deviousness of some of the workings. Simultaneously the 300 ft. tunnel, elevation 1450 ft., 780 ft. long, was driven straight for this orebody. It is the drainage level of the mine, no deeper one being possible, owing to topographical difficulties. The old company had started a prospecting shaft, or winze, from the 300 ft. level, on the nut of the Main orebody. This was enlarged and continued on taking over the mine, and has now reached a depth of 1000 ft. below surface. In addition, a more centrally situated sink, the Main shaft, was taken in hand from the 200 ft. level, and located at the opposite, i.e., the northern end of the main workings. It is also down to the 1000 ft. level, and is kept in advance of the other. For the development of the larger bodies these are first crosscutted for, off the shaft, and then main drives, or galleries, are driven into the heart of the deposits, following the estimated line of strike, and the ground floor is at once beaten out as fast as possible, to establish the area of ore. Connection with the upper levels is made chiefly by means of rising, large permanent passes, built up of heavy, key-shaped timbers, placed in a circle, and presenting the endwood to the wear, being put in for the passage of filling, ventilation, etc. In addition to the exploitation work at the shafts the general policy is to pursue closely the contact of the two rocks, both to the north and south, in search of further orebodies, and to explore both walls, but mainly the schist, either by driving or diamond-drilling. At present ore is being broken in all levels, from the 200 ft. level down to the 850 ft. level, though the stopes on the Main orebody proper are now depleted, filled, and finally dealt with. The 1000 ft. level is just being opened up and promises extremely well.

The system of rill stoping is in use on the larger masses, the filling, which is won in the open cut, being let down through nearly vertical passes from the surface. These main passes are divided into steep branches below, which lead the mullock to place without handling. Depending upon the safety of the ground, the filling is either carried up close to the roof, or "back," allowing only sufficient space for elbow room, or else larger cavities are opened in better standing ground. On the ground floors of the levels a single square-set hight of timbers is erected, on the usual flat timber-foundation, for catching up the level from below. The sets are not carried higher, except for special reasons, and mullock is stowed between them on the ground floor, leaving only the permanent thoroughfares open. The support of the roof above the filling, where necessary, is done by the use of crib-work, or "pig-styes," open or filled, resting on the filling, a method which, in various modifications, has been brought to greater perfection in Australia. On the whole, the mining is somewhat difficult, owing to the irregularly jointy ground, especially in the higher

levels, but it is done with cleverness and ingenuity. Full square-setting is only used in subsidiary orebodies, to which filling cannot be passed, and in which the pressure of the ground is slight. The mining timber is of local origin, principally gum (eucalyptus).

Both shafts are served by electrically driven winding engines, drawing their power from the reduction works generating plant, with a former air-driven hoist retained as a standby, at the Main shaft. The latter will, however, shortly receive an Ilgner hoisting plant, the first to be installed in Australasia. Underground storage bins at the brace discharge into long trains of side-tipping trucks, which are gravitated out of the tunnel, and pulled, a little over one mile, on a dead level, by a Krauss locomotive, to the haulage line bins, situated on top of the divide. From these the ore is let down to the reduction works by the general haulage system. As stated, the drainage of the North Mount Lyell mine takes place through the 300 ft. level. The mine is not especially wet. The water, as yet only of surface origin, however, is very acid, more so than that of the Mount Lyell mine, owing to looser ore-ground.

At both mines cement copper is won by precipitation with scrap-iron in an elaborate system of launders, or boxes, with recoveries of 80 per cent. on an average. The North Mount Lyell mine water carries 50 grains of copper, normally, to the imperial gallon.

Nearly all of the underground mining work is done by contract, not alone driving and sinking, but also stoping, tool-sharpening, trucking, etc. The contracts for stoping are let to small parties, who supply their own explosives, etc., and who are paid by the ton delivered to the bin, the dry weight of the smelter returns forming the basis of payment. The wages of all ranks at the mine average 10s. 9d. per eight hour shift (inclusive of "crib-time" here) the rates running from 9s. 9d. a shift to 12s. 3d. for first-class miners. As usual, changing takes place, not before the face, but at the tunnel mouth. The usual percussion air-drills, etc., are used; the chief explosive is gelignite. The amount of ore broken per man at the North Mount Lyell mine, counting in all hands, is one ton per day, figured on the ore, equivalent to 2.5 tons per miner actually engaged in stoping, on contract (inclusive of mullocking, timbering, etc.). Most of the ground is extremely hard, and driving and sinking are correspondingly retarded.

The Eastern orebody, referred to above, consisted of an interesting detrital jumble of iron pyrites and quartz boulders, of all sizes, more or less mineralized by chalcopyrite and some bornite, occurring in a mass of bituminous, sandy clay, which latter was in itself occasionally rich in copper, generally richer than the fragmental material it enclosed. Boulders of bornite also were found. This clastic deposit stood in some relation to a small pyritic body, evidently still *in situ*, but did

not live down quite to the 200 ft. level. It was depleted by an open cut 300 ft. long by 150 ft. wide, and 90 ft. deep. Its situation was between conglomerate on both sides, the northern wall being the flank of Mount Lyell, and the southern that of a narrow tongue of conglomerate rock. On the other side of this tongue a similar, but smaller-sized, and less highly decomposed body occurred, which extended nearly to the 300 ft. level.

With respect to the ore reserves available in the North Mount Lyell mine, the following may be said: At the time when the amalgamation of the two companies was broached (February, 1903), the reserves were estimated to be 250,000 tons of 7 per cent. copper ore, chiefly at the hand of the main orebody. Of this amount 25,000 tons were extracted during negotiations. Prior to the date mentioned, the North Mount Lyell company had mined and sold, or smelted, about 142,000 tons of 12 per cent. ore, including 82,000 tons of 6 per cent. ore sold to the Mount Lyell company on contract. Since amalgamation, the further reserves added have permitted of the treatment of 450,000 tons up to Oct. 1, 1907, so that the total tonnage established in the various known bodies, from the surface down to the present lowest level, now exceeds a million tons, averaging between 6 and 7 per cent. copper, and this quantity is being increased by further development.

Lyell Tharsis Mine.—This is the southern prolongation of the outrunners of the larger North Mount Lyell occurrence. It contained a mineralized schist belt, or ore-course, about 250 ft. long by 40 ft. wide, tapering downward, and pinching out about 150 ft. from the surface, where it practically disappears for the present. The schist was of the decomposed type, with nodular inclusions of quartzite, and a mineralization in the foliation planes, i.e., studded with bornite and chalcopyrite, as described above. This belt is apparently obliquely cut off in dip by the ordinary green, barren schist, occasionally strongly ferruginous. There are several old working levels, now connected up with the North Mount Lyell drives coming along on the line of contact of the two rocks. An open cut, now exhausted, supplied a fair amount of good-grade smelting ore, and was 300 ft. long by 100 ft. wide, by 80 ft. deep. On the west of this bornitic schist belt is a pyritized belt of common schist, carrying stringers of chalcopyrite, which forms an important asset for future treatment.

ANALYSES OF THE LYELL THARSIS ORE AND SCHIST.

	SiO_2 Per Cent.	Fe Per Cent.	Al_2O_3 Per Cent.	Cu Per Cent.	Ag Oz.	Au Oz.
Ore.......	64.00	5.08	17.25	4.02	0.68	0.00
Schist.....	59.70	13.60	11.90	1.52	0.30	trace

South Tharsis and Royal Tharsis Mines.—These mines do not present

any special features. They are contiguous to each other, on the same pyritized schist channel. They were opened up by means of a couple of side-hill tunnels, by the original owners. Exploration work was extended by the Mount Lyell company, establishing the existence of two mineralized cores, about 200 to 250 ft. by 80 to 120 ft. in horizontal area. The South Tharsis body has been worked by an open-cut, as far as surface extraction could be profitably carried, the size of the opencast being 200x100 ft., by 60 ft. deep. The Royal Tharsis deposit, which is a little higher in assay-value, has so far called for little work. Underground extraction, however, is contemplated, and for this purpose an inclined shaft, made accessible from the North Mount Lyell ore-tram, has been sunk. Like all the rest of the orebodies of the district, of whatever nature, except the Lyell Tharsis, these two deposits have their greatest dimension downward. Diamond-drill boring has proved the existence of pyritized schist, of the same value, to a depth of from 400 to 500 ft. below the surface.

SOUTH THARSIS AND ROYAL THARSIS ORE.

	SiO_2 Per Cent.	Fe Per Cent.	Al_2O_3 Per Cent.	Cu Per Cent.	Ag Oz.	Au Oz.
South Tharsis....	62.67	8.05	12.17	1.67	0.24	0.019
Royal Tharsis....	61.17	9.94	10.44	2.04	0.22	0.021

In addition to the above, the company also owns several less important mineral deposits, not calling for special mention, mostly of the fahlband type. One of them, the North Crown Lyell, situated about two miles to the north of the Mount Lyell mine, and immediately at the foot of the western end of Mount Lyell, is traversed by the contact of the two rocks, which is here accompanied by a small amount of bornite.

The Central Lyell section is adjacent to the Mount Lyell mine, but barren, as far as known, except for the continuation of the Mount Lyell pyritic body across the common boundary line. This extension was established in the very early days by the Central Lyell company, by means of three vertical diamond-drill holes, 650 to 800 ft. deep. Shaft-sinking, however, was out of the question, the pyrites being too low in value.

The King Lyell mine lies at the foot of the Mount Lyell mine, in a gully at the base of the knoll, and is undoubtedly a detrital deposit derived from the pyritic body. It is varicolored clay, containing small quantities of native copper and some glance, heretofore not worth working. It has been explored by means of shaft and diamond-drill to a depth of 285 ft. In addition to the material mentioned, strata of carbonate of iron and limestone were perforated.

An analogous deposit, situated below the North Mount Lyell mine,

is the Mount Lyell Blocks clay-deposit. The clay occurs in thick tabular form, of almost vertical dip, abutting against the southern flank of Mount Lyell, and measuring about 1000 ft. in length, by 30 to 50 ft., or more, of payable width. Its strike is a little west of north, and the dip is inclined slightly to the SW. Its existence has been established by shaft for over 500 ft. from the surface. Although naturally dry, it forms exceedingly difficult mining ground to keep open, when broken into. As the value is low, on the whole, it is being worked from tunnels at the lower end by a precarious system of mining, managed, however, with local skill and ingenuity. The copper contents of the payable portions are 1 per cent. and less. There is an abundance of limonite concentrations in places, with which the native copper is also associated. In part the deposit is thickly fossiliferous, showing the same brachiopod casts as the western quartzite belts. The treatment consists in puddling and blanketing, the latter similar to well-known rudimentary dressing methods for gold, vanning having, in this instance, yielded inferior results. The result is a non-argentiferous copper product, running up to from 70 to 80 per cent. copper, which is shipped to Europe. A lower product of 20 to 40 per cent. copper carries about 20 oz. of silver per ton, and is sold to the Mount Lyell company.

Machinery.—Motive power at the company's mines is supplied in part from the reduction works generating station, by means of electric transmission, or is derived from steam on the spot. For the latter purpose there is a fine, wholly Australian-built compressor-plant at the Mount Lyell mine, consisting of four multitubular boilers and appurtenances, and two high-class, compound duplex compressors, Corliss type, with mechanically actuated air-valves, and surface condensers and intercoolers. They supply a total of 2660 cu.ft. of free air per minute, at 90 lb. pressure. The compressed air is at present conducted to the North Mount Lyell mine from the same source, by means of 3300 ft. of 6 in. tubing and 2700 ft. of 5 in. tubing, the smaller underground winches and pumps being driven by this power, in addition to the drills, etc. The compressed air supply at this property will, however, shortly be derived from an individual plant driven by electricity, and the current will also be used for pumping, etc.

Hoisting of ore in the Mount Lyell open-cut, from below the No. 4 level, is performed by means of an inclined, self-dumping skip-hoist, served by an Australian-made 200 h.p., motor-driven winding engine, supplied by a direct cable line from the reduction works, $1\frac{1}{2}$ miles away. A second cable line, $2\frac{1}{4}$ miles long, similarly transmits 500 h.p. to the North Mount Lyell mine, for winding, etc. The current used on these lines is three-phase, alternating, of 3000 volts, stepped up from, and down to, 550 volts, with a periodicity of 50 cycles per second. Other power appliances of subordinate character have already been referred to. The

Ilgner hoist to be erected at the North Mount Lyell mine in 1908 is for a gross load of four tons, to be lifted from a depth of 1500 ft., at a maximum speed of 1500 ft. per minute. The high tension current will be applied direct to the flywheel converter set.

A well-equipped machine- and fitting-shop is situated at the Mount Lyell mine, for current repairs. All mine workings are lighted by electricity and a complete underground and surface telephone system is installed. The mine office and laboratory, casualty ward, etc., are situated at the foot of the Mount Lyell knoll, close to Gormanston. The water-supply is piped $1\frac{1}{4}$ miles from a creek on the western face of Mount Owen, 1750 ft. high, traversing a valley 400 ft. deep on the way, to a high pressure reservoir above the Mount Lyell open-cut, of 500,000 imperial gallons capacity, and in addition there is a further storage, for boilers, condensers, etc., of 900,000 imperial gallons, at the compressor plant.

The Reduction Works

Site.—The smelting plants are situated at the foot of the western fall of the divide, at an elevation of 570 ft. above the sea, on a fairly convenient sidehill location. The natural topography was a difficult one for the selection of a site, owing to the steep sidling ground, and the absence of flat areas, except in the two water-logged valleys. The terrace system of construction is employed, with its attendant advantage of utilization of gravity. In fact, a unique feature of the whole enterprise, from the mines to the railway to Strahan, is the manner in which gravity has been availed of to the greatest possible extent. The special reason for choosing the site was that it formed the head of the easiest entry for the railway, the original intention having been to connect the smelting-works with the mine by a tunnel through the saddle, or else by means of an extension of the railway. The latter, however, was never seriously contemplated, on account of the distance and cost, not to mention the severe gradients on both sides of the hill, which it would have involved. Though to the uninitiated the smelting-plant thus seems to be on the wrong side of the hill, the question of transportation of ore to the furnaces has not at any time caused any difficulties. On the contrary, it has been solved most completely, with simplicity and cheapness. The other side of the divide, i.e., Linda valley, would have been a seriously injudicious selection, as there is neither water, timber, nor limestone available in that basin, and, as in the case of the North Mount Lyell company, it would have been necessary to build the works a number of miles from the mine.

Transportation.— The first means of ore-transit installed originally and designed as a temporary expedient, but permanently retained on account of its suitability, is a counter-balanced, surface haulage line, which traverses the divide in a straight line, from the ore-bins at the

mouth of No. 4 tunnel, to the western foot of the ridge. It is a 2 ft. tram-line, running from the mine storage bins direct over the but slightly corrected natural declivities. The mine-side run is the shorter of the two. The descending smelter-side run is connected with the smelting-site by a direct continuation, the "through tram," on a 1 in 20 gradient, three-quarters of a mile long, which conveys the haulage trucks to the smelter bins, without transfer. The mine side of the haulage line is 1484 ft. long on the slope, the smelter side 2252 ft. long, and the yard on top of the saddle, 224 ft. long. The horizontal distance between the end points is 3777 ft. The rise on the mine side is 441 ft., and the fall on the smelter side 714 ft., the difference in elevation making the line perfectly self-acting during transit of ore. The haulage trucks are of a stout construction, side-discharging, and carry two tons of North Mount Lyell ore, or 2½ tons of Mount Lyell ore. Winding is done on a differential system, by means of a special winding appliance installed on top of the saddle, which consists of two wide drums, of 14 ft. and 9 ft. diameter, respectively, fixed on the same shaft, and which take a plough-steel rope each, of 4½ in. in circumference and 90 tons breaking strain, the ropes, of course, running in opposite directions and alternately, the full trucks on the smelter side hauling up the other fulls on the mine side. The lowering is done in trains of five wagons, the movement controlled by two powerful hand friction-brakes. The return trip of the wagons is facilitated, on account of comparative flats on the smelter descent, by two 90 h.p. motors, geared on to the larger drum, or by a small reversible steam-engine as a stand-by. The shunting of the trucks on top, from the mine descent to the smelter slope, is effected on suitable low gradients, by hand. The mechanical features are not perfect, but the line has eminently suited its purpose, and is a good type of its special variety. Its capacity, although originally laid out for only 500 tons in 24 hours, has been increased to easily 1000 tons a day. For a period of years it alone carried all materials from and to the mine. It is now chiefly used for the delivery of the North Mount Lyell ore, this being done directly from the top of the saddle with the aid of a "rake" of counter-balancing trucks on the mine side. The line is run in two shifts, the principal ore-haulage taking place at night. The number of men employed per shift is 13, including loaders, engine-drivers, guards or brakemen, etc., and the cost of transit across, from mine to through tram, is 6d. a ton. The ordinary speed of running down the slopes is 1000 ft. a minute. The life of the ropes is six months, though they are simply discarded for safety's sake, without being worn out. Traffic on the through tram is done entirely on the brakes, in trains of 10 trucks at a time, the return empties being brought back by a 10-ton Krauss locomotive. Arrived at the smelting-plant, the entire train of trucks is weighed in one draft, on a specially constructed Pooley dial

weighbridge, 75 ft. long and of 75 tons capacity, which is capable of weighing down to 56 lb.

The principal method of transportation of the ore, however, is the aerial ropeway. This is used for the delivery of the Mount Lyell pyrites, having a total capacity of up to 1500 tons in 24 hours. It was erected in 1898-99 and is of the Otto-Pohlig three-rope type, i.e., two parallel lines of stationary carrying ropes, and an endless traction-rope. Its construction, like that of the haulage line, was not an easy matter in this climate, or topography, the alignment, which is straight from the mine storage bins to No. 1 smelting plant, being very rough, and the profile broken up by numerous transverse gullies. The general direction of the line parallels that of the haulage and through tram. At the No. 1 plant a turn-out central station, situated just before the main line terminus, deviates the ore, nearly at right angles, to the No. 2 smelting-plant, the total burden of ore being deliverable to either plant. When the line is fully loaded it is entirely self-acting, and the main line drives the flatter side-branch through gearing in the central station. The principal particulars of the line are the following: main line, from mine ore bins to central station at No. 1 plant, horizontal length, 6750 ft.; length of branch from central station to No. 2 plant, 890 ft.; length of branch from central station to No. 1 plant, 300 ft.; vertical ascent on mine side to top of divide, 430 ft.; vertical descent on smelter side, from top of divide to central station, 920 ft.; mean gradient on mine side 1 in 2.9; mean gradient on smelter side, 1 in 2.5; mean gradient, mine to works, 1 in 14.7; number of field standards, 13, also 1 viaduct (double standard), 1 tension gear (intermediate station connecting two ropes) and a rail passage, 330 ft. long, crossing the top of the divide; hight of standards, lowest 8 ft., highest 51 ft.; tension weights on large carrying rope, 16 tons, on small carrying rope, seven tons, on traction rope, 1.9 tons; longest spans between standards, 1155 ft., 1188 ft. and 1320 ft., the last just before the central station; greatest hight of rope above ground, 220 ft.; distance of carrying ropes apart, 6½ ft.; capacity of buckets, 11 cwt. (1232 lb); sizes of ropes, loaded carrying-ropes, 40 mm. diameter, empty carrying-rope, 35 mm. diameter, traction-rope ¾ in. diameter; speed of travel of traction-rope, 6 to 7 ft. per second; rate of delivery, over one ton a minute, i.e., one bucket each half minute; number of buckets on line at one time, 80; time required for round trip, mine to mine, 45 minutes, including circuits around terminal stations.

The number of hands on the ropeway per shift is 24, which not only includes the runners serving the line direct, but all loaders and unloaders of buckets, binmen, fitters, blacksmiths, foremen, weighers, etc. In order to ensure the even loading of the line, the weight of the pyrites in the buckets is adjusted approximately to 11 cwt. at the mine end, but

final weighing is done on arrival at the smelting-works, on a hanging-scale which can be thrown in and out of line in a moment, and which weighs each bucket separately. An automatic, self-registering weighing-machine was not a success. At the central station the contents of every twenty-fifth bucket are discharged and sent to the sampling-works. The shunting of the buckets at either terminus at the ore bins, is done by hand, on the usual overhead-rail system. The cost of carriage and delivery from the mine bins to smelter bins, per ton of dry ore carried, varies from 6d. to 7d., which figure embraces all expenses attached to the work, labor, maintenance, supervision, supply of new ropes, all breakages, accidents, and repairs, electric lighting, and all costs of loading, shunting and discharging the buckets at the end stations, etc. During stoppages of the ropeway the haulage line takes its place temporarily. The local influences affecting the ropeway, such as acid smelter fumes and perennial rain, have not prevented it from giving magnificant satisfaction.

In addition to this means for the transportation of ore between the mines and reduction works, the latter are served by a complex system of-2-ft.-gage trams, of a total length of about 10 miles. This convenience was introduced even before the excavation of the site was advanced to any extent, for wagon-roads are impossible in the locality. There are in use eight four-wheel coupled tank locomotives of 6, 7½ and 10 tons weight, and 156 vehicles. The chief work, in addition to the through tram ore-service, consists of the carriage of fluxes, firewood and supplies to and from the smelting plants. The annual burden thus carried is about 400,000 tons, at an average all round cost of 6d. per ton. The smelting-plants are also connected with the main line of railway, by the customary 3 ft. 6 in. gage, on a three-rail road, formed by compounding both gages. The length of this service is five miles. The gradients to which the local topography lends itself are remarkable in the respect that the 1 in 20, or 5 per cent. gradient, is the ruling one, embracing 83 per cent. of the total mileage run. The West Coast is, in fact, eminently suited for narrow gage railways of a light caliber, and possesses several instructive applications of this system. The mixed gage smelter service is arranged on a loop, or circle system, from the valley to the smelting-plant, with a rising 1 in 29 gradient and a short 1 in 16 down gradient.

The tram-system at the works is supplemented by another 2-ft. service in the bush, for firewood supplies, an important portion of which was situated on the neighboring Madame Howard plains, some 500 ft. above the reduction works valley, and connected with the latter by an inclined surface-haulage, three-rail system, self-acting, with the up trip by electric winder. Finally, many miles of wooden rail bush-tram, 2 ft. 6 in. gage, have been constructed in course of time. These are a special feature of the country, and consist of firewood rails fastened on split firewood,

corduroy, road-beds, roughly formed, the down trips made by gravitation, the returns of empties by horse, the bolster wagons employed having wheels with very wide flanges. The loading is done from stacks of firewood accumulated at convenient points along the line, by passing the split firewood down the hillsides in wooden chutes. In place of these trams, a special method of water-fluming the firewood to the railway is also employed, where feasible, and with great success.

Fluxes.—The fluxes used consist of limestone and quartzite, or silica, and of the low grade fahlband ores, or pyritized schists, which are regarded as metal-bearing fluxes. In addition, all materials won in mining development work, and averaging about 1½ per cent. copper, with a modicum of silver and gold, which are usually very silicious, are sent to the furnaces. The basic character of the Mount Lyell pyrites necessitates the use of free silica, and the source of this is now principally the North Mount Lyell ore. Before the amalgamation it was chiefly derived from barren silicious rock.

Considering the general geological nature of the locality, it is a fortunate circumstance, from a smelting point of view, that limestone occurs in an easily accessible bank of fair size, at a distance of only half a mile from the smelting-plant. This deposit was opened up from the general level of the reduction works valley, along the course of a short and narrow gully, in which the strata stand almost vertically, with a very slight dip to the west, striking parallel to the sides of the gully, north and south. The bank is uncovered for 660 ft., until it disappears into the hill heading the gully. It is 460 ft. wide and stands in a face up to 130 ft. high. Quarrying is done in five straight benches, following the strike. An electric air-drill is in use. For some years back the stone has not been crushed, but is put into the blast-furnaces in the sizes in which it breaks in the quarry, without spalling. The character of the stone is that of a dense, dark gray, bituminous limestone, fossiliferous in places, always with obscure stratification. Though not very pure, the limestone is sufficiently so for all local requirements. It makes an excellent burnt lime. Two analyses are given in the accompanying table.

LIMESTONE ANALYSES.

Kind.	SiO_2 Per Cent.	Fe_2O_3 Per Cent.	Al_2O_3 Per Cent.	$MgCO_3$ Per Cent.	$CaCO_3$ Per Cent.
Light..........	10.92	0.57	2.88	3.00	83.40
Dark..........	3.26	0.85	0.86	Trace	95.10

The barren quartz, or silica, flux was won from the same hill, on the eastern flank of which the limestone occurs. It is one of the usual mighty banks of this rock, so abundant on the coast, and absolutely devoid of mineral value. Its character is that of a white quartzite, though often

sandstone-like, generally obscurely fossiliferous. Strike and dip are conformable to that of the limestone. The lower face of the hill proving too sandy, quarrying was conducted on the summit, 500 ft. above the valley, where the stone is more compact. The quarry was worked there in three and four benches, and at one time with the aid of 3½ miles of tram on top of the hill. The stone was sent down to a crushing and storage plant in the flat below, by means of a three-rope Otto ropeway, of the same type as the larger aerial ropeway above mentioned, 1500 ft. long, and of a capacity of 30 tons an hour. The crushing-plant consisted of two large-sized rotary crushers, mounted on top of a set of bins of 1000 cu.yd. capacity. Sandy layers in the rock necessitated the special removal of this material, which was effected by self-acting "flying foxes," or blondins, alternate motion type, with two fixed and an endless moving rope, and tripping gear for the discharge of two one-yard buckets, about 50,000 cu.yd. being tipped annually, into a side valley. All these operations, however, have ceased since amalgamation, though a little stone, suitable for converter use, is still being broken at the foot of the hill.

ANALYSES OF SILICIOUS FLUX.

Kind.	SiO_2 Per Cent.	Fe_2O_3 Per Cent.	Al_2O_3 Per Cent.
White	91.44	1.54	3.09
Dark	88.72	3.60	7.03

The overburden and waste dumps of the mines also supplied, and still supply, a certain small amount of silicious flux, in the shape of varieties of conglomerate or quartzite. Accompanying is an analysis of Mount Lyell overburden, together with two of silicious stuff from underground exploration work.

MOUNT LYELL OVERBURDEN.

	SiO_2 Per Cent.	Fe Per Cent.	Al_2O_3 Per Cent.	Cu Per Cent.
Mount Lyell quartzite	83.2	5.0	7.5	....
Lyell Tharsis dump	60.0	12.7	13.6	1.5
North Mount Lyell waste	76.4	8.0	6.6	0.82

All these materials are smelted without being crushed.

The clay for converter use may be mentioned in this place. Owing to the ready decomposition of the local schists, this important material is extremely abundant, and occurs in many varieties. It is cheaply won, close to the smelting-plant. As an indication of its composition the following may be cited.

CLAY FOR CONVERTER USE.

	SiO_2 Per Cent.	Al_2O_3 Per Cent.	Fe_2O_3 Per Cent.	CaO Per Cent.	MgO Per Cent.	H_2O & undetermined. Per Cent.
White clay...	62.52	23.89	0.26	0.25	0.4	12.68

Certain kinds of this material are made into a good fire brick, on the spot, while the more ferruginous varieties, in the form of dry, decomposed schist rock, furnish the material for the red bricks used.

Water Supply.—The topography at the reduction works is unsuitable for cheap or easy water-storage on a large scale, on account of the steep and very narrow character of the gullies, and the permeable nature of the ground, so that no large storage reservoirs have been built—in fact, the almost constant rainfall makes the conservation of water unnecessary. The supply diminishes slightly in the summer-time, for a month or so, but this only exacts more attention from the furnace-hands in the regulation of the slag-granulating. For high pressure purposes there is a reservoir, situated close to the smelting-works, at an elevation of 180 ft. above the slag-dump level, of 300,000 imperial gallons capacity, which is fed from a stream off Mount Owen by a line of 1¾ miles of fluming and race (ditch). The main supply is derived, however, from the Queen river in the valley, by pumping, the average quantity lifted being 2¼ million imperial gallons daily, raised into tanks 160 ft. above pump-intake. The older pumps used for the purpose were two duplex double-acting plunger pumps, of 10x36 in. cylinders, driven by 75 h.p. motors, but these are now replaced by three electrically driven centrifugal pumps, of 1440 r.p.m., and a discharge of 1¾ million imperial gallons daily, each. Special fire pumps, with a complete reticulation through all parts of the plant, need not be mentioned. All the condenser water used flows into the granulated-slag launders. The surface water of the district, especially in drier times, is progressively becoming more acid in character, which effect increases the naturally great corrosion of all iron parts, characteristic of the climate. For this reason furnace pipe-fittings, valves, unions, etc., are made of gun metal only, brass being found useless. Lead pipe is also very extensively used for jacket connections, both on account of its easy bending, as well as its standing qualities, etc.

Construction and Repair Works.—In point of repairs and construction the reduction works are wholly self-reliant, all but the very largest jobs being done on the spot. A well appointed machine-shop, situated in the valley, with a full complement of motor-driven machine-tools, serves this purpose, supplemented by a pattern shop with a large stock of local patterns, a complete foundry with two cupolas and a travelling crane, capable of casting 3½ to 5 tons of metal at a time, and other plants, like

brass-foundry, fitting-shop, boiler-shop, tinner's-shop, carpenter- and wood-turning shop, etc. In addition, a complete brick- and pipe-making works, and two sawmills, are on hand. The introduction of the foundry has reduced the consumption of slag- or matte-pots, from 2000 a year down to a single one in two days. Mention may also be made of an oil-refinery, for the purpose mainly of dehydrating and settling spent lubricant, which has reduced the weekly oil-consumption to 13.5 imperial gallons of cylinder oil, and 37.8 imperial gallons of machine oil, these figures standing for the whole smelting and construction departments, but, with the discarding of the rotary blowers, they will be still further decreased.

Local timbers consist principally of myrtle (*Fagus Cunninghamii*), King William pine (*Arthrotaxis cupressoides*) and Huon pine (*Dacrydium Franklinii*), with some celery top pine (*Phyllocladus rhomboidalis*) and gum (*Eucalyptus*), and, with the exception of the second, are of great tensile strength, surpassing the ordinary timbers of the northern hemisphere. The third mentioned, especially, is a very durable, high-class wood. The last mentioned is the ordinary building timber. The local forests are distinguished by the occurrence of many varieties of useful and ornamental timbers. A peculiarity of the woods is that the majority of them are heavier than water. As, furthermore, their hardness is great and the moisture in them runs high, carpentering is made slow and costly.

Fuels.—For the generation of steam, and for heating purposes, the local wood-supply is almost exclusively used, i.e., there is only a slight consumption of coal, this coming from Newcastle in New South Wales. Local (Tasmanian) coals are relatively dearer, owing to inferior quality. The most abundant firewood is myrtle, and the best a variety of "tea-tree" (*Leptospermum*, or *Melaleuca*), locally known as manuka. The average heating power of these woods, though seemingly low, on account of a very short, non-smoking flame, and the exudation of abundant water from the ends of the sticks during combustion is, in reality, more than equal to that of the ordinary wood-fuels (pines) of the other side of the world. The extent of deforestation, on account of timber and firewood-getting, is now considerable, and reaches for six miles around the reduction works, the weekly consumption for all purposes, at the mine and smelting plants, being about 1200 tons with a simultaneous coal consumption of about 100 tons a week, in all departments, not including the railway. The difficulties of delivery are chiefly responsible for the cost of firewood, which ranges from 8 to 10s. a ton (80 cu.ft.), landed at the boilers. The relative heating power of the wood being rather high, compared with coal, it has, however, so far kept the latter out.

The company manufactures the coke consumed in the smelting operation at its own coke-works at Port Kembla, near Wollongong, in New

South Wales, some 730 miles distant from Mount Lyell. The works are situated on the sea-coast, with good loading facilities, and the coal used, though rather dry, is fairly well-coking, with about 14 per cent. of ash. In its better varieties it yields a 36-hour coke of great hardness. It is roughly dressed before coking, the ash contents of the coke being 16 per cent. The ovens are of the European tunnel-shaped type, 24 ft. long, with circulating side- and bottom-flues, and provided with accessory hydraulic apparatus, i.e., coke-pusher, door-lifts, etc. The weekly capacity is 10 tons each, the number 62. The usual production is about 500 tons a week, only half of which is consigned to the reduction works, the rest being sold for foundry and smelting purposes. The company was the first Australian smelting enterprise to make its own coke, notwithstanding the small consumption, and, needless to say, the venture has splendidly justified itself. The coke is shipped in bags, as being the least costly method, but is subject to a great deal of disintegration on the long journey, and from repeated handlings between coke works and furnaces, so that it sometimes carries a high percentage of breeze, besides unavoidably accumulating a good deal of moisture.

Sampling-Works.—All metal-bearing materials entering and leaving the reduction works are carefully sampled, for an elaborate, accurate record is kept of the flow of their metallic contents. A complete automatic sampling works, situated adjacent to No. 1 plant, and accessible from the ropeway and haulage line by means of short sidings, is installed for the sampling of ore and all other metal-bearing materials handled, and consists of the usual arrangement of storage-bins, stone-breakers, crushing-rolls, elevators, etc., the sample being cut out by Vezin sampling-machines. In the case of the Mount Lyell pyrites, the contents of every twenty-fifth bucket are taken to this establishment; in the case of the North Mount Lyell ore every twelfth haulage truck, and, of other ores and materials, from one-sixteenth to the entire bulk is sampled. The Mount Lyell ore consignments are divided into parcels running from 4000 to 7000 tons at a time; those of North Mount Lyell ore into 900 to 1200 tons. In the case of middle-products the custom is to form lots by cutting off the production weekly. The result of the sampling is from a half to one and a half tons of quarter-inch stuff, which is further reduced by the usual Jones sampler, followed by sample-grinding machines, etc., the final pulp being sent to the assay-office under lock and key.

The assay-office and laboratory follow well-known American lines, not alone in arrangement and equipment, but also with regard to the assay-methods used, and call for no special remark.

Smelting-Plant.—The reduction works consist of three establishments, viz., Nos. 1 and 2 smelting plants and the converting plant. The first-mentioned is now wholly out of commission. The original two furnaces

erected in it in 1895 - 96 were increased by four more during the first two years, the total capacity, under the hot-blast, low-pressure method then followed, being about 500 tons of Mount Lyell ore daily, together with the requisite tonnage of barren fluxes, four to five furnaces doing the work. The dimensions of the furnaces at the tuyeres were 168x40 in., except No. 3, which was made 126x36 in., with the object of using it for the concentration of matte only. The size of throat in each case was 15 in. larger all round; the distance from tapping floor to feed floor, 20 ft.; the hight of ore-column, from tuyeres to edge of downcast, 9½ ft. Nos. 1 and 2 were the largest copper-matting furnaces erected up to that day, which is also true of furnaces Nos. 7 to 11, subsequently built, though the respective sizes have since been quite outgrown elsewhere. The diameter of the tuyeres was 3 in. throughout, there being 24 in No. 3 furnace and 32 in the other five furnaces in No. 1 plant. The general construction (designed in 1894) followed ordinary lines of continuous-flow copper-matting furnaces, with elevated bottom-plate, downcast, etc., the most important departure, perhaps, being a steeper bosh on all four walls, the closer placing of the more numerous tuyeres, a greater hight of the latter above the bottom of hearth than usual, i.e., 29 in., and, generally, a stouter construction, with special features required by the use of a hot blast. The bosh-lines started 1 ft. above the center of the tuyeres, and the bosh itself was made 7½ in. in 4 ft. vertical. The original introduction of the simplified sump-furnace discharge of the slag and matte, with inclined bottom-plate and tymp-jacket, in connection with pyrite furnaces, I may claim for myself, as it was used during experiences in Montana and Colorado, as far back as 1891, though it has since been introduced in various modifications, some of which are patented. It was, however, nothing more than the translation, into the modern mechanical language of waterjackets, of a well-known, ancient German "Ofenzustellung" Besides the sump- (or trapped blast) overflow, the furnaces were provided with a low-lying taphole at each end, and one behind. The charge-floor was surmounted by a brick and iron hood, with an escape-stack for the emission of that portion of the fumes which is not drawn into the downcast. The furnaces were set lengthwise of the tapping-floor, a departure first suggested for pyrite smelting by me, though no credit is claimed for it, since it was only copied from the old blast-furnaces of the Parrot works in Butte. The furnaces of No. 1 plant were blown in successively as follows:—No. 1, June 25, 1896; No. 2, October 6, 1896; No. 3, January 24, 1897; Nos. 4 and 5, September 13, 1897; and No. 6, May 26, 1898. Matte was concentrated in all of them from the beginning, as dictated by necessity.

The five furnaces of No. 2 plant were erected in 1898, and made larger, viz., 210x42 in. in the tuyere region, and 15 in. extra all around the throat,

with originally 32, but now 48, three-inch tuyeres, all other features, like furnace-hight, bosh, etc., remaining the same as in the first set. But the hight of column, i.e., distance from tuyeres to edge of downcast, has since been increased to 11 ft. The doubling of the plant had the object of increasing the capacity from 500 tons daily to 1000 tons, the ore-smelting being performed chiefly in No. 2 plant, and matte-concentration in some of the furnaces of No. 1 plant, keeping a total of six to eight furnaces in blast. This batch of furnaces was blown in as follows:—No. 7, Oct. 3, 1898; No. 8, Oct. 18, 1898; No. 9, Jan. 26, 1899; No. 10, May 8, 1899; No. 11, June 29, 1899.

The general design and arrangement of the smelting-plants is the usual terraced one, followed in American lead and copper smelting. The furnace-gases discharge into the ordinary straight flue, immediately behind the row of furnaces, the further wall of which flue constitutes the retaining wall of the bin-floor above, while the out-of-doors portion of the flue ascends a hillside to a high chimney behind the works, the length and size of flue being made sufficient to guarantee a satisfactory recovery of flue dust. The capacity of the ore-bins at No. 2 plant is 9500 tons of ore, fluxes, etc. They are provided with hopper bottoms, and filled directly from above, from the shunt-rail system of the ropeway, and the trams.

The first five furnaces of No. 1 plant were originally fitted with a double tier of steel-plate jackets, later on supplemented by a third tier of cast-iron "water-boxes." The plate jackets, however, soon grew disastrous, through constant repairs, in consequence of extensive, uncontrollable corrosion. No. 6, when built, was constructed of cast-iron jackets throughout, all except the tymp-jacket of the sump. This important jacket was part and parcel of the original furnace-design, and was made up of a plate of steel, bent to shape, with a welded seam on top, the ends being closed by two heavy plates riveted in. Use is latterly made of a cheaper, more easily made, pipe-cooled, solid cast-iron construction. In No. 2 plant the former slanting upper jacket and the vertical water-box above it, were united into a single cast-iron jacket, having the reverse shape of the ordinary knee-form, so that the inner inclined faces of the first and second tier together formed the bosh-plane. This new second tier was then surmounted by a third, so that the greater part of the furnace-shaft is water-jacketed, although it is not considered advantageous to water-jacket the furnace all the way up to the feed-floor. At the present time, the pipe cooling of these large water-jackets is being extensively made use of, instead of the usual water-space construction, an important departure in jacket-design, which, although not wholly novel, is worth special mention, as it leads to a very much prolonged life of the castings. No. 12 furnace, recently erected as a sixth furnace at No. 2

plant, is very largely composed of pipe-cooled jackets, the water-space, however, being retained in the battery surrounding the hearth, which seldom wants renewal. The hight of this new furnace is also greater, having the distance from center of tuyeres to edge of downcast 18 ft.

The ordinary rectangular forehearth, but stationary, is employed, a main one, 19x5½ft. by 2½ ft. deep, water-cooled by means of pipe-coil side-plates, now receiving the molten materials, and collecting the matte, and three or four other, smaller ones, of older types, ranged before it, being traversed by the slag-stream. On some of the furnaces the main forehearth is still only 5x8 ft. by 2 ft. deep, it being held that the better separation presumed to characterize a single, more spacious forehearth is more adequately obtained by placing three or four forehearths in line, thus achieving an equal, or longer travel of slag, with an additional advantage derived from the drop into each forehearth. The total distance travelled by the slag is thus from 20 to 30 ft. The auxiliary forehearths all furnish a small quantity of matte, and are tapped several times a day.

The rate of concentration being high, the matte-fall irregular, and the variation in grade of ores, also the fickleness of the mechanical conditions within the furnace, being great, the tenor of the matte varies to a degree which makes it necessary to remain in closest touch with it, so as to effect instantaneous changes in the furnace mixture. For this and allied reasons, of a local nature, the older system of cooling the matte between furnace and remelter has been adhered to up to date, notwithstanding the extra handling and the cost of remelting involved. The matte is tapped from the main forehearth through a small solid cast-iron, pipe-cooled tap-jacket, originally adapted by me, years ago, from the well-known Lürmann cinder-block, into matte-pots of ordinary size, which are wheeled a few paces, and either allowed to cool, or poured in layers on a slag floor, within squares ("paddocks") formed by heaping up four low walls of granulated slag. The grade of the matte is controlled with the assistance of the shift bosses, who are taught to make a quick copper determination of the Parkes cyanide method, with simple apparatus (improvised balance, burette, etc.), using standardized solutions sent up from the laboratory. The copper-assay of the matte is made from a rod-sample in a few minutes, with an accuracy of about 1 per cent. The expedition, and the advantage of thus relieving the assay-office, are obvious. All the work is done in eight-hour shifts, the wages ranging from 7s. 6d. for ordinary labor to 8s. 6d. for "tappers," and 10s. for "furnacemen," the last in charge of two furnaces each at a time.

The blast-furnaces were wholly fed by hand until recently, for the introduction of the smelting mixture in a proper manner is the most delicate and important feature of pyrite smelting. Since October, 1906, the operation is facilitated and cheapened by the use of a locally evolved,

patented, feeding-appliance, which pushes the charges bodily directly into the furnace off the charge-floor. The contrivance is hydraulically actuated, and consists of a steel-frame, provided with a line of hinged pushing-plates in front. Each side of the furnace is served alternately. The exclusive replacement of the feeder's skill by mechanical means does not appear calculated to lead to the best results achievable in this line of work. The local mechanical feeder permits of reducing the attendants to one feeder and a boy per furnace. Since the furnaces were built a little low for present practice, they are filled up to the very charging-plate on the wall opposite the downcast, in order to utilize the highest possible column, making the latter over 13 ft. high on that wall. The delivery of the charge constituents is done in the older way, by means of one-man, two-wheeled hand-carts, well balanced, and of a capacity of half a ton of pyrites, the extremely variable requirements of the smelting, in point of composition of mixture, not yet having countenanced the introduction of purely mechanical means of delivery.

The slag is granulated, the stream off the last forehearth being simply struck by a jet of water from a 1 in. pipe, with a pressure of 60 lb. per square inch, and dropping into a launder, in which the entire jacket-water of the furnace is united. The disposal of the granulated slag, however, exhibits some novel features, for it has for some time past been necessary to lift it. For this purpose the underground slag-launders of five (or six furnaces) combine outside of the building in a grated hopper, out of which the mixture of slag and water flows to a special, motor-driven pump of local design. This raises the slag-laden water to a hight of 24 ft., and forces it through a 10 in. level pipe over distances of up to over 500 ft., delivering the stream at any desired spot on the dump, from which a free run may be allowed it. The whole arrangement is in triplicate. The pumps require 40 to 45 brake horse-power each, and singly cope with the entire slag-production (some 900 to 1000 tons a day), together with a volume of water equal to 16 times the weight of the slag. Their construction is that of a diffusor-chamber centrifugal pump, diameter of impeller 20 in., number of revolutions 650 per minute, inlet 10 in., outlet 9 in. The life of the pumps is short, but the cost of making them is small and they give no trouble. The cost of the entire work, including power, attendance, pumps, piping, repairs, duplicates, etc., is equal to a shade over one penny per ton of ore treated.

The hot-blast stoves formerly used were of the hanging U-tube type, substantially built, rather more capacious than has been customary under similar circumstances, and arranged on the counter-current principle. No. 1 plant had four stoves of 56 pipes each, No. 2 plant the same number, with 70 pipes each. The pipes were arranged in parallel, suspended from heavy girders with ceiling plates between. The two legs

of the U-pipe were joined, so as to form two D-shaped spaces, with a central partition not reaching quite all the way down the pipe, and with an elbow at the upper extremity of each leg, diameter of each tube 9 in., length 9 ft. The construction was very free from leaks. The path of the fire-gases was lengthened by means of baffle-walls in the heating chamber. At first blast-temperatures of about 650 deg. F. were striven for, but in view of the heavy fuel-consumption this was gradually lowered, until, at the end of the hot-blast regime, temperatures of 250 deg. only were used. The efficiency of the stoves varied with the temperature desired. In the early days it was about 50 per cent. for 500 to 600 deg. When the firing was reduced, in conformity with the lowering of the blast temperature, to such an extent that the grate area used became only one-third of that originally provided, the efficiency rose correspondingly, until it amounted, for months at a time, to 80 per cent. The desire to minimize the cost of heating the blast, however, led to experimental comparisons with cold blast, which finally brought about the total abandonment of hot blast in 1902-03. The innovation began in the matte-concentration operation, a small extra addition of coke for a while supplementing the increment of heat formerly supplied by the heated blast. Simultaneously, hot blast was applied to other furnaces, *without* coke. Eventually, the present method was settled upon, which consists of the use of a cold blast, with the application of a modicum of coke, although it is considered that even the latter can be discarded absolutely, under intensified pneumatic conditions and a better utilization of the heat generated within the furnace, especially for the preparation of the descending mixture.

The departure from hot blast to cold blast required the application of the total blast-capacity of the works, formerly serving six to eight furnaces in both plants, to only four furnaces, running constantly, in No. 2 plant. For this purpose all the available blowing paraphernalia was congregated in the latter plant. This had thus the original nine No. S Roots' blowers, of a nominal displacement of 116 cu.ft., each direct-coupled to a vertical compound condensing engine, and all discharging into a common blast main leading to the furnaces. The indicated horse power utilized for blast from these blowers was 1380. Latterly, however, a Parsons steam-turbine-driven turbo-blower was added, capable of supplying 18,000 cu.ft. of free air per minute, of 54 oz. pressure, 460 brake h.p. The annual delivery of the Roots' blowers was thirty thousand million cubic feet of free air (calculated on the basis of a delivery of 66 per cent.); of the turbo-blower mentioned, seven thousand millions; total, thirty-seven thousand million cubic feet of free air, equivalent to about 2.7 tons of air per ton of ore-bearing material smelted. The nine Roots' blowers, which were not designed for the work that was, in the course

of time, demanded of them, have just been replaced by two further Parsons turbo-blowers, each 1025 brake horse power and with a delivery of 36,000 cu.ft. of free air per minute, of 64 oz. pressure. The steam consumption of the turbo-blowers is $13\frac{1}{2}$ lb. per brake horse power, very much less than that of the discarded Roots combinations.

For the supply of electric energy there is a model installation, consisting of three 400 kw. Parsons Brown-Boveri turbo-generators, the power being sent to some 30 points, including the mines, as already mentioned. A complete electric light installation, with generators in duplicate, need not be specially mentioned. It also supplies the township of Queenstown. The power current is three-phase alternating, 550 volts, and 50 cycles per second.

The steam generation plant of the reduction works consists of 16 Babcock & Wilcox and three Stirling boilers, supplemented by superheaters (including the Watkinson type), Allen surface condensers with Edwards air-pumps, two Green economizer installations with draft fans, etc., besides 10 smaller boilers, mostly multitubular, in remoter plants. The total boiler capacity is 110,000 lb. of steam per hour. The cost of steam generation is relatively high, but the substitution of water power has not been justified by circumstances, the local utilization of this natural power being encompassed by unprofitable difficulties and uncertainties.

Smelting Results.—As the purpose of this article is only descriptive, the nearer discussion of the smelting-process cannot be entered upon.

The establishment at Mount Lyell is the oldest of its kind; in fact, it first demonstrated the applicability of the pyrite smelting process to copper ores on a large scale. It would appear that, to this day, the process is here conducted in its nearest approach to the ideal form, notwithstanding the local conditions under which it is carried out, though favorable, are far from easy or perfect. Nevertheless, although weekly runs are possible without the use of coke, or other carbonaceous fuel, except for raising steam, the difficulties surrounding the method are still such that the total exclusion of fuel in the blast-furnace cannot yet be made permanent practice. This is largely due, no doubt, to the fact that the smelting apparatus employed does not permit of reaping the full benefit of the thermal energy of reaction which pyrites, air and silica are capable of. The necessity still exists of adding a small amount of carbon, not in the heating of the blast, which is a less efficient method of applying it, but as solid fuel, coke. The pyrite furnaces of the day may be regarded, among other things, as not permitting of a sufficiently high charge-column for the best work. There is too great a loss of heat still at the throat in "pyrite" work, as well as, apparently, in all smelting of sulphides. The location of the stock-line is, or should be, a function of the air-supply. There should not be any escape of free oxygen, or

air, from the throat, nor of highly heated gases. As far as possible with the apparatus at hand, i.e., furnace-hight, it is believed that the Mount Lyell practice achieves these desirable ends. The furnace gases, as a rule, show only traces of free oxygen, or none. Their sulphurous acid contents by volume run up to 12 per cent. and over, within the furnace (reduced to much less, by inrushing air, above the throat). However, even inclusive of the deficiencies referred to, the utilization of heat in the Mount Lyell furnaces compares very favorably with that of iron, or the better conducted lead blast-furnaces, and considerably surpasses that current in ordinary matting processes. Thirty-five per cent. of the total heat generated at Mount Lyell is devoted to the chemical work of smelting and the fusion of the solid products. Another 35 per cent. is absorbed by the dissociations preceding or accompanying the chemical reactions, and an abductive heat-loss of 30 per cent., from various causes, makes up the balance. This direct efficiency of 70 per cent. is high. It represents the degree to which the heat actually evolved during the smelting does the work which is expected of it. The absolute efficiency of the furnaces, i.e., the extent to which the thermal capacity of the fuel used, viz. FeS_2, is utilized, is 60 per cent., and this figure also compares favorably with ordinary matting processes with the use of carbonaceous fuel, in which the absolute efficiency of the furnace scarcely transcends 40 per cent., the analogous figure in the case of the best conducted lead smelting furnaces being 60 to 70 per cent. In this connection it should, moreover, be borne in mind, that the application of the full calorific value of FeS_2 as a standard puts a special strain on the conditions under which the work of pyrite smelting is valued, for the standard is a severely high one. This fuel is not fully consumed, nor can it be. On the one hand, three-sevenths of the sulphur present in the FeS_2 is at once volatilized, without doing anything more than robbing so much heat. Then, a further quantity is eliminated by heat without being oxidized, and a certain, unaffected remnant is necessary to enter the matte. A minimum of 95 per cent. of the iron is oxidized in the Mount Lyell furnaces, and simultaneously only about 35 per cent. of the total sulphur present in the pyrites, the remainder escaping unburnt within the furnace, or going into the matte.

The accompanying figures represent the work done during the last three years, since the coke consumption has fallen below 2 per cent., taking into account the entire consumption of this fuel, including its use for blowing-in and other purposes about the furnaces, and comprising in its weight the presence of a large amount of breeze, and up to 10 per cent. of moisture.

Average slag analysis for the period: SiO_2, 32.47 per cent.; FeO, 52.15 per cent.; CaO, 4.77 per cent., Al_2O_3, 7.22 per cent.; BaO, 0.90 per cent.; S, 0.88 per cent.; Cu, 0.39 per cent.; Ag, 0.189 oz. per ton.

RESULTS FOR THREE YEARS FROM APRIL, 1904, TO APRIL, 1907.

	Tons, dry	Tons, dry	Average Copper Per Cent.
Mount Lyell ore	867,213	....	0.89
North Mount Lyell ore	341,536	....	6.01
Purchased ores	2,991	....	7.54
Total	....	1,211,740	2.35
Metal-bearing fluxes	32,926	....	1.45
Total ores	....	1,244,666	2.33
Flue dust (blast-furnaces and converters)	22,253	....	4.81
First matte	27,543	....	15.04
Converter slag	49,984	....	2.12
Converter linings	6,771	....	11.18
Total metal-bearing materials	....	1,351,217	2.65
Barren silicious flux	13,680	....	....
Limestone	102,819	....	....
Blast-furnace slag	150,218	....	....
Total material furnaced	....	1,617,934	2.21

Coke consumed for all purposes 24,671 gross tons =1.525% of burden furnaced =1.83% of metal-bearing material furnaced. =1.98% of ores furnaced.

During the same period the amount of air supplied was 3,356,463 tons; the average blast-pressure, 47 oz.; the average matte tenor, 40.57 per cent. copper, and the average rate of concentration, 17.4 : 1, figured on the ore. This was done in a single smelting, it may be remarked, the "first matte" listed in the table being merely matte which had not quite reached the converter grade through accidental causes.

As these figures are averages extending over a long period, they do not represent the more recent aspects of the work, which has been progressively improved, so that, for instance, the rate of concentration during the official year ended Sept. 30, 1907, has been 20 : 1, figured on the mineral-bearing substances, i.e., the formation of a 44.3 per cent. copper matte out of a 2.25 per cent copper ore. This is still achieved in a single smelting, and the grade of the matte might be raised yet higher, were there any advantage.

With regard to the treatment of the flue dust, all the current methods, such as hand-bricking, machine briqueting, kiln-burning, reverberatory fusion, etc., have been exhausted in the course of time, without satisfaction, and abandoned as too expensive, though the cheap method first mentioned, of forming rough, hand-made lumps, by adding a very small amount of common clay, and drying them, was retained the longest, on account of its relative cheapness and suitability. At present, flue dust is clinkered by a special, patented method, with either none, or only a minimum of oxidation, the prime object of not roasting the material, but retaining its sulphide condition unaltered, being dictated by the general scheme of treatment.

The general recoveries in smelting, inclusive of those in the bessemerizing operation, the results of which cannot well be differentiated, are as

follows, for the entire period since the beginning of smelting operations: Copper 85.72 per cent., silver 92.57 per cent., gold 104.280 per cent. These figures are obtained from careful records, accurate sampling and close assaying, and by debiting the smallest traces of metals, and cover a period of 11 years, and the treatment of the following tonnages of mineral-bearing materials. It is to be understood that the results include the double smelting formerly practised (i.e., when the ore was smelted into low-grade matte and this resmelted) and all experimental troubles.

BLAST-FURNACE RESULTS SINCE COMMENCEMENT TO SEPT. 30, 1907.

	Dry Tons.	Assay.		
		Copper. Per Cent.	Silver. Oz. per Ton.	Gold. Oz. per Ton.
Mount Lyell ore	2,753,707	1.99	2.32	0.0840
North Mount Lyell ore	466,767	6.11	1.82	0.0051
Purchased ores	164.563	6.84	2.97	0.0075
Total	3,385,037	2.79	2.28	0.0696
Metal-bearing fluxes	135,299	1.60	0.27	0.0150
Total ores	3,520,336	2.75	2.20	0.0670
Flue dust (blast-furnace and converter)	67,102	4.73	4.08	0.1120
First matte	408,838	15.04	12.17	0.4260
Converter slags	129,324	3.11	1.81	0.0400
Converter linings	18,622	9.56	7.04	0.1860
Total metal-bearing material	4,144,222	4.03	3.23	0.1030
Silica flux	420,998	....	....	....
Limestone flux	367,970	....	....	....
Blast-furnace slag	567,466	....	....	....
Total burden	5,500,656	3.02	2.43	0.0777

Coke, 131,083 gross tons =2.38 per cent. on burden =3.16 per cent. on total metal-bearing material =3.72 per cent. on total ores.

Converting Plant.—This establishment was erected in 1896-97 along the lines of the older Parrot works style of arrangement of parts, with the vessels blowing into an elevated flue, beneath which the converters are moved to the lining-room by means of an hydraulic car, differing, however, by having the run very short and by mounting the vessels there, by their trunnions, on trestles or horses, in an elevated position for relining. The plant consists of 14 vessels and six stands, with hydraulic tilting gear set up completely above ground, with a hollow fixed piston and a movable cylinder bearing the rack, actuated by a high-pressure, self-regulating pump, accumulator, etc. The vessels are of the Stalmann square type, slightly modified, the only instance, it is believed, of their retention, but locally due entirely to the satisfaction they have given. The vessels are 8 ft. high by 5 ft. square externally, and they blow from 25 to 40 tons of blister a day, according to the grade of the matte, the present output of matte being blown to copper in four or five days per week; with between

three and four stands. A vessel crew, consisting of three men, attends to two vessels at a time, working 12 hour shifts. Large barrel converters are on hand from the late North Mount Lyell company, but their substitution does not present any advantages under the local circumstances. Two steam-driven, horizontal, Riedler blowing-engines, compound condensing duplex type, capable of supplying 3000 cu.ft. of free air per minute each, and two small electrically driven Roots' blowers for the remelting furnaces, furnish the blast, the pressure at the converters being 8 lb. per square inch. One thousand three hundred and twenty-eight millions of cubic feet (1,328,000,000) of free air are compressed annually. For the preparation of the linings the only plant necessary is two grinding-pans. The lining-room is controlled by a travelling crane, which only has to lift the hoods, or tops, of the vessels. For the condensation of the fumes and flue dust there is a large system of flues and dust-chambers, connecting with the chimney of No. 1 smelting plant situated on an adjacent hill.

The converting operation presents no novelties, except minor local features. Comparative working data of these small vessels, for extreme grades of matte, however, may be of some interest, and are given for two full years.

CONVERTING OPERATIONS.

	1899	1906
Average copper tenor of matte blown	51.1%	40.95%
Average copper tenor of blister produced	98.82%	98.87%
Number of full days in operation annually	279.6	285.6
Number of vessels continually blowing during that time	2.35	3.74
Size of initial charge of matte, in long tons	1	1.6 (a)
Size of final charge of matte, in long tons	3.6	3.6 (a)
Number of blows per converter in 24 hours	14.84	10.32
Duration of blow from beginning to end in minutes	30 to 120	60 to 120
Duration of blow per ton matte, in minutes	21	15.94
Duration of blow per ton blister, in minutes	42.89	45.13
Life of lining in blows (campaigns)	8 to 10	4 to 6
Average capacity of vessel-life, in tons matte	12.45	11.02
Average capacity of vessel-life, in tons blister	6.131	3.89
Average capacity of plant daily, in tons matte	68.17	90.37
Average capacity of plant daily, in tons blister	33.58	31.9
Consumption of air per ton of matte, in cubic feet	49,533	58,403(b)
Consumption of air per ton blister, in cubic feet	100,564	165,423(b)

(a) Repeated, or double-banked.

The blister copper is poured into cakes of about 24x16x2½ in., in iron molds of a special construction, possessing an average life of 1½ to 2 years per mold, owing to a wearing center-piece. The blister is parcelled into 50 ton lots, consisting of 500 cakes. Each cake is sampled by two ⅜ in. drill perforations, located over the surface on a 150-hole chess-board distribution, and drilled alternately into the top and bottom surfaces.

All the blister is shipped to the Baltimore Copper Smelting and Rolling Company, in Maryland, U. S. A., where it is treated on contract, its

identity being merged in that of the well-known brands sold by that concern. The question of refining the blister locally has been exhaustively investigated, and no commercial or technical advantage has been found to rest in the idea.

GRADE OF MATTE AND BLISTER.

Since the beginning, in 1897.	Copper. Per Cent.	Silver. Oz.	Gold. Oz.
The average grade of the blister is.........	98.82	85.68	2.955
The average grade of the matte is..........	44.21	36.31	1.236
The average grade of the matte at present is	45.01	35.49	0.980

The recoveries in the bessemerizing operation stand as follows, ascertained with all possible care: Direct, in the converter *per se*, crediting the full content of the various middle products, like converter slags, linings, scrap, etc., they are, copper 98.83 per cent.; silver 99.32 per cent.; gold, a plus. The recoveries, after debiting the losses incurred in the blast-furnace treatment of the slags and linings, are: Copper, 97.98 per cent.; silver, 98.82 per cent.; gold, a plus.

Financial.—The following is a complete table of the treatment costs since the beginning, expressed in half-years, and standing for the long ton of ore, exclusive of metal-bearing fluxes:

COST OF MINING AND SMELTING.

Half-year ended	Mining		Overburden removal		Smelting		Converting		Total		
	s.	d.	s.	d.	s.	d.	s.	d.	£	s.	d.
Mar. 31, 1897	1	8.27	2		18	1.64	3	10.39	1	5	8.30
Sep. 30, 1897	1	4.88	2		16	2.44	3	7.78	1	3	3.10
Mar. 31, 1898	2	4.64	2		17	9.87	2	9.18	1	4	11.69
Sep. 30, 1898	2	5.83	2		16	5.31	2	5.33	1	3	4.47
Mar. 31, 1899	2	3.56	2		16	3.61	1	11.19	1	2	6.36
Sep. 30, 1899	2	5.31	2		17	10.45	2	2.15	1	4	5.91
Mar. 31, 1900	2	11.76	2		15	10.70	2	2.30	1	3	0.76
Sep. 30, 1900	3	5.00	2		14	11.89	2	0.29	1	2	5.18
Mar. 31, 1901	3	1.40	2		15	9.65	2	1.16	1	3	0.21
Sep. 30, 1901	2	6.86	2	1	14	8.04	2	0.45	1	1	4.35
Mar. 31, 1902	2	3.89	2	1	14	8.17	2	2.15	1	1	3.21
Sep. 30, 1902	2	1.01	2	1	14	0.24	1	6.06		19	8.31
Mar. 31, 1903	2	2.43	2	1	12	9.81	1	5.13		18	6.37
From Aug. 11, 1903 to Mar. 31, 1904	5	1.56	1	6	8	5.54	1	6.06		15	1.16
Sep. 30, 1904	5	6.55	1	6	7	6.33	1	7.37		14	8.25
Mar. 31, 1905	4	11.91	1	6	6	5.13	1	7.35		13	0.39
Sep. 30, 1905	5	1.89	1		6	7.15	1	6.10		13	3.14
Mar. 31, 1906	5	3.82	1		6	9.98	1	5.25		13	7.05
Sep. 30, 1906	6	0.87	2		8	2.36	1	7.22		15	10.45
Mar. 31, 1907	5	10.12	2		7	5.32	1	1.53		14	4.97
Sep. 30, 1907	5	10.69	2		8	4.19	1	2.32		15	5.20

There was no half-yearly balance-sheet issued for the period ended Sept. 30, 1903, the amalgamation of the two companies having been effected on Aug. 11, 1903, and the new company first reporting from that date to March 31, 1904. The mining costs since the first mentioned date combine the underground mining at the North Mount Lyell mine,

etc., and the open-cut excavations at the Mount Lyell mine, including the overburden charge. The latter is given in brackets.

The mining costs embrace the complete expenditure incurred in the work, such as ore-breaking, explosives, development, motive-power, supervision, upkeep of all kinds, and also contain certain costs not truly of a mining nature, such as the delivery of ore to storage bins, etc. The smelting costs likewise comprehend the usual expenditure attaching to the operation, such as labor, coke, stores, fluxes, motive-power, etc., commencing with the sampling of the ore, and finishing with the delivery of the converter matte to the converters, but they also include various incidental costs, not truly of a smelting nature, such as transportation of ore from mine-bins to smelter-bins, maintenance of smelting plants and buildings, water-supply, dead work at the quarries, the collection and re-treatment of fluedust and other middle products, both at the furnaces and converters, all transportation of products between plants, laboratory work, telephone service, office expenses of all kinds, maintenance and repairs of construction plant of all descriptions, etc. The converter item is equally comprehensive, in fact, the figures tabulated represent all costs attaching to the entire work performed, except Melbourne and London office expenses and amortization. The shipping and refining of the blister copper also is not included, not being local cost.

A dissection of the lowest cost of blast-furnace smelting, i.e., that for the half-year ended March 31st, 1905, is the following:

COST OF SMELTING.

Transportation from mines to furnaces, by means of aerial ropeway, calculated on the general ton of ore	5.04d.		
Ditto, by means of haulage line and through tram	6.80d.		
		11.84d.	
Sampling of all ores and middle products, fluxes, etc.		1.72d.	
Blast-furnace labor		1s. 9.32d.	3s. 6.11d.
Blast-furnace coke (1.1 per cent. on burden)		8.08d.	
Barren quartz flux		0.06d.	
Barren limestone flux		2.85d.	
Metal-bearing fluxes (fahlband ores, etc., including mining and delivery)		1.97d.	
Stores, (supplies, materials used for furnace repairs, tools, etc.)		6.13d.	
Fluedust (collecting and briquetting)		1.7 d.	
Water-supply (pumping, etc.)		2.37d.	
Motive-power (steam-generation, blowers, slag-elevating, electricity, etc.)		1s. 3.12d.	
General expenses (administration, laboratory, maintenance of buildings, telephones, electric light, etc.)		3.98d.	
Total		6s. 5.13d.	

In each item the cost of repairs and maintenance, including renewals, transportation, etc., is included, and only in the case of the furnace manipulation is the respective outlay (stores — chiefly water-jackets and tools) separately mentioned. The items composing the cost of smelting, taken in the restricted sense of the work immediately attaching to the operation, are bracketed together, and total 3s. 6.11d. The figures are based on the general long ton of ore, i.e., Mount Lyell ore, North

Mount Lyell ore and purchase ores, together, exclusive of metal-bearing fluxes, the respective tonnage having been 203,992 tons.

A similar dissection of the bessemerizing costs is the following, representative of the present, covering the period of the last half-year, and expressed in terms of a penny per pound of fine copper produced in the operation.

COST OF CONVERTING.

Remelter labor	0.0224d.	0.2262d.
Remelter coke	0.0632d.	
Converting labor	0.0704d.	
Lining labor	0.0390d.	
Lining materials (clay, quartz, etc.)	0.0092d.	
Stores (supplies for remelting, converting and lining)	0.0141d.	
Fluedust (collecting, handling, clinkering, labor and stores)	0.0079d.	
Motive-power (labor, fuel and stores)	0.0958d.	
Dressing and sampling blister copper	0.0234d.	
Loading blister copper into trucks	0.0012d.	
General expenses (adminstration, laboratory, maintenance of buildings, etc.)	0.0242d.	
Total	0.3708d.	

The cost of bessemerizing in the more restricted sense, leaving out motive-power and the other items below it is 0.2262d. per pound of fine copper. The cost of motive-power is fairly high under the local conditions. It will always vary with the locality, so that it is not intrinsically a metallurgical operating cost.

In judging these local costs it should be borne in mind that Tasmania is not a cheap place for industrial operations. Its geographical isolation and remoteness, in conjunction with purely local factors, raise the cost of even common-place articles and everyday trade products to extraordinary figures.

The mining, smelting and railway departments deal with each other on independent financial footings, but collectively account to the head-office in Melbourne. The mine department is not given credit for the value of the metals it supplies.

The wages paid in Tasmania only, since 1894, amount to £3,182,000. For 1907 the wages paid in connection with the mining, smelting and railway operations totalled £294,974.

Superphosphate Works.—Since 1905 the Mount Lyell company has also been manufacturing superphosphate and other artificial fertilizers, using pyrites from the Mount Lyell mine for acid making. The phosphate rock comes from Ocean Island in the Pacific. A large plant in two units, following American lines of construction, was erected for the purpose, at Yarraville, near Melbourne, Victoria, and a similar new works is just being completed at Port Adelaide, South Australia. Commercial reasons of a purely local nature, but of an emphatic conclusiveness, argued against the utilization of the blast-furnace fumes for acid-making at the reduction works.

General.—According to local custom, the mining and smelting returns, i.e., tons of ore treated, and the metal output, are published four-weekly, also, fortnightly, a progress report of the mining, etc., performed. This publicity of operations and results necessitates systematic work and records, and, although invested with drawbacks, accords with the requirements of the shareholder. The affairs of the company are in the hands of a Melbourne and a London board of directors. The capital of the company is £1,300,000 in 1,300,000 shares of £1 each, of which 1,200,000 shares have been issued, fully paid up. The dividends paid by the Mount Lyell company since July 1, 1897, to the end of 1907, amount to £2,106,547. For the year ended June 15, 1907, i.e., during the recent high price of copper, they were £405,000.

The number of men employed on an average, in 1907, was 870 at the mines. The number of smelter operatives and artisans was 1050 (including 140 firewood-getters). The two railway lines operated by the company, inclusive of all hands on the tram-service, are manned by 160 men. The grand total is thus 2080 employés, or about 30 per cent. of the number of persons engaged in the mining industry in Tasmania.

COPPERAS.

The makers of iron and steel sheets and wire continued to produce practically all of the copperas (sulphate of iron) as a by-product in acid cleaning. The United States Steel corporation was the largest producer in 1907, as it was in previous years, furnishing more than 90 per cent. of the copperas made in the United States. Wickwire Brothers, of Cortland, N. Y., did not operate their plant in 1907, but resumed production early in 1908. Other producers were the Pennsylvania Salt Manufacturing Company, of Philadelphia, Penn., the E. I. Dupont de Nemours Powder Company, of Wilmington, Del., and the Stauffer Chemical Company, of San Francisco, Cal.

PRODUCTION OF COPPERAS IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Year.	Short Tons.	Value.	Year.	Short Tons.	Value.
1893	16,000	$95,440	1898	11,285	$ 58,105	1903	20,240	$121,440
1894	(a)	(a)	1899	13,770	108,508	1904	16,956	118,692
1895	14,118	69,846	1900	12,374	96,517	1905	21,103	147,721
1896	11,170	52,662	1901	23,586	112,336	1906	22,839	228,390
1897	11,924	56,565	1902	19,784	118,474	1907	26,771	294,481

(a) Statistics not collected.

The above table includes only the copperas recovered as a by-product, and disregards the production as an intermediate product in the manufacture of venetian red by certain paint-makers.

Market and Prices.—The New York market remained unchanged throughout 1907. Most of the product was disposed of in the regular channels and the surplus supply was too small to effect any variations in price. Quotations remained stationary on the basis of 55c. per 100 lb. for copperas in bulk, 65@75c. for copperas in barrels and 60@70c. for copperas in bags, depending upon quantity and terms of sale. The average prices received by the producers ranged from $10@12.50 per 2000 lb.

CORUNDUM AND EMERY.

By Claude T. Rice.

Practically all the corundum and emery used in the United States is imported. The emery is imported crude as ballast from Turkey and Greece, so as to escape the duty of 1c. per ℔. on ground or manufactured emery. Thus the emery industry in the United States has become mainly a manufacturing industry. Corundum is imported mainly from Canada in pulverized form. Owing to its similarity to emery, which is specifically mentioned in the tariff law, a duty of 1c. per ℔. is charged. The importers have been fighting this ruling and trying to import corundum as "sand, crude or manufactured," which is free of duty, but on appeal the lower court was sustained in its decision that corundum should be taxed the same duty as emery. The accompanying table shows the production and trade in corundum and emery in the United States.

STATISTICS OF CORUNDUM AND EMERY IN THE UNITED STATES.

Year.	Production. (a)		Imports.				
			Grains.		Ore and Rock.		Other Mfrs.
	Short Tons.	Value. (b)	Pounds.	Value.	Long Tons.	Value.	Value.
1897	2,193	$111,810	520,095	$20,022	5,209	$107,644	$2,211
1898	3,742	207,430	577,655	23,320	5,547	106,269	3,810
1899	3,970	228,570	728,229	29,124	7,435	116,493	11.514
1900	5,030	247,100	661,482	26,520	11,392	202,980	10,006
1901	4,305	146,040	1,086,729	43,217	12,441	240,856	10.926
1902	4,251	104,605	1,665,737	49,107	7,157	151,959	13,776
1903	2,542	64,102	3,595,230	100,272	10,884	194,468	17,829
1904	1,932	57,235	2,281,193	109,772	7,054	138,931	11,721
1905	(c)2,315	19,677	3,209,914	143,729	11,072	185,689	17,996
1906	(c)2,147	22,780	4,655 168	215,357	13,840	286,386	19.105
1907	(c)10,69	12,294	4,282,228	186,156	11,235	211,184	15,282

(a) Statistics of the United States Geological Survey for 1901–1903. (b) Values have not much significance owing to the wide variation in the quality of the materials combined in the totals. (c) Emery only.

The Corundum and Emery Industry in the United States.

Mining.—The production of corundum in Georgia and North Carolina has practically ceased, the visible supply being almost exhausted. Montana has recently produced some good corundum. Emery is mined near Peekskill, Westchester county, N. Y., and at Chester, Mass. In

1906 a small quantity was produced in Kansas. In 1907 no emery was produced in North Carolina.

The emery coming from Peekskill, N. Y., is mostly of poor grade and does not compare well in quality with Turkish or Grecian emery. The emery produced at Chester is still of fine quality; in fact in its prime Chester furnished probably the best emery, but of late years the mining has been done intermittently and mostly from the narrow parts of the old stopes where the mineral is "petering" out.

At Chester, the Ashland Emery and Corundum Company continues to pick over the old dumps in the summer, and to mine a small quantity of emery from the old pockets in the winter. No new pockets have been found, as no exploratory work has been done and the narrow parts of the stopes, where the small tonnage now mined is obtained, have not widened out. No work has been done at the Wright mine. The mill runs part of the time on emery from the dumps and the mine, and part of the time on alundum which the Ashland company is crushing for the Norton company, of Worcester.

Grinding.—During 1907, the industry went through the same vicissitudes as did other manufacturing enterprises. In the first half of the year trade was flourishing, but during the last three months it fell off rapidly owing to the failure to receive the usual orders for wheels.

The Ashland Emery and Corundum Company is the largest grinder of emery in the United States; it has plants in operation at Chester, Mass., at Perth Amboy, N. J., and at Easton, Penn. Other grinders are the Abrasive Mining and Manufacturing Company, at Chester, Mass., and Plymouth, Ind.; the Hamilton Emery and Corundum Company, at Chester, Mass.; and the Oriental Emery Company, at New Haven.

During 1907 the Abrasive company doubled the capacity of its plant at Plymouth. The Hamilton company at Chester is building a new mill which will have a capacity of 8 to 10 tons of emery per 10 hours. This will give this company a total capacity of 13 to 15 tons per 10 hours. In fact during 1907 the increase in capacity of mills belonging to what is known in the trade as the "independent" companies was a noteworthy feature. The Pittsburg Emery Wheel Company, whose plant is at Pittsburg, Penn., began operations in 1907. Besides the companies mentioned above, many of the companies manufacturing emery wheels grind a considerable part of the emery that they use.

Corundum and Emery in Foreign Countries.

Canada.—As will be seen from the accompanying table, the production of corundum in Canada declined slightly in 1907. Canada is now the chief source of pure corundum. The productive district is in the vicinity of Craigville, Ontario, where the Canada Corundum Company is the largest

operator. The Abrasive company, of Chester, Mass., is interested in this field.

THE CORUNDUM INDUSTRY OF CANADA.

Schedule.	1900	1901	1902	1903	1904	1905	1906	1907
Production, tons	60	534	1,137	1,119	1,665	1,644	2,274	1,892
Value	$6,000	$53,115	$83,871	$87,600	$150,645	$149,153	$204,973	$177,922
Number of men	35	68	95	186	202	....	....	....
Wages paid	$10,000	$30,406	$34,674	$106,332	$ 139,548	....	....	....

Greece.—The island of Naxos is the oldest known source of emery. From it was obtained all the emery used prior to 1847, in which year the Turkish deposits near Smyrna were discovered. The deposits in Asia Minor and in Naxos are the most important sources of supply. The emery much resembles fine-grained crystalline hematite, and has been frequently mistaken for that mineral.

All the emery shipped from Naxos is inspected by government officials since the establishment in 1900 of the government depot at Syra, the port from which the ore is shipped. During this time 47,635 metric tons have been shipped, of which 502 tons, or a little over 1 per cent., has been condemned as foreign material, and 460 tons as second-grade emery. The sale of Naxos emery during the last 10 years has been as follows: In 1897, 3125 metric tons, valued at £13,515; in 1898, 4500 tons, valued at £19,170; in 1899, 5139 tons, valued at £21,848; in 1900, 6023 tons, valued at £25,618; in 1901, 6080 tons, valued at £25,840; in 1902, 4315 tons, valued at £18,280; in 1903, 5813 tons, valued at £24,763; in 1904, 6353 tons, valued at £27,064; in 1905, 6395 tons, valued at £27,179; in 1906, 8030 tons, valued at £34,208; in 1907, 10,982 tons, valued at £45,505; total, 66,755 metric tons, valued at £282,990.

In 1906, 4200 metric tons of emery were shipped to Holland, 1900 tons to England, 1050 tons to the United States, 730 tons to France, 100 tons to Russia, and 50 tons to Austria.

Turkey.—The emery occurs as irregular deposits in limestone. At present most of the deposits that are being worked are situated about 200 miles southeast of Smyrna. The emery from the mines nearer Smyrna is generally better for polishing, while that coming from the more distant mines is better for the wheel trade. The old mines near Smyrna are practically exhausted, and only those at Cosbounar and Azizieh, on the Aidin Railroad, 50 miles south of Smyrna, are working. Therefore the mine owners have begun to develop the properties in the more remote districts. The emery is brought to the sea coast or to the railroad station on camels and donkeys. Owing to the high cost of this transportation and the heavy

government tax, the deposits in the remote districts at present only pay when worked by open-cut methods. In fact all the mining is carried on in a very primitive way, and the emery is quarried without the use of explosives. In the largest producing districts, Kuluk, Moulah and Kuyudjak, the labor of mining the emery costs from $1.25 to $1.75 per ton of mineral mined, while at Cosbounar, Azizieh and Sarakeuy it costs from $6 to $7 per ton.

The mines most distant from Smyrna are those near Hierapolis and Aphrodisia, which produce excellent ore for the wheel trade. The emery from the various mines above mentioned fetches $17 to $19 per ton f.o.b. Smyrna. The emery mined at Kuluk and shipped from that port on the gulf of Mandalia brings $13.50 per ton f.o.b. Kuluk. The Turkish emery mines ship about 20,000 tons per annum. About 60 per cent. of this quantity is shipped to the United States and about 40 per cent. to European markets. The production of emery has been greatly stimulated by the new mining laws, which are much more favorable than the old laws.

Classification, Properties, Methods of Milling and Uses of Corundum and Emery.

Classification and Properties.—When aluminum oxide occurs in nature in clear, transparent crystals, it is a gem mineral, called, according to color, sapphire, ruby, oriental topaz, oriental emerald, or oriental amethyst. When opaque the aluminum oxide is called corundum. Corundum occurs in crystals, masses, or grains; its cleavage is rhombohedral so that when crushed the grains have sharp cutting edges. In hardness corundum is next to the diamond, so far as natural minerals are concerned, but it is considerably inferior to carborundum, an artificial product. Corundum occurs in limestone, peridotite, granite, gneiss, and other crystalline rocks, and is usually associated with chloritic minerals, and frequently with magnetite, feldspar and the micas.

When iron oxide is intimately mixed with the aluminum oxide, the mineral is called emery. Emery is usually black, but sometimes it is a dark gray. The percentage of iron in emery varies greatly, and this decreases the hardness of the mineral. Emery, which is merely an impure corundum, occurs in the same kind of rocks in which corundum is found. Mica is often associated with emery.

Corundum and emery are graded according to the mesh of the screen through which they pass when they are milled. The screens generally used for crushed emery and corundum are of 4, 6, 8, 10, 12, 14, 16, 20, 24, 30, 36, 40, 46, 54, 60, 70, 80, 90, 100, 120, 150, 180, and 200-mesh. Flour, which is obtained by settling the overflow from the washers, is divided

into 1F, 2F, 3F, 4F, 5F, and 6 F. The sizes generally used for the wheel trade are from 16- to 150-mesh.

Emery depends for its abrasive action upon the amount of corundum that it contains. However, the chemical analysis is not a good indication of its quality. In the trade any standard method for determining aluminum oxide is used. As most of the mineral at present employed in the United States comes from Turkey and Greece, and as this emery has few impurities other than magnetite and mica, little precaution is needed in analyzing the emery rock to guard against wrong results because of the presence of aluminum oxide in associated minerals. The value of emery and corundum depends far more upon their physical characteristics than upon chemical analysis, and therefore purchasers of emery test it for its abrasive quality and its capability for use in wheels. A simple method of testing the abrasive quality of powdered corundum or emery is to rub the powder between two weighed pieces of plate glass for a certain length of time. The amount of attrition when compared with that produced on the same plates by a powder of known reliability is a good indication of the abrasive quality of the sample tested.

Methods of Milling.—The methods of milling emery and corundum are simple, because corundum is much heavier than most of the impurities except magnetite. The mineral is crushed to the approximate size almost always by rolls, but in some cases ball mills have been used. The crushed mineral goes to the washers, a type of edge runner in which the runners are made of beech or maple, for only a mulling effect is desired so as to remove adhering impurities. Water is fed continuously to the washers, and the impurities are carried to the overflow, while the heavier minerals, magnetite and the corundum or emery, settle to the bottom whence from time to time they are shoveled to the drier. The overflow is taken to the "floats," or settling launder, where the emery flour is obtained. The settling launder is simply a series of concrete troughs placed side by side, so as to save room, and connected together. The launder is just wide enough so that a square-point shovel can be passed through it to remove readily the settled emery.

The flour and the abrasive from the washers have to be dried separately, the flour on a plate drier, and the other in a coil drier, the emery being placed on coarse screens so that, as it becomes dried, the emery falls through the screen.

The graders are simply trays bumped by cams. Generally the powdered abrasive is divided at 36-mesh; the oversize goes to trays on which wire screens are used, while the undersize goes to screens made of silk. The graded abrasive is then sent to the cleaners. These are of two types, the blower and the suction cleaner. The blower cleaner is simply a vertical tube through which the abrasive drops in an upward current of air. This

removes the light impurities. The suction cleaner employs a gentle current of air, produced by suction, through which the abrasive is passed; this raises any light impurities and carries them away, but the cleaning action is not as good as in the blower. The sizes above 150-mesh are therefore treated in the blower, but sizes below 150-mesh must be treated in the suction cleaner, as otherwise there would be too much of a loss. In some mills the abrasive is then given a further cleaning by passing it under magnets. Abrasive flour is not cleaned after being dried.

Uses.—Corundum is used mainly for the manufacture of vitrified wheels. Emery is also used for making vitrified wheels, but on account of the presence of iron in the emery, which increases the fusibility of the aluminum oxide, only the best grades can be used for this purpose, as the wheels have to be baked at a temperature of about 3000 deg. F. Generally the Naxos emery is better than the Turkish for the vitrified trade, although some Turkish emery is adapted to this use, and therefore Naxos brings a higher price in the market than the ordinary Turkish emery.

For polishing purposes emery flour is better than corundum because of the fact that it does not scratch so much as corundum. In the smaller sizes, where a smoothing or polishing rather than an abrading effect is desired, Turkish emery is considered by the trade to be better than Naxos, and as a result it is difficult to dispose of the finer sizes of Naxos, therefore the grinders of Naxos emery have to increase the price of the coarser sizes of Naxos on this account.

In the coarser sizes, emery and corundum come into competition with the artificial abrasives, such as carborundum, alundum (artificial corundum) and adamite. Carborundum and adamite are used only in coarse grinding owing to their great tendency to scratch. They cut more rapidly than corundum and for that reason are preferable to corundum for certain uses.

CRYOLITE.

The importation of cryolite into the United States in 1907 amounted to 1238 long tons, valued at $28,902. There were no important developments at the mines in Greenland, which continue to be the only commercial source of this mineral. The consumption of cryolite in the United States and elsewhere decreased in common with things generally in 1907, because of the disturbed industrial conditions. The imports into the United States during recent years are given in the following table:

IMPORTS OF CRYOLITE IN THE UNITED STATES.

Year.	Long Tons.	Value.	Av. per Ton.	Year.	Long Tons.	Value.	Av. per Ton.
1898	6,201	$88,501	$14.27	1903	7,708	$102,879	$13.35
1899	5,879	78,676	13,38	1904	959	13,708	14.30
1900	5,437	72,763	13,37	1905	1,600	22,482	14.05
1901	5,383	70,886	13.17	1906	1,505	29,683	19.72
1902	6,188	85,650	13.84	1907	1,284	28,902	22.51

J. H. Cameron, of St. Peters, Colo., reported striking a vein of cryolite at a depth of 110 ft. in sinking a shaft to mine silver-lead ore. Examination of the mineral by the U. S. Geological Survey showed it to be a massive alteration product. There was no production from this source in 1907, but the owners are seeking a market for their mineral, which they believe they possess in considerable quantity.

Cryolite continues to be used chiefly in the chemical industry, particularly by the Pennsylvania Salt Manufacturing Company. It used to be employed by the manufacturers of aluminum as the electrolyte (fused), but now they employ artificial cryolite, obtained with fluorspar.

FELDSPAR.

In the United States feldspar is obtained chiefly from Pennsylvania, Maryland, Connecticut, Maine and New York, named in the order of their importance as producers. Although deposits of feldspar occur in various other parts of the country, they have not been exploited to any great extent, either on account of factors preventing profitable mining, or the presence of such impurities as iron oxide, black mica, garnet, black tourmaline and quartz, which make the material undesirable for the manufacture of pottery. Statistics of production during recent years are given in the accompanying table.

FELDSPAR IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year.	Crude.		Ground.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1901	9,960	$21,699	24,781	$198,753	34,741	$220,422
1902	21,870	55,501	23.417	194,923	45,287	250,424
1903	13,432	51,036	28,459	205,697	41,891	256,733
1904	19,413	66,714	25,775	199.612	45,188	266,326
1905	14,517	57,976	20,902	168,181	35,419	226,157
1906	39,976	132,643	32,680	268,888	72,656	401,531
1907	....	....	....	....	....	....

(*a*) Statistics reported by the U. S. Geological Survey.

Market and Prices.—In January and February, 1907, best ground feldspar brought from $12 to $15 per short ton. During the remainder of the year the best grade sold for $14 per ton, other grades bringing from $7 upward, depending on quality. During January and February, 1908, $15 was again paid for best ground feldspar. The price of crude feldspar varied greatly during 1907. From $3 to $4 per ton, and in some cases a little more, was paid, the price depending on the quality of the product.

Feldspar Mining in the United States.

Connecticut.—The feldspar industry is an old one in this State, operations having first started about the year 1835 and continued, with intervals of quiet, ever since. The principal deposits operated at the present time are situated in Hartford and Middlesex counties. The spar occurs

in pegmatite dikes, associated with quartz, mica, tourmaline, garnet, etc. The Consolidated Feldspar Company, organized during 1907, leased the Johnson property at White Rock, near Middle Haddam. Early in 1908 work was begun on a grinding plant. This company will operate on a large scale. There was great activity in both mining and milling operations in the vicinity of New Haven and South Glastonbury.

Maryland.—The feldspar industry centers in Baltimore county, although there are some operations in Cecil, Howard and Harford counties. The feldspar occurs in intrusive dikes and the deposits are generally similar to those of southeastern Pennsylvania. Great activity was displayed in mining operations during 1907 and the production increased almost four-fold over that of 1906; however, in 1906 the production was far below normal. The deposits near Henrytown were not worked during 1907 but operations will probably be resumed in 1908.

New York (By D. H. Newland).—The feldspar quarries near Bedford, Westchester county, were worked during 1907 by P. H. Kinkel's Sons who also made a small output from the new Hobby quarry in the town of Northcastle. Most of the product from this section is sold to potteries. There were two quarries in the Adirondacks that were active, one situated near Batchellerville, Saratoga county, and owned by the Claspka Mining Company, and the other at Rock pond, Essex county, formerly belonging to the International Mineral Company, but now operated by the Barrett Manufacturing Company. The former yields pottery material, while at Rock pond the feldspathic rock is crushed and shipped unsorted for roofing purposes as a substitute for common gravel. The product is said to be superior to gravel owing to the fact that the feldspar with its smooth cleavage planes has greater adhesive properties when applied to tarred surfaces. Another operation, started in the early part of 1908, is that of the Crown Point Spar Company, with property near Crown Point, Essex county. The quarry is opened on an extensive body of pegmatite outcropping a short distance south of Crown Point and one-third mile west of the Delaware & Hudson Railroad. A large mill has been erected close to the railroad, to which the rock is conveyed by a cableway. The installation is very complete and will enable the company to supply feldspar in any marketable form. The separation of the feldspar from the accompanying minerals (quartz, mica and tourmaline) is effected entirely by mechanical means after crushing, whereas in other quarries the usual method is by handcobbing. The production of feldspar for pottery uses in 1907 was 5023 tons, valued at $25,923.

Pennsylvania.—Chester county is the chief source of supply of feldspar in this State, although some mining is done in Delaware county. In the southwestern part of Chester county the deposits of feldspar seldom carry quartz, and the principal iron-bearing mineral is green hornblende; garnet

occurs only occasionally. On account of much weathering near the surface some of these quarries have produced kaolin. The principal operations are in the neighborhood of Brandywine Summit. Here the Johnson property, which is under lease to the Brandywine Summit Kaolin and Feldspar Company and is being operated in conjunction with its other properties, produces alone about 3000 tons per year.

Virginia.—The Pinchback mines, about two miles northeast of Amelia Courthouse, in Amelia county, were operated on a small scale during 1907. Besides feldspar, these mines produce mica and kaolin, the latter resulting from decomposition of the feldspar.

Composition and Uses of Feldspar.

Feldspar is a double silicate of either sodium, potassium or calcium, known respectively as albite, orthoclase and anorthite. Although these are the principal classifications, feldspar occurs as various combinations of the elements mentioned. Orthoclase is the most abundant and occurs in the purest form.

Feldspar is used largely in the manufacture of pottery, mainly as a constituent of the body and glaze in porcelain, white ware, and various sanitary and enameled wares. When used as a body in these wares from 15 to 35 per cent. of feldspar is employed; for glaze from 30 to 50 per cent. is used. Feldspar also finds application in glass making, in dentistry and in the manufacture of some polishes and soaps.

The United States Department of Agriculture has been, and is still, conducting experiments looking to the utilization of potash feldspars for fertilizing purposes. Potash is an important plant food which has heretofore been supplied in the form of easily soluble potash salts. The following is an abstract of the conclusions arrived at, based on experimental work up to April, 1907 (Bulletin No. 104, U. S. Dept. of Agriculture. Bureau of Plant Industry): No claim has been made that ground feldspar is an efficient substitute, under all circumstances, for potash salts. The question is still open, and systematic and long-continued experimentation is the only possible method of obtaining conclusive information on the subject. The evidence so far obtained appears to indicate that under certain conditions and with certain crops feldspar can be made useful if it is ground sufficiently fine. On the other hand, it is highly probable that under other conditions the addition of ground feldspar to the land would be a useless waste of money. At the present stage of the investigation it would be extremely unwise for anyone to attempt to use ground rock, except on an experimental scale that would not entail great financial loss.

The subject must be approached conservatively, with due regard to business economy. Sensationalism and exaggeration invariably do harm. It is extremely unlikely that ground rock will ever entirely displace the

use of potash salts, for its availability must inevitably depend upon many modifying conditions, such as the nature of the soil, the amount of moisture present, the character of the other fertilizers used, and the varying root action of different crops. With tobacco the results so far obtained have been encouraging but it is possible that this plant, which is a voracious feeder, can make use of the potash in fine-ground feldspar to a greater extent than other fast-growing crops, such as potatoes and the cereals, some of which mature in practically 60 days and must therefore find their plant food in a highly available condition.

Preparation for Market.—The methods employed for grinding feldspar are practically the same in all parts of the United States where the material is mined. The general scheme is as follows: The material is first crushed in what is called a chaser mill, consisting of buhrs $3\frac{1}{2}$ ft. in diameter and about 1 ft. thick. After leaving the buhrs the material is screened, overs being returned to the chaser mill and the fines going to ball mills for final grinding. The ball mills are of the cylindrical type and are lined either with wooden blocks or highly silicious brick; the pebbles are either Norway or French flint, 2 to 3 in. in diameter. After four to six hours grinding in these mills the feldspar is reduced to a fineness of at least 200-mesh, when it is ready for shipment.

FLUORSPAR.

At the beginning of 1907 the conditions in the fluorspar industry were exceedingly promising and it was expected that the production for the year would greatly exceed that of 1906. The failure to realize this expectation was due, not to the inability of the producers to furnish the material, but to a greatly lessened demand during the latter part of the year. Fully 80 per cent. of the fluorspar produced in the United States is consumed in connection with the manufacture of iron and steel, and the financial depression, which had a marked effect on those industries, naturally curtailed the demand for fluorspar. As may be noted in the accompanying table, the production during 1907 showed a slight increase over that of 1906.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES.
(In short tons.)

Year.	Tons.	Value.	Per Ton.	Year.	Tons.	Value.	Per Ton.
1898	12,145	$86,985	$7.16	1903 (a)	42,523	$213,617	$4.28
1899	24,030	152,655	6.35	1904 (a)	36,452	234,755	6.44
1900	21,656	113,430	5.24	1905	39,600	232,452	5.87
1901	19,586	113,803	5.81	1906	34,683	201,481	5.78
1902	48,018	271,832	5.19	1907	36,350	202,736	5.58

(a) Statistics of the U. S. Geological Survey.

The principal fluorspar deposits of the United States are in Caldwell, Livingston and Crittenden counties, Kentucky, and Hardin and Pope counties, Illinois; however, promising prospects exist in neighboring counties, where considerable development work was done during 1907. Extensive deposits, from which some shipments have been made, also exist in Colorado and Tennessee. Arizona, which has reported the production of varying quantities of fluorspar in recent years, has dropped out of the producing list.

Southern Illinois contributed over 21,000 tons to the total output in 1907, nearly 15,000 tons coming from western Kentucky. The principal operating companies in the Illinois field are, the Fairview Fluorspar and Lead Company, and the Rosiclare Lead and Fluorspar Company, of Rosiclare; Pierce Brothers and the Crystal Fluorspar Company, of Golconda.

The principal operators in Kentucky are the Kentucky Fluorspar

Company, Marion; Pope Mining Company, Louisville; American Fluorspar Company, Paducah, and the Sunny Brook Lead and Fluorspar Company, Marion. The Marion Zinc Company, of Marion, which reported a small production of fluorspar in 1906, produced barytes during 1907. The Pope Mining Company, which produced ground fluorspar during 1907, will also mine and market No. 1 lump during 1908. This company began production in April, 1907. The American Fluorspar Company reported, in February, 1908, a rapidly increasing demand for mineral better than 90 per cent. calcium fluoride for both open hearth and cupola use.

Illinois (By H. Foster Bain).—The fluorspar deposits of Illinois are directly related to those of Kentucky. The character of the ore is the same as is also the mode of occurrence, the deposits being along faulting fissures in the Mississippian limestones and near dikes of basic igneous rocks. As heretofore, the major production continues to come from Rosiclare, where the mills of the Rosiclare and the Fairview companies are supplied from extensive deposits worked to depths of more than 300 ft. Less important amounts of spar are marketed from Golconda, Elizabethtown and Cave-in-Rock. A considerable amount of quiet development is going on but there are large areas demanding much more systematic work than any yet done.

Market Conditions and Prices.—There was little change in the price of fluorspar during 1907, quotations ruling about as follows: Domestic f.o.b. shipping port; lump, \$8@10 per short ton; ground, \$11.50@13.50; gravel, \$4.25@4.50. Foreign crude, ex. dock, brought from \$8@10 per ton. The use of imported English spar has increased in the Pittsburg district, while in the South the domestic material seems to have the preference.

FLUORSPAR OUTPUT OF THE PRINCIPAL PRODUCING COUNTRIES.
(In metric tons.)

Year.	France.	Germany.	Spain.	United Kingdom.	United States.
1897	2,722	23,232	2	302	3,973
1898	3,077	23,787	5	57	11,021
1899	5,140	24,306	310	796	21,806
1900	3,430	30,310	4	1,471	19,646
1901	3,970	28,741	*Nil.*	4,232	17,768
1902	2,650	(*a*)14,177	93	6,388	47,190
1903	2,447	(*a*)13,028	4,000	12,102	38,577
1904	2,047	(*a*)13,540	(*b*)	18,451	33,069
1905	2,434	(*a*)15,019	(*b*)	38,606	35,299
1906	4,218	(*a*)15,493	70	36,860	28,657
1907	(*b*)	(*a*)16,624	(*b*)	40,873	32,969

(*a*) Exports. German statistics no longer report production. (*b*) Not reported.

FULLER'S EARTH.

In the United States fuller's earth is widely distributed and in large quantities. The deposits which have received the greatest attention occur in Florida, Georgia, Alabama, Texas, South Carolina, Arkansas, Colorado, South Dakota, California and New York. The production and imports are shown in the accompanying table.

STATISTICS OF FULLER'S EARTH IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Year.	Production.		Imports.	
	Sh. Tons.	Value.	Sh. Tons.	Value.		Sh. Tons.	Value.	Sh. Tons.	Value.
1898	15,553	$87,365	9,355	$81,044	1903	20,693	$190,277	17,100	$120,671
1899	13,620	81,900	11,558	69,460	1904	29,480	168,500	10,221	74.000
1900	11,813	70,565	9,154	64,790	1905	25,745	157,776	15,181	105,997
1901	14,112	96,835	12,061	80,697	1906	28,000	237,950	14,827	108,696
1902	14,100	109,980	15,135	102,580	1907	34,039	323,275	14,648	122,221

Market and Prices.—Fuller's earth is usually bought in powdered form, although occasional sales of lump are made. The price of lump was steady throughout 1907 at 80@85c. per 100 lb. in large lots. The powdered material fluctuated within narrow limits. In January the quotations were from 80c. to $1.25 per 100 lb. With the exception of a slight increase in June, the price from February to October was 85@90c. per 100 lb. For the remainder of the year the quotation was 80@85c. per 100 lb. These prices were for large lots and varied according to the size of the order, terms of payment and quality of the material.

Review by States.

Alabama.—The Standard Reduction Company, of Chicago, Ill., purchased 200 acres of land containing deposits of fuller's earth in Baldwin county, six miles from Hurricane, and completed a plant for mining and preparing the material for market. The present capacity is one carload daily.

Arkansas.—During the latter part of 1907 the Arkansas Fuller's Earth Company was incorporated for the purpose of working the deposits near Benton, in Saline county. The deposit extends over an area four miles in length and two miles in width. Development work has uncovered a

sufficient quantity of the earth to warrant the erection of a plant for mining and preparing the material for market. Plans are being made to produce about 30 tons daily; later it is expected that the tonnage will be increased.

California.—In Kern county, near Bakersfield, is an extensive deposit varying from 15 to 50 ft. in thickness. Some work has been done and the product used on the Pacific coast, but the deposits are rather remote from transportation.

Florida.—By far the greater portion of the production of fuller's earth comes from Florida, chiefly from the northwestern portion near Quincy, and from near Palmetto on the Manatee river, where a large and well-constructed plant is in operation, turning out the finished product at a low cost. At the Palmetto deposits, the product is directly loaded into vessels. The Florida output is used almost exclusively for bleaching petroleum, although a small proportion is employed on tallows and other low-grade oils and fats.

(By E. H. Sellards.) The first discovery of fuller's earth in the United States was made in Gadsden county, Florida, in 1893. The deposits have since been shown to occur in Leon and Liberty counties, adjoining Gadsden, and also to extend across the State line into Georgia. Elsewhere in Florida fuller's earth is known to occur in Alachua and Manatee counties. During 1907 fuller's earth was mined in two localities in Gadsden county and in Manatee county, in southern Florida. The earth is of a light buff to a light blue color. The overburden is of variable thickness, and mining is done chiefly with pick and shovel by the open pit method; steam shovels for the removal of the overburden are, however, in use. Upon removal from the pit the fuller's earth is dried and ground; it is then passed through sieves of proper mesh, sacked for market and numbered according to degree of fineness. The product is used principally in the United States, although a part of the 1907 product is reported as having been exported to foreign markets. The demand in general is good. The total amount of fuller's earth mined in Florida during 1907, as reported by the producers, was 24,148 short tons, valued at $235,443.

Georgia.—Fuller's earth is found throughout a broad belt of country extending from northwest Florida to the neighborhood of Georgetown, S. C. At Longstreet, Ga., the strata has a maximum thickness of 40 ft. The overburden varies from 3 to 40 ft. and the deposit is underlaid by marl, which accounts for the unusually large amount of calcium carbonate present. In Wilkinson county numerous outcrops occur. In Twiggs county a plant is located on a very large deposit and is at present producing on a small scale. At Groveton, about 15 miles from Augusta, there is another outcrop upon which a plant was erected, but owing to the presence of alum in the earth, production ceased.

South Carolina.—At Summerville an earth closely resembling the English product in color occurs. In the region about Gaston are a number of deposits. Near Sumter earths of excellent bleaching qualities have been uncovered and on the banks of the Black river, near Kingstree, a deposit about 25 ft. in thickness occurs. A plant is now operating on the last mentioned deposit.

Texas.—The Fort Worth Fuller's Earth Company, after a thorough investigation of its deposits near Fort Worth, let a contract during the summer of 1907 for the construction of a mill. Pending the completion of the mill, crude material was mined and shipped to other plants.

Toward the end of the year the Texas Fuller's Earth Company, of Dallas, completed the erection of its plant and prepared for active operations. The plant is situated on the Houston, East & West Texas Railroad, between Austin and Bonham.

Technology of Fuller's Earth.

Properties and Uses.—Fuller's earth varies greatly in its properties and no two deposits seem to be exactly alike. It is generally of a light brownish or grayish color, but may be any intermediate shade or white, cream, yellow, pink, or even black. Its specific gravity varies from about 1.75 to 2.5. Some varieties fall to a powder when placed in water and do not generally show plasticity. However, the Southern earths retain their shape in water.

Some deposits of fuller's earth yield a product which has the power to hasten greatly the oxidation by the air of oils with which it is mixed, and this action is so rapid and violent that the contents of filter presses have been known to burst into flame immediately on being opened to the air. This property is, of course, fatal to the use of that particular earth for bleaching an oxidizable oil. Most samples of fuller's earth when dry and unground adhere strongly to the tongue, this property being due to their ability to absorb water rapidly.

Fuller's earth was first used to remove grease from woolen cloth in the process of shrinking, or pulling, by means of moisture and heat; to this use it owes its name. The chief use, however, is in bleaching and decolorizing oils, fats and greases. In smaller quantities it is used in the drug trade as an absorbent similar to talcum powder, in which capacity it is much more efficient. Mixed with glycerol it is sold as a well-known proprietary medicine used for external application.

Mining and Preparation.—Fuller's earth is mined in much the same way as any clay bank. The overburden must first be removed and this is usually done with pick and shovel, or scrapers; the same tools are then used for digging the earth. Steam shovels may be used to advantage in this work.

The lumps of earth are first dried, during which process all except 4 or 5 per cent. of moisture is eliminated. This may be done under sheds, by exposure to the air, followed by heating at a low red heat on open iron plates; but when the material is handled on a large scale the kiln or rotary dryer is found more economical. If artificial dryers are employed it is highly important that a temperature but little above the boiling point of water be used, as removal of the constitutionally held water seems to lessen the bleaching power for animal and vegetable oils. Indirect firing of rotary dryers is ordinarily used, for the earth should not be mixed with any solid products of combustion; but when petroleum can be obtained, as is the case with some of the rotary dryers used in Florida, direct firing has many advantages, the hottest part of the dryer being the point where the moist earth enters.

When dry the earth must be ground and perhaps bolted. The grinding is usually done in buhr mills, or in ball mills, and the degree of fineness is regulated by the purpose for which the earth is to be used.

American vs. English Fuller's Earth.[1]—Although America furnishes all of the fuller's earth used on petroleum, it supplies but a small portion of that used on edible oils, the English earth being considered superior to any American earth yet marketed for this purpose.

For a fuller's earth to supplant the English earth in American practice, it must have the following properties, given in the order of their importance: (1) It must bleach as well as the English; (2) It must not cause the color of the oil to revert; (3) It must filter well; (4) It must absorb no more oil than the English; (5) It must not catch fire when removed from the presses; (6) It must give no permanent taste or odor.

To find all these qualities in one earth is no easy matter and all American earths put on the market have failed in one or more particulars, although the writer is confident that the fault has not always been inherent in the earth itself and there is no really good reason why a pound of earth should be imported.

Most American earths do not bleach as well as the English but the writer has had samples from three deposits that bleach decidedly better. The extent of the bleaching power is, of course, the first requisite. This bleaching power can be much lessened by the addition of lime water to the earth. On the other hand, every attempt to treat the earth so as to increase the bleaching power has met with failure.

The question of color reversion is deemed important by certain refiners and these claim that with certain American earths the color tends to come back to cotton seed oil in the treatment after bleaching. This is a quality inherent in the earth and if extensive would be fatal to its use.

---

[1] Abstract from paper on Fuller's Earth by C. L. Parsons, in *Journ.* Am. Chem. Soc., April, 1907.

If the earth does not filter well much time is lost, the filter presses become clogged, the oil is in contact with the earth longer than it should be and oil is mechanically lost. The difficulty is due to slimes and depends on the way the earth breaks down under the action of the mills. Fine grinding is essential to the full bleaching action but certain earths can be ground even to 200-mesh without forming slimes, owing to their peculiar cleavage, while others if ground fine enough to bleach properly will completely choke the presses. This is not due to the chemical composition of the earth as some have supposed, but rather to its mechanical condition.

Absorption of oil is not altogether a quality of the earth, for even with the English the amount left behind varies greatly with conditions and with the thoroughness with which the presses are blown out with steam. The amount left behind in the best practice is about 10 per cent., while the average will probably be from 15 to 20 per cent. Samples of spent English earth have been sent to the writer containing as high as 24 per cent. of oil and it can be seen at once that this is no small part of the cost of the process. American earths are reported to generally absorb too much oil. The reason is really the same as that which causes difficult filtration, for where slimes are formed they hold much oil mechanically.

Certain earths cause a vigorous oxidation to take place in the oil with which they are saturated on exposure to air. The action is so intense that the mass sometimes catches fire as soon as the presses are opened. At other times it takes place later in the waste piles of spent earth. The English earth is without this trouble and this property if well marked would naturally condemn any material. No reason can be assigned why one earth should have this property more than another. The fact simply remains and must always be reckoned with.

Most American earths give a decided taste and rancid or oxidized odor to oils, and this increases with the temperature used. With lard or lard oil this is a serious matter, but in the practice of most cottonseed oil refiners a subsequent treatment removes the taste and odor imparted by the earth. This action is in some way connected with the "acidity" of the earth and can be entirely overcome by neutralizing this absorptive power with lime, although the bleaching power unfortunately goes at the same time.

Besides these difficulties which must be met before an American earth can become successful, there is another within the refinery, and this is due to the fact that the workmen are always prejudiced against any new earth and any special change in their measures or methods. They accordingly lay all difficulties which may arise in the refining process to the innovation and condemn the earth to their superiors.

In spite of all these facts it is probable that imports of English earth will show a great falling off before many years have past.

GARNET.

The chief source of the supply of abrasive garnet in the United States is New York; Pennsylvania and North Carolina furnish a comparatively small proportion of the output. The production for the last 10 years is shown in the accompanying table.

PRODUCTION OF GARNET IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Value per Ton.	Year.	Short Tons.	Value.	Value per Ton.
1898	2,882	$82,930	$28.74	1903	4,413	$146,955	$33.30
1899	2,565	72,672	28.33	1904	2,952	89,636	30.36
1900	3,285	92,801	28.25	1905	3,694	114,625	31.01
1901	4,444	158,100	35.51	1906	5,404	179,548	33.22
1902	3,722	122,826	33.00	1907	6,723	209,895	31.22

New York (By D. H. Newland).—The abrasive garnet industry in the Adirondacks contributed an output of 5709 short tons, valued at $174,800, in 1907, a gain of nearly 1000 tons as compared with the production of 1906. During the first six months of 1907 the production was proportionately larger than in the latter half of the year, as the demand slackened with the general business depression and operations were curtailed to some extent. The garnet trade of the country is now concentrated more than ever in this region, since the mines elsewhere have not kept pace with its progress of late years. In the last decade the output from the Adirondacks has enlarged by about 300 per cent., while the total for the country has grown from 2261 tons in 1897 to 5404 tons in 1906, or about 150 per cent. The export trade in garnet is relatively small, but it has prospective importance.

The garnet is obtained from four localities. The North River Garnet Company operates on Thirteenth lake, Warren county, where an immense body of garnet rock, affording a quarry face 142 ft. high, has been opened. The material is crushed and concentrated mechanically. The company has added another unit to its separating plant and is in position to turn out from 8000 to 9000 tons of crystal garnet yearly if the trade should warrant it.

The Gore Mountain and Garnet Peak deposits near North river are

worked by H. H. Barton & Son Company, and the American Glue Company. The garnet is separated by hand cobbing and operations are carried on only during the open months of the year.

In the northern part of Essex county a deposit has been under development recently and made shipments in 1907. Most of the product is of the massive variety. The garnet occurs in lenticular and irregular bodies up to 40 ft. thick and is quite fine, except for inclusions of small pyroxene crystals. It generally shows little tendency toward crystal development or regular fracture. The property is owned by G. W. Smith, of Keeseville.

North Carolina.—Mining for garnets has come to be an important industry in Madison county, N. C., quantities of the mineral being shipped each month from Marshall and Hot Springs. Perhaps the richest garnet mine in western North Carolina is situated in what is known as the "Little Pine section." The garnets taken from the mine are used mainly for commercial purposes, but some handsome and clear stones have been found. Several clear garnets of two carats and one of five carats have been taken from this mine. Less than 1000 tons of garnet were shipped from the State in 1907, but the production for 1908 will probably show an increase.

GLASS.

By A. Van Zwaluwenburg.

The glass industry, which consumes enormous quantities of sand, lime and other mineral products, experienced unsatisfactory conditions during 1907. Dullness in the market, due largely to over-production during 1906, brought about a considerable curtailment of production in the early part of 1907. Labor troubles increased the difficulties, especially in the window glass industry. There was little change in prices throughout the year and weakness was apparent generally. Toward the end of the year only about 400 pots were in operation out of a total of 2800 employed in the United States in the manufacture of window glass, not including those controlled by the interests using machines for the manipulation of the glass.

The production of glass sand is one of the great branches of the domestic mining industry. According to the U. S. Geological Survey the output of this material increased from 1,060,334 short tons, valued at $1,107,730, in 1905, to 1,089,530 short tons, valued at $1,208,788, in 1906, an increase in quantity of 29,096 short tons and in value of $101,058. Pennsylvania ranks first in production and value of this material, the output in 1906 being reported at 342,967 short tons, valued at $510,910; West Virginia comes next, with an output of 158,093 tons, valued at $227,225; Illinois is third, producing 238,178 tons, valued at $156,684. Other States having an output of over $50,000 in value are, in order of rank, New Jersey, Ohio, and Missouri.

During 1907 a new discovery of glass sand of remarkable purity was reported from the Arbuckle mountains, Okla., by Prof. C. N. Gould, of the State University at Norman. The material which was analyzed in the university laboratory showed 99 per cent. silica with no trace of iron. The deposit is within a few miles of the recently developed natural gas fields of Oklahoma.

The Industry in Foreign Countries.—Reports from official sources in England, Belgium and Canada, show unfavorable conditions for glass manufacturers in those countries. High costs of raw materials, high wages and keen foreign competition is the universal complaint. The British export trade is greatly restricted by foreign tariffs, and foreign manufacturers have invaded the home markets, especially in the

supply of electrical glassware. The Belgian and Canadian industries suffered from loss of trade and general lack of demand.

New Uses for Glass.—Consular Agent Gustav C. Kothe, of Cassel, states that an architect of that city has been granted patents for the manufacture of glass telegraph and telephone poles. A company has been organized and a factory has been built at Grossalmerode, near Cassel. The glass mass of which the poles are made is strengthened by interlacing and intertwining with strong wire threads. One of the principal advantages of these poles would be their use in the tropical countries, where wooden poles are soon destroyed by the ravages of insects and where climatical influences are ruinous to wood.

According to *L'Electricien*, a Vienna firm has placed on the market brushes made from glass, which are to replace emery cloth for cleaning and polishing the commutators of dynamos and motors. These brushes are said to clean the commutators without scoring the metal, and their use avoids the inconveniences and dangers of emery cloth.

Hollow bricks and building blocks for house construction promise to be used extensively for novel and artistic effects. The first glass bricks were solid; the hollow glass blocks can be made at much less expense. The bricks were sealed hermetically when hot, and are placed in walls with a colorless mortar made of special glass.

The Glass Sand Industry in West Virginia.[1]

By G. P. Grimsley.

Glass chemically is a fused mixture of alkaline silicates, alkaline earths and metals, usually a sodium-lime silicate. The most important ingredient in quantity is sand, forming 52 to 65 per cent. of the mixture. It is therefore very important in any glass manufacturing center to have available a supply of suitable glass sand at a reasonable cost. West Virginia, on account of its cheap natural gas fuel, has become one of the important glass manufacturing States, and glass plants are scattered throughout its natural gas districts. At Martinsburg is found a deposit of high-grade limestone containing 98 to 99 per cent. calcium carbonate, which is there crushed to a fine flour to supply lime for these plants. There are at a number of places almost inexhaustible deposits of pure glass sand, and a prosperous glass-sand industry has thus been developed which supplies the home plants and those in other States.

A good glass sand must be free from iron, which gives to the glass a tinge of color more or less deep according to the amount of iron present. Clay tends to make the glass cloudy. The quality of the glass therefore depends largely on the quality of the sand used.

[1] Published with the permission of the State Geologist of West Virginia.

Berkeley Springs.—Berkeley Springs, the most important glass sand producing center in the State, is situated in Morgan county, 150 miles west of Baltimore and 210 miles east of Pittsburg. At this place and northeast to Hancock, the Oriskany white sandstone forms the summit of Warm Spring ridge for a distance of eight to ten miles as an almost snow white rock, and can be followed long distances to the north and south as a more impure sandstone. This ridge follows a direction north 25 deg. to 30 deg. east, with an elevation of 1000 to 1100 ft., descending to 700 ft. at its northern end. It separates the valley of Sir John's run at the west from that of Warm Spring run at the east. The town of Berkeley Springs is situated in the eastern valley at the foot of the ridge, and is reached by a branch of the Baltimore & Ohio Railroad.

The crest of this ridge and its upper slopes are composed of the white Oriskany sandstone, while the lower slopes are composed of the Hamilton and Chemung shales. The structure is part of a monoclinal fold.

The Hancock plant of the Pennsylvania Glass Sand Company, white started 40 years ago, was rebuilt in 1905. The sandstone is brought from the quarry to the mill down an incline tram road by a small engine. A large spring in the mountain back of the mill furnishes an abundant supply of clear pure water for the mills. The water used in these mills must be free from iron and mud and the plants of this whole area are supplied with an abundance of such water from numerous springs along this ridge. In the washing of the sand a very large quantity of water is required.

The sandstone at the Hancock plant quarry is very friable; it is readily crushed in an ordinary eight-foot wet pan under rotating wheels or mullers similar to those used in potteries. After crushing, the loose sand with its clay and other impurities is conveyed with water into a circular, rotating riddle or screen of 16-mesh, and then through a second similar screen. The coarsely screened sand is carried by a trough with a large surplus of water into the washing department, which consists of a series of inclined wooden boxes or troughs, 10 to 12 ft. in length. Each alternate trough contains a rotating screw conveyer with wide blades which carries the sand to the top where it is discharged into the next trough without a conveyer and is washed to the bottom, there passing into the next conveyer trough.

In the Hancock plant there are five conveyers to one wet pan and set of screens, and the sand leaves the fifth conveyer free from the clay. The loss of this clay is clearly seen in the change of color of the water-sand mixture in the different conveyers. The liquid has a muddy yellow color in the first conveyer, and somewhat lighter color in the second, while in the fifth the water is clear and the sand in it pure white. Since the rock from this quarry contains a considerable quantity of small flint

pebbles, the washed sand from the last conveyer is carried into a third circular screen which separates these pebbles from the sand. The white sand free from pebbles is carried into bins above the dryer. A Cummer hot-air dryer is used at this plant with coke fuel and a 22-ft. cylinder which has a capacity of 120 tons in 10 hours. The sand from the dryer is screened again through a 20-mesh screen and taken in elevators to the bins from which it is loaded in bulk in box cars. The float sand and clay from the washers is sold to fire brick factories, while the more impure sand is used by the railroad as engine sand. The small pebbles from the third screen find a ready sale for use in pebble asphalt roofing. At the Hancock plant there are two sets of wet pans, screens, and washing conveyers; in other words it is a two-mill plant with a daily capacity of 150 tons of No. 1 glass sand.

In an adjacent building is the pulverizer mill in which the sand is crushed to a flour of various grades of fineness. The sand is ground in two tube mills, 24 ft. long and 6 ft. in diameter filled one-third full of flint pebbles. The sand is ground so as to pass through 90-, 100- and 110-mesh screens, and is used in paint, scouring and polishing powders, and soap. One mill has a capacity of 18 tons of flour in 10 hours when crushed to pass a 110-mesh screen, or 10 to 12 tons with a 140-mesh screen. For certain special uses, such as silver polish, a 260-mesh screen is used.

The quarry, situated near the top of the mountain and 280 ft. above the floor of mill, shows the sandstone to be very friable, often breaking into sand under the sledge; pockets of loose sand are found. Very little blasting is required except to loosen large portions of the wall. The rock dips 45 deg. to the southeast, so that the blocks loosened by a shot or by wedge roll down the slope to the bottom of the quarry. The deposit was formerly worked by tunnels, but the workings are now in the form of a large open cut or pit 370 ft. long, 120 ft. wide and 100 ft. deep to the main floor of the quarry; a portion is now worked 30 ft. deeper. The stone lies under little cover and the total thickness as exposed is 160 ft.

The Berkeley plant of this same company is four miles up the valley from Hancock toward Berkeley Springs and has an equipment similar to that of the Hancock plant, except that the sand is dried in a steam dryer. The capacity is 180 tons daily and the quarry is on the ridge immediately back of the plant.

The West Virginia & Pennsylvania Sand Company has two plants, the first or West Virginia plant containing three mills, and the Pittsburg plant, nearer the town, two mills. The sand is stored in steel tanks with a storage capacity of 300 tons. The daily capacity of this company is 400 tons. The company also operates the second pulverizing plant in this field at the town of Berkeley Springs. The Speer White Sand

Company operates a plant of 140 tons daily capacity near the town, with equipment similar to that of the other plants; it is a double mill. The plant farthest south on the ridge is in the town, and is owned by the Berkeley Springs Sand Company.

There are seven plants in this field with 13 mills and a daily capacity of 955 tons of No. 1 glass sand. There are two pulverizing plants with three tube mills. The composition of the sand is shown by the following analysis: silica, 99.889 per cent.; alumina, 0.094; oxide of iron, 0.006; lime, 0.011.

The Silica Sand Company opened a sand quarry in 1904 in the Medina white sandstone on the west slope of Cacapon mountain, 1½ miles west of Berkeley Springs. The deposit is a very hard and compact white sandstone and the quarry rock is hauled down a 500-ft. incline to the mill at the side of the Baltimore & Ohio Railroad. The rock is crushed and washed in a manner similar to the methods used at the Berkeley plants, but it crushes to a sand mixed with much finely pulverized sand, almost a flour, and the work so far has not been as successful as had been hoped.

Holmes.—The plant of the White Rock Sand Company is situated three miles east of Terra Alta, in Preston county, on the main line of the Baltimore & Ohio Railroad, and was started in fall of 1906; the capacity was doubled in 1907. The sandstone is crushed in a Jeffrey pulverizer, then in an eight-foot wet pan, washed in three conveyer troughs, screened through a 16-mesh circular riddle, and dried in a steam dryer. Water for washing is supplied by Snowy creek, a stream of good size near the plant. The daily capacity is 200 tons of sand including glass and building sand. The quarry is connected with the mill by an incline track; the sandstone is 60 to 80 ft. thick, but the face selected for working is 22 ft. thick. This is the Pottsville or Homewood sandstone near the top of the Pottsville series of the Carboniferous, which is worked for glass sand at a number of places in this State and in Pennsylvania.

Deckers Creek, near Morgantown.—The works of the Deckers Creek Stone and Sand Company is situated on Deckers creek, at the town of Sturgisson, nine miles southeast of Morgantown on the Morgantown & Kingwood Railroad. The stone from the same horizon as the Holmes rock is broken in a crusher, ground in a wet pan, washed in eight conveyer troughs, and dried in a Ruggles-Coles rotary dryer 25 ft. long. The capacity of the plant is 90 tons of sand daily, and both glass sand and building sand are sold. The quarry is 620 ft. above the mill and a 35-ft. face is worked.

Craddock, Upshur County.—The plant of the Silica Sand Company is situated on a branch of the Baltimore & Ohio Railroad, on the Buchannon river, 47 miles from Weston, and was opened in 1905. The equipment

includes an Austin No. 5 crusher, an 8-ft. wet pan, five washing screw conveyers, 8- and 14-mesh screens, and a Cummer dryer. The daily capacity of the works is 120 to 150 tons of glass sand, and 15 tons of pebble and coarse gravel used for concrete. The sand is stored in a steel tank of 400 tons capacity and in bins holding 80 tons. The quarry is 420 ft. above the mill and is connected with it by an incline 1450 ft. long. The horizon of this sandstone is the Homewood, or the same as that at Deckers creek and at Holmes. The quarry face is 10 to 15 ft. high; the lower portion of the rock is a conglomerate. Two miles further south is the mill of the Enterprise Silica Sand Company, built in 1904, using a similar rock from the same horizon; but this plant was idle during 1907.

A 36-blower glass tank for window glass uses 150 tons of sand a week, or during a six months' season 3600 tons of glass sand and about 600 tons of limestone. Glass plants of this capacity are found at the following towns and cities in the State: Buchannon, Weston, Clarksburg, Fairmont, Salem, Grafton, Mannington, Cameron, Morgantown, Wheeling, Wellsburg, and Moundsville.

GOLD AND SILVER.

By Frederick Hobart.

The gold production of the United States was less in 1907 than 1906, showing a halt for the first time in five years. In 1906 the total was $94,373,800. In 1907 the preliminary figures of the Director of the Mint, supplemented by official returns from several States, show a total of $89,191,726. The aggregate, however, is still greater than that of any year prior to 1906. The three chief producing States show decreases;

PRODUCTION OF GOLD IN THE UNITED STATES. (a)

States	1904		1905		1906		1907	
	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value.
Alabama	1,417	$29,300	2,008	$41,500	1,137	$23,500	(a)	(c)
Alaska	450,091	9,304,200	722,090	14,925,600	1,033,537	21,365,100	882,923	$18,251,610
Arizona	161,761	3,343,900	130,203	2,691,300	132,891	2,747,100	122,849	2,539,516
California	924,427	19,109,600	928,742	19,197,100	911,041	18,832,900	841,454	17,394,363
Colorado	1,180,147	24,395,800	1,243,401	25,701,100	1,109,452	22,934,400	990,398	20,471,527
Georgia	4,688	96,900	4,586	94,800	1,146	23,700	(c)	(c)
Idaho	72,742	1,503,700	52,085	1,075,600	50,102	1,035,700	52,616	1,087,655
Maryland	116	2,400	818	16,900	....	....	(c)	(c)
Montana	246,606	5,097,800	236,541	4,889,300	218,752	4,522,000	203,482	4,206,345
Nevada	208,390	4,307,800	259,269	5,359,100	448,852	9,278,600	711,339	14,704,658
New Mexico	18,475	381,900	12,859	265,800	12,877	266,200	12,778	264,162
N. Carolina	5,994	123,900	5,990	123,900	4,397	90,900	(c)	(c)
Oregon	63,336	1,309,900	60,227	1,244,900	63,860	1,320,100	57,082	1,179,988
S. Carolina	5,892	121,800	4,601	95,100	3,609	74,600	(c)	(c)
S. Dakota	339,815	7,024,600	334,490	6,913,900	319,512	6,604,900	197,634	4,085,446
Tennessee	208	4,300	160	3,300	39	800	(c)	(c)
Texas	110	2,300	92	1,900	164	3,400	(c)	(c)
Utah	203,902	4,215,000	248,713	5,140,900	248,208	5,130,900	225,086	4,652,941
Virginia	184	3,800	242	5,000	498	10,300	(c)	(c)
Washington	15,862	327,900	17,900	370,000	4,983	103,000	5,000	103,350
Wyoming	793	16,400	1,147	23,700	276	5,700	(c)	(c)
Other States	....	....	....	....	....	....	12,101	250,165
Total	3,904,986	$80,723,200	4,266,120	$88,180,700	4,565,333	$94,373,800	4,314,742	$89,191,726
Porto Rico	....	....	....	....	....	....	59	1,219
Philippine Islands	....	....	....	....	....	....	280	5,766
Total	3,904,986	$80,723,200	4,266,120	$88,180,700	4,565,333	$94,373,800	4,315,081	$89,198,711

(a) The statistics in this table are as reported by the Director of the Mint, those for 1907 being the preliminary figures (subject to revision), except that later official figures for 1907 have been used, when available. (c) Included in other States.

in fact almost every State reports a diminished production, the only important gain being $5,426,058 in Nevada. The gold production for 1907 includes $1219 from Porto Rico and $5766 from the Philippine Islands; the first time anything from these outlying possessions has been reported.

For the first half of 1907, the output from the mines of Utah, Idaho, Montana and Arizona indicated increased yields for those States, but the sudden drop in the value of the ores which give gold as a by-product,

PRODUCTION OF SILVER IN THE UNITED STATES. (*a*)

	1904.		1905.		1906.		1907.	
	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.
Alabama	200	$116	300	$181	100	$67	(*d*)	(*d*)
Alaska	210,800	122,264	169,200	102,116	203,500	135,920	148,609	$97,082
Arizona	2,744,100	1,571,578	2,005,700	1,672,592	2,969,200	1,983,158	2,715,564	1,773,996
California	1.532,500	144,313	1,082,000	653,009	1,517,500	1,013,553	2,326,184	1,519,622
Colorado	14,331,600	8,312,328	12,942,800	7,811,239	12,447,400	8,313,743	12,059,202	7,877,915
Georgia	1,500	870	900	543	300	200	(*d*)	(*d*)
Idaho	7,810,200	4,529,916	8,125,600	4,903,962	8,836,200	5,901,786	8,491,356	5,547,148
Michigan	127,800	74,124	253,000	152,690	186,100	124,298	(*d*)	(*d*)
Montana	14,608,100	8,472,698	13,454,700	8,120,181	12,540,300	8,375,792	12,118,000	7,916,226
Nevada	2,695,100	1,563,158	5,863,500	3,538,740	5,207,600	3,478,208	7,767,510	5,074,281
N. Mexico	214,600	124,468	354,900	214,189	453,400	302,830	431,246	281,720
N. Carolina	14,800	8,584	13,200	7,966	24,700	16,497	(*d*)	(*d*)
Oregon	133,200	77,256	88,900	53,653	90,700	60,579	(*d*)	(*d*)
S. Carolina	500	290	200	121	100	67	(*d*)	(*d*)
S. Dakota	187,000	108,460	179,000	108,030	155,200	103,658	(*d*)	(*d*)
Tennessee	59,200	34,336	95,400	57,576	25,600	17,098	(*d*)	(*d*)
Texas	469,600	272,368	417,200	251,788	277,400	185,278	307,545	200,910
Utah	12,484,300	7,240,894	10,319,800	6,228,205	11,508,000	7,686,298	11,747,562	7,674,330
Virginia	6,700	3,886	200	121	100	67	(*d*)	(*d*)
Washington	149,900	86,942	119,400	72,060	42,100	28,119	(*d*)	(*d*)
Wyoming	4,400	2,552	2,700	1,630	1,100	735	(*d*)	(*d*)
Other States	....	....	(*c*)13,000	7,846	31,300	20,906	737,752	481,951
Total	57,786,100	$33,515,938	56,101,600	$33,858,438	56,517,900	$37,748,757	58,850,530	$38,445,181
Porto Rico	....	....	....	....	....	....	5	3
Philippine Islands	....	....	....	....	....	....	80	52
Total	57,786,100	$33,515,938	56,101,600	$33,858,438	56,517,900	$37,748,757	58,850,615	$38,445,236

(*a*) The statistics in this table are reported by the Director of the Mint, those for 1907 being the preliminary figures (subject to revision); except that later official figures for 1907 have been used, when available. (*c*) Includes Maryland, 100 oz.; Missouri, 2,900 oz. (*d*) Included in other States.

TOTAL PRODUCTION OF GOLD AND SILVER IN THE UNITED STATES.

Years.	Gold.	Silver.	Years.	Gold.	Silver.	Years.	Gold.	Silver.
	Dollars.	*Ounces.*		*Dollars.*	*Ounces.*		*Dollars.*	*Ounces.*
1792—1834	14,000,000	*Nil*	1883	35,000,000	35,730,000	1896	53,088,000	58,835,000
1835—1844	7,500,000	103,365	1884	30,800,000	37,800,000	1897	57,363,000	53,860,000
1845—1854	343,036.769	386,730	1885	31,800,000	39,910,000	1898	64.463,000	54,438,000
1855—1864	479,300,000	20,806,518	1886	35,000,000	39,685,513	1899	71,053,000	54,764,000
1865—1874	454,950,000	154,390,609	1887	33,000,000	41,721,592	1900	79,171,000	57,647,000
1875	33,400,000	24,533,993	1888	33,175,000	45,792,682	1901	78,666,700	55,214,000
1876	39,900,000	30,010,054	1889	32,800,000	50,000,773	1902	80,000,000	55,500,000
1877	46,900,000	30,783,509	1890	32,845,000	54,516,300	1903	73,591,700	54,300,000
1878	51,200,000	34,960,000	1891	33,175,000	58,330,000	1904	80,723,200	57,786,100
1879	38,900,000	31,550,000	1892	33,000,000	64,900,000	1905	88,180,700	56,101,600
1880	36,000,000	30,320,000	1893	35,955,000	60,000,000	1906	94,373,800	56,517,900
1881	34,700,000	33,260,000	1894	39,500,000	49,500,000	1907	89,198,711	58,850,615
1882	32,500,000	36,200,000	1895	46,610,000	55,727,000	Total	2,969,819,580	1,732,668,437

Note.—To the end of 1872, the statistics are those of R. W. Raymond, United States Mining Commissioner; subsequent statistics are those reported by the Director of the Mint.

caused a great curtailment in the yields of these mines in the last part of the year, by reason of the closing down of many of the big producing mines.

The production of silver was affected in the same way as gold, but nevertheless the total for the year was 58,850,615 oz., showing an increase of 2,332,715 oz. over 1906; this total being the largest reported since 1893, though it exceeded that of 1896 by only 15,615 oz. The production of silver continues to be made in connection with that of lead, copper and other metals, very few mines being worked in this country for silver alone.

GOLD AND SILVER PRODUCTION OF THE WORLD, 1493-1850.

According to Dr. Adolph Soetbeer.

Period.	Estimated Production In Kilograms.		Ratio of Silver to Gold. Weight.	Ratio of Gold to Silver. Value.	Period.	Estimated Production In Kilograms.		Ratio of Silver to Gold. Weight.	Ratio of Gold to Silver. Value.
	Gold.	Silver.				Gold.	Silver.		
1493–1520	162,400	1,316,000	8.1	10.75	1701–1720	256,400	7,112,000	27.7	15.21
1521–1544	171,840	2,164,800	12.6	11.25	1721–1740	381,600	8,624,000	22.6	15.08
1545–1560	136,160	4,985,600	36.6	11.30	1741–1760	492,200	10,662,900	21.7	14.75
1561–1580	136,800	5,990,000	43.8	11.50	1761–1780	414,100	13,054,800	31.5	14.73
1581–1600	147,600	8,378,000	56.8	11.80	1781–1800	355,800	17,581,200	49.4	15.09
1601–1620	170,400	8,458,000	49.6	12.25	1801–1810	177,780	8,941,500	50.3	15.61
1621–1640	166,000	7,872,000	47.4	14.00	1811–1820	114,450	5,407,700	47.2	15.51
1641–1660	175,400	7,326,000	41.8	14.50	1821–1830	142,160	4,605,600	32.4	15.80
1661–1680	185,200	6,740,000	36.4	15.00	1831–1840	202,890	5,964,500	29.4	15.75
1681–1700	215,300	6,838,000	31.8	14.97	1841–1850	547,590	7,804,150	14.3	15.83

GOLD PRODUCTION OF THE WORLD, 1851-1907

Year.	Value.	Year.	Value.	Year.	Value.	Year.	Value.
1851.......	$ 67,600,000	1866......	$121,000,000	1881......	$103,102,000	1896......	$211,242,081
1852.......	132,800,000	1867......	104,000,000	1882......	102,000,000	1897......	237,833,984
1853.......	155,500,000	1868......	109,700,000	1883......	95,400,000	1898......	287,327,833
1854.......	127,500,000	1869......	106,200,000	1884......	101,700,000	1899......	311,505,947
1855.......	135,100,000	1870......	106,900,000	1885......	108,400,000	1900......	258,829,703
1856.......	147,600,000	1871......	107,000,000	1886......	106,000,000	1901......	260,877,429
1857.......	133,300,000	1872......	99,600,000	1887......	105,775,000	1902......	298,812,493
1858.......	124,700,000	1873......	96,200,000	1888......	110,197,000	1903......	329,475,401
1859.......	124,900,000	1874......	90,800,000	1889......	123,489,000	1904......	349,088,293
1860.......	119,300,000	1875......	97,500,000	1890......	118,848,700	1905......	378,411,754
1861.......	113,800,000	1876......	103,700,000	1891......	130,650,000	1906......	405,060,969
1862.......	107,800,000	1877......	114,000,000	1894......	146,292,600	1907......	412,556,136
1863.......	107,000,000	1878......	119,000,000	1893......	158,437,551	....	....
1864.......	113,000,000	1879......	109,000,000	1894......	182,509,283	....	....
1865.......	120,200,000	1880......	106,600,000	1895......	198,995,741	....	....

SILVER PRODUCTION OF THE WORLD, 1851-1907.

Year.	Kilograms.	Years.	Kilograms.	Year.	Kilograms.	Years.	Kilograms.
1851-1855..	4,430,575	1881......	2,592,639	1891......	4,479,649	1901......	5,438,443
1856-1860..	4,534,950	1882......	2,769,065	1892......	4,985,855	1902......	5,121,469
1861-1865..	5,505,575	1883......	2,746,123	1893......	5,339,746	1903......	5,386,044
1866-1870..	6,695,425	1884......	2,788,727	1894......	5,205,065	1904......	5,669,124
1871-1875..	9,847,125	1885......	2,993,805	1895......	5,667,691	1905......	5,638,183
1876.......	2,323,729	1886......	2,902,471	1896......	5,496,178	1906........	5,683,947
1877.......	2,388,612	1887......	2,990,398	1897......	5,663,304	1907........	6,033,121
1878.......	2,551,364	1888......	3,385,606	1898......	5,575,336	....	....
1879.......	2,507,507	1889......	3,901,809	1899......	5,529,024	....	....
1880.......	2,499,998	1890......	4,180,532	1900......	5,599,216	....	....

Included in the mint returns for 1907 silver from our island possessions appeared for the first time. The quantity, however, was very small; only 5 oz. from Porto Rico and 52 oz. from the Philippines.

Alabama (By Eugene A. Smith).—In Cleburne county there are several

GOLD PRODUCTION OF THE WORLD.

Countries.	1905			1906			1907		
	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.
America, North:									
United States....	4,265,742	132,682.0	$88,180,700	4,565,333	142,001	$94,373,800	4,315,081	134,215	$89,198,711
Canada..........	700,863	21,800.0	14,486,833	581,709	18,093	12,023,932	399,844	12,437	8,264,765
Newfoundland....	4,550	141.5	94,049	4,475	139	92,500	4,315	134	89,191
Mexico (*a*).......	779,181	24,236.0	16,107,100	805,000	25,039	16,639,350	862,119	26,816	17,820,000
Central America..	73,212	2,277.0	1,513,400	(*e*) 58,321	1,814	1,205,500	63,328	1,970	1,310,000
South America:									
Argentina........	(*e*) 446	13 .9	9,200	(*e*) 731	23	15,100	731	23	15,100
Bolivia..........	(*e*) 147	4.5	3,000	(*e*) 1,064	33	22,000	1,209	38	25,000
Brazil...........	(*a*) 117,396	3,651.5	2,426,575	(*e*) 120,948	3,738	2,500,000	146,218	4,548	3,022,326
Chile............	(*e*) 30,812	958.4	636,900	(*e*) 30,963	966	640,000	33,624	1,046	695,000
Columbia........	(*e*) 95,513	2,970,8	1,974,400	(*e*) 95,791	2,980	1,980,000	96,517	3,002	1,995,000
Ecuador.........	(*e*) 6,430	200.0	132,900	(*e*) 6,430	200	132,900	6,193	193	128,000
Guiana (British).	82,300	2,559.9	1,701,141	79,682	2,478	1,647,031	66,802	2,078	1,381,797
Guiana (Dutch)...	34,442	1,071.3	711,916	38,162	1,187	788,820	34,883	1,085	721,031
Guiana (French)..	(*e*) 86,532	2,691.5	1,788,800	78,028	2,427	1,612,863	74,427	2,315	1,538,406
Peru............	(*e*) 17,406	541.4	359,782	(*e*) 17,900	557	370,000	18,500	575	382,395
Uruguay.........	(*e*) 1,227	40.0	25,368	(*e*) 1,210	38	25,000	1,210	38	25,000
Venezuela........	(*e*) 14,512	451.4	300,000	(*e*) 14,998	466	310,000	14,271	444	295,000
Europe:									
Austria..........	(*e*) 2,283	71.0	47,190	(*e*) 2,225	69	46,000	2,254	67	46,590
France..........	....	....	....	....	....	....	46,296	1,440	956,952
Hungary.........	(*e*) 117,949	3,668.7	2,438,006	(*e*) 117,078	3,828	2,420,000	119,598	3,720	2,472,090
Germany (*c*).....	126,446	3,933.0	2,613,639	135,094	4,202	2,792,439	150,526	4,682	3,111,372
Italy............	(*e*) 325	10.1	6,718	(*e*) 319	10	6,600	322	10	6,645
Norway.	(*e*) 350	10.9	7,234	387	12	8,000	387	12	8,000
Portugal.........	(*e*) 40	1.3	827	(*e*) 48	1	1,000	48	1	1,000
Russia...........	1,063,883	33,402.3	22,197,155	1,087,056	33,812	22,469,432	1,282,934	39,905	26,578,253
Spain............	(*e*) 257	8.0	5,316	(*e*) 266	8	5,500	322	10	6,645
Sweden..........	(*e*) 1,958	60.9	42,235	1,984	62	41,000	1,929	60	39,872
Turkey..........	(*e*) 1,400	43.5	29,000	(*e*) 1,403	44	29,000	1,447	45	29,909
United Kingdom.	(*e*) 13,584	422.5	280,781	12,849	400	266,500	1,891	59	39,087
Africa:									
Madagascar......	66,258	2,060.9	1,369,553	56,585	1,760	1,169,608	54,012	1,680	1,116,428
Rhodesia........	348,518	10,840.4	7,203,865	479,089	14,902	9,902,873	545,082	16,954	11,266,845
Transvaal........	4,897,221	152,324.1	101,225,558	5,786,617	179,988	119,609,373	451,494	200,669	133,352,381
West Coast.......	165,844	5,158.4	3,427,995	199,432	6,203	4,122,260	272,277	8,468	5,627,970
Asia:									
Borneo (British)..	(*e*) 42,745	1,329.5	883,539	(*e*) 42,332	1,317	895,000	42,332	1,317	875,000
China (*e*)........	85,918	2,673.0	1,776,100	(*e*) 217,688	6,771	4,500,000	217,688	6,771	4,500,000
East Indies (Dutch)...	68,426	2,128.0	1,414,500	55,636	1,419	1,150 000	60,234	1,874	1,245,000
India (British)....	576,889	17,943.7	11,924,308	533,658	16,599	11,030,711	524,995	16,330	10,857,648
Japan (*g*)........	161,105	5,011 0	3,330,300	169,747	5,280	3,508,670	164,753	5,090	3,403,378
Korea...........	(*e*) 58,055	1,805.7	1,200,000	(*e*) 120,948	3,771	2,500,000	120,948	3,771	2,500,000
Malay Peninsula......	(*e*) 18,990	590.7	392,522	16,933	527	350,000	15,627	485	325,000
Australasia (*d*)...	4,159,220	129,369.2	85,970,779	3,984,538	123,927	82,358,207	3,669,636	114,132	75,849,349
Other Countries...	72,570	2,257.2	72,570	(*e*) 72,570	2,257	1,500,000	72,570	2,257	1,500,000
Totals.....	18,360,945	571,422.5	$378,411,754	19,504,407	609,348	$405,060,969	19,958,764	620,766	$412,556,136

(*a*) Figures based on exports and coinage. (*c*) Includes output from domestic ores only. (*d*) Six States and New Zealand (*e*) Estimated. (*f*) Includes Servia, Persia, West Indies, Formosa and British New Guinea. (*g*) Exclusive of Formosa.
Note—The value of gold is $20.67 per troy ounce, which is equivalent to $664.55 per kilogram.

places where a large amount of gold has been obtained in the past. The best known are Arbacoochee and Chulafinnee. At the former the gravels on Dine creek have yielded the greater part of the gold, but some years ago a quartz vein was exposed which carried free gold. Recently dredging

operations have been carried on in this section. In the Turkey Heaven mountain and along its flanks, are many places where developments have been made for gold, but at none of these, at this time, is active work going on. Similarly in Clay county, much prospecting and some mining work have been carried on at a number of localities, as in the Goldberg district, near the eastern line of Clay bordering on Randolph, and in the Idaho district at the Franklin mine and the Ivey mine; but at present there is nothing doing at either of these places. In Randolph county is one of the oldest gold mines in the State, the Pinetucky, which, until

WORLD'S PRODUCTION OF SILVER.

Country.	1906			1907.		
	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.
America, North:						
United States	56,517,900	1,757,944	$37,748,757	58,850,615	1,830,501	$38,445,236
Canada	8,568,685	266,522	5,723,110	12,750,004	396,579	8,329,205
Mexico (*a*)	68,500,000	2,130,638	45,751,835	65,600,000	2,040,435	43,814,896
Central America	(*e*) 670,000	20,838	447,500	685,000	21,306	447,390
America, South:						
Argentina	(*e*) 70,000	2,177	46,754	68,000	2,115	44,422
Bolivia	(*e*) 6,650,000	206,843	4,441,602	6,750,000	209,984	4,409,573
Chile	(*e*) 850,000	26,439	567,724	875,000	27,216	571,211
Colombia	(*e*) 980,000	30,482	654,552	950,000	29,549	620,606
Ecuador	(*e*) 40,000	1,244	26,716	40,000	1,244	26,131
Peru	(*e*) 5,100,000	158,631	3,406,341	5,250,000	163,297	3,429,668
Uruguay	(*e*) 1,000	31	668	1,000	31	65
Europe:						
Austria	(*e*) 1,270,000	39,502	848,246	1,349,862	41,915	881,824
Hungary	515,000	16,019	343,974	424,489	13,204	277,306
France	19,500	607	13,024	225,918	7,027	147,585
Germany (*c*)	11,649,160	393,442	7,780,590	12,439,896	386,933	8,116,610
Greece	(*e*) 820,000	25,505	547,686	755,525	23,500	493,562
Italy	(*e*) 800,000	24,883	534,328	695,051	21,619	454,056
Norway	170,000	5,288	113,545	175,475	5,458	114,633
Russia	120,241	3,740	80,310	166,183	5,169	108,562
Spain	3,825,000	118,974	2,554,756	4,099,125	127,500	2,677,835
Sweden	(*e*) 21,500	669	14,360	32,375	1,007	21,150
Turkey	550,000	17,107	367,351	522,438	16,250	341,293
United Kingdom	162,500	5,054	108,535	138,245	4,300	90,311
Asia:						
Dutch East Indies	185,000	5,754	12,356	190,000	5,910	124,121
Japan	2,431,447	88,151	1,623,987	2,431,447	88,851	1,623,987
Australasia	(*e*)13,519,410	420,510	9,029,749	17,516,433	544,803	11,442,960
Africa	(*e*) 495,000	15,397	330,615	510,000	15,863	333,168
Other Countries (*d*)	(*e*) 50,000	1,555	33,396	50,000	1,555	32,663
Total	184,552,343	5,683,947	$123,152,367	193,542,381	6,033,121	$127,520,029

(*a*) Statistics compiled from export and coinage. (*c*) Silver produced from domestic ores only. (*d*) The output is mostly from China and Persia. (*e*) Estimated.

Note.—Unless specified to the contrary, the statistics have been taken from official sources. The average commercial value of silver for 1906 was 66.791c. per oz., equal to $21.47 per kg.; for 1907 it was 65.327c. per oz., or $21.00 per kilogram.

recently, was in continuous operation for many years. In Tallapoosa county there are two principal belts of gold bearing rocks, running northeast and southwest; the Goldville-Hog mountain belt, and the Silver Hill belt. Both of these held active mining camps before the discovery of California gold; and in both districts the mining for gold has been going on more or less continuously ever since, but on no considerable scale, except in the vicinity of Hog mountain, at the mountain itself, at the old

Ely pit and at the Ulrich mine on Hillabee creek. At Hog mountain Messrs. Aldrich have installed a cyanide plant and have been conducting mining operations on a rather large scale for a number of years, and these operations are still in progress. On the old Ely vein some recent developments have been made by the Hood Bros., who have sunk a shaft and a slope and erected a mill. A cyanide plant was also, for a time at least, in operation in the Dutch Bend on Hillabee creek at the old Ulrich mines. In the Silver Hill region, mining operations have been confined practically to Silver Hill, Blue Hill and Gregory Hill.

In Talladega county, along the eastern flank of the Talladega mountains at Riddle's Mill, on the Woodward tract, at the Story mine, and at a few other points, a thin but high-grade vein of gold-bearing quartz has been worked at intervals for a number of years. This work seems to have been confined to the free-milling surface ores, and no attempt has been made to utilize the more refractory ores below the water level. A company recently organized will reopen the Story mine.

West of the Coosa river, in Elmore and Chilton counties, are several placers well known to the gold miners in the early days and which have also been worked for the gold in comparatively recent times. Many of the failures attending the attempts at gold mining in this State, especially after the rich placers of Arbacoochee and Chulafinnee and the Goldville and Silver Hill regions had been well worked over, are to be attributed to bad management, and the use of methods which are ill adapted to the character of the ore.

Alaska.—The decrease of $3,113,490 in the gold production of Alaska in 1907 was much less in proportion than that of the adjoining region of the Canadian Yukon. It shows the drop which comes to all placer mining countries after the first rich ground is worked over, and mining comes down to a lower, but more substantial basis. Alaska is in the period of transition now, but the decreased production of many of the placers was offset by the discovery or opening of several new districts; as in the Fairbanks district, on the Shushitna river and in the Tanana country.

The larger quartz mines on Douglas island continue to work, paying dividends from their treatment of large deposits of very low-grade ore. There is but little quartz mining in other parts of the territory.

Seward Peninsula (By John Power Hutchins).— The gold production from this important section of Alaska showed an increase in 1907, differing somewhat from those of many placer regions, as such districts generally attain their maximum production within five years after discovery. The Seward peninsula is different, for many of its placer deposits are extremely inaccessible; consequently it has taken time to discover these deposits and to install the ditches and machinery needed to exploit them. Indeed it is probable that the annual gold production in Seward peninsula will

increase for some time to come as the more inaccessible regions are developed and exploited on their proper scale.

The year was favorable in some respects, but in others unfavorable for production. The construction of new ditches and the installation of machinery at the placers helped production. On the other hand labor troubles, culminating in strikes, were a considerable drawback during the winter of 1906-07 and curtailed production. The fact that in spite of this, there was an increase in the production during 1907 is a good index of prosperous conditions.

At Nome some very rich gravel was mined on the third beach line, notably on Cooper creek, where a mine, employing about 30 men, yielded approximately $10,000 per day during the second half of 1907; litigation, resulting from the unsatisfactory state of the mining laws, prevented a longer working season. Other mines on the same and adjacent creeks, where these cross the third beach line, were also very profitable.

On Seward peninsula the rainfall is greater than in the interior of Alaska so that the water supply was ample during 1907. The season was in general a normal one, although the freeze-up was a little early. Ordinarily open-cut mining by hand methods begins about June 15 and continues until about the first of October. The labor supply during 1907 was ample except during the time that the strike was on. Wages are high; 40c. per hour and board and 30c. per hour and board is the wage for unskilled labor during summer and winter, respectively.

Open-cut, drift and vein mining are all used in exploiting the gold deposits of Seward peninsula. In open-cut mining hand methods, dredges, and hydraulic monitors are used. Owing to the fact that it is more often possible in the districts of Seward peninsula than in regions farther in the interior, such as the Tanana and Klondike districts, to obtain a gravity head of water, hand methods, the rocker, etc., are less important than in the interior districts; in fact the amount of gold mined by hand methods is relatively small.

Open-cut mining by hydraulic methods is done by straight hydraulicking, by hydraulic elevating and by ground sluicing. Straight hydraulicking is not carried on extensively, as the topography is generally unfavorable to such mining; in most cases sufficient grade for sluices and dump for tailings cannot be obtained. Hydraulic elevating has been conducted successfully. One of the larger plants washed about 1200 cu.yd. per day. The yardage handled by ground sluicing was relatively small.

Open-cut mining by dredging was conducted on Solomon river and Ophir creek. The dredge on Solomon river is of the Oroville type; the buckets have a capacity of 5 cu.ft. This dredge is said to have excavated more than 3000 cu.yd. per day and to have worked very successfully in alluvium about 15 ft. in depth, but which is generally unfrozen. The

dredge on Ophir creek has 5-cu.ft. buckets and is of comparatively light construction. This dredge had several unprofitable seasons, while working in frozen material, but, it is said, was operated profitably during the past season, as the alluvium handled was less frozen than formerly. Steam thawing was used to prepare the material for dredging. A dredge having 9-cu.ft. buckets was installed on Bourbon creek. This dredge is electrically driven. The power plant uses California petroleum for fuel and the generators are driven by steam turbines.

Besides the difficulties due to a short working season, high cost of installation and operation greatly hampers dredging; moreover the placers on Seward peninsula are partly or wholly frozen and so are difficult to excavate. Frozen gravel, which greatly resembles concrete, cannot be excavated rapidly by any form of excavator, no matter how powerful. Blasting has little effect on frozen gravel, for, after shattering with any kind of powder, the material re-unites along the lines of fracture by regelation. Frozen ground must therefore be thawed before it can be dredged. The experience on Seward peninsula is similar to that in the Klondike.

Open-cut mining with a steam shovel was used on Shovel creek, a tributary of Solomon river. This plant has the difficulties principally due to immobility common to such installations. Rich ground permitted profitable operation, although the working cost was about 50c. per cubic yard.

More than one-third of the gold mined on Seward peninsula during 1907 was obtained by drift mining. This method is used especially in mining the old beach deposits back of Nome, as the pay gravel occurs on bedrock and is rarely covered with less than 50 ft. of poorer gravel. As this gravel is frozen, it must first be thawed with steam, then it is broken down and hoisted to a storage bin at surface; the hight of this bin is sufficient to allow the sluices to be placed at the proper grade and still give plenty of dump room for the tailings.

During 1907 one gold vein was worked; this mine is on a tributary of Solomon river. The results of this work are said to be disappointing.

Mining costs did not vary materially during 1907. The cost of drift mining was about \$3.50 per cu.yd.; hydraulic mining costs were about 30c. per cu.yd. for straight hydraulicking and about \$1 per cu.yd. where hydraulic elevating is necessary. The cost of dredging was about 20c. per cu.yd. in ground, most of which was unfrozen; when steam thawing was necessary the cost was about 60c. per cu.yd.

During 1907 several long ditches, whose total length approximates 100 miles, were built. The ditch, 40 miles long, which will furnish the water for working the placer deposits at Candle creek by hydraulic methods was finished. Ditch construction is very difficult on Seward peninsula for the ground is generally frozen and in many areas what is called "glaciers"

or "ground-ice" lies immediately below the surface of the soil. These circumstances make the cost of construction and maintenance high. Besides in the past there has been a tendency to construct long ditches without sufficient preliminary investigation not only of the deposits to be mined but also of the route to be followed. This is true of some of the ditches dug in 1907.

Near the coast coal is generally used for fuel as the Seward peninsula is, in general, treeless. The coal comes from the Puget Sound region. At Nome coal costs about $20 per ton. Its cost farther inland varies with the distance from the coast. In some cases California petroleum is used as fuel at mines inland. Two railroads lead from the coast to the interior. These have assisted in developing the country tributary to them, especially the Kougarok district. Wagon roads are few and poor. The excellent roads built and maintained in Klondike have no counterpart on Seward peninsula, or, for that matter, in any other part of Alaska. The high cost of mining in Alaska is largely due to poor facilities for transportation. There are large areas that cannot be exploited until good wagon roads are built to them.

California.—In California the production of gold keeps on at about the same rate with very little variation. The average annual yield is between $18,000,000 and $19,000,000 but 1907 fell considerably below the lower figure. There are more than 1000 productive gold mines in the State, large and small, and over 3000 more which are in the development stage or held by assessment work. There are more placer mines (hydraulic, dredge, drift, surface) than quartz or deep mines, but the latter still yield the larger proportion of the gold. The yield from the quartz mines is about the same from year to year, though for 1907 it was evidently somewhat less than usual, owing to labor strikes which affected several large properties. The placers, however, are showing a material annual increase, which is entirely due to gold-dredging operations. In fact, while most of the counties in which placer mining predominates show an increase of output, those where quartz mining is the prevailing interest show a falling off. The deep mines yield about 64 per cent. of the annual output, the rest coming from the various forms of placers. In placer mining the work carried on by the dredge is by far the most important. They now produce about 70 per cent. of the total placer yield, or nearly $3,000,000 more than the hydraulic, drift and surface mines combined. The dredges now dig out considerably more than one-quarter of all the gold yield of the State, and this proportion is gradually increasing from year to year as new and larger dredges are installed.

One effect of the increased importance of dredging is to place Butte county in the lead of gold production, a position held for many years by Nevada county, through the operation of its quartz properties. There

are more dredges in Butte county than elsewhere in the State, though the individual machines near Marysville, on the Yuba, are of large capacity. The largest one in California is that now operating in the Folsom field in Sacramento county. The dredges of the State obtained a little over $5,000,000 in 1906, but their output for 1907 was $1,300,000 greater. Several very large machines were installed during 1907, as well as a number of ordinary capacity. Some new dredging fields are being exploited in Shasta, Siskiyou and Trinity counties, and others in Fresno, San Bernardino and Mariposa counties are expected to show some production in 1908.

There has been some disposition shown to increase the number of operating hydraulic mines in California, though few of these seem to be on a very large scale. Numerous permits to mine by this process have lately been asked for from the California Debris Commission in the older mining counties where this particular form of placer mining has languished of late years. However, the recent decision of the State Supreme Court to the effect that the permit or license to mine by hydraulic process, issued by the California Debris Commission, is not final and does not bar suit being brought to close such owner down when damage can be shown, is a severe blow to the hydraulic mining industry, in the drainage basins of the Sacramento and San Joaquin rivers where the Camenetti law applies. Hydraulic mining is at its best in the northwestern counties. Siskiyou county has the largest number of these mines, but the yield from Trinity from this source is the largest in the State.

There are only about 100 drift mines now at work in the State; few have been opened of late. The yield from this source is not as great as formerly. It is a form of gravel mining which takes considerable capital, and usually a long time must pass after beginning work before profits can be realized, owing to the necessity of running long adits to reach the buried auriferous gravel channels. For this reason it does not seem a favorite form of gravel mining, although the results are generally rather profitable. A number of new projects of this character in the older fields of the higher Sierra are being undertaken, but it will be some time before results are known.

Surface placer mining continues to be carried on upon the rivers and water courses of the foothill and mountain region, but the gold output from this source is small.

At the ratio of increase shown in the past few years, the various forms of placer mining combined will be yielding more than the quartz mines of the State. The older quartz mines in the principal producing sections, such as Nevada county and the Mother Lode sections, seem to be keeping up their yield about the same as usual. Every year, however, there are changes, some mines ceasing production and closing down while others begin to yield profitably.

The mines about Alleghany, in Sierra county, are being exploited pretty thoroughly, both gravel and quartz; some of the latter are now making excellent yields. At Angels, Calaveras county, where there are several large producers, labor strikes materially reduced the output in 1907.

Generally speaking, the gold mines in the southern part of the State are not making as good a showing as formerly, especially in Kern, Riverside, San Bernardino and San Diego counties. In San Bernardino county considerable prospecting is being done and a number of new camps were established in 1907. These are as yet mainly in course of development with few producers of note. In Inyo county a great deal of prospecting was carried on during 1907, especially in the Greenwater copper district, but there was no production to note. In the Mother Lode counties in the upper central portion of the State, a few of the old producers shut down, but several other properties are being brought to the productive stage. In Sierra county there was more active interest in 1907 than in most of the counties. In Plumas many new enterprises are being established. Sacramento county is making a large yield from its few dredges, and Yuba county, from an abandoned hydraulic region, is yielding now over the million mark. All this is from dredges. Amador county is now not far behind Nevada county in gold yield. Six counties of the State, Amador, Butte, Calaveras, Nevada, Tuolumne, and Yuba, now show a yield of over $1,000,000 a year in gold; one of these yields $3,000,000, while two yield over $3,000,000. El Dorado, Fresno, Madera, Mariposa, Mono and a few others show a falling off in yield.

As to silver output, something over $800,000 per year, more than half comes from Shasta county, where the principal copper smelters are situated. No other county produces $100,000 per year except Kern and Mono. In the other counties, most of the silver is obtained from gold ores, and the proportion is small comparatively. The closing down of some of the Shasta county copper smelters during the latter part of 1907 caused a material reduction in the yield of silver in California for the year.

Colorado (By George E. Collins).—During 1907 Colorado had a somewhat uneventful history. The long expected new bonanza camp remains undiscovered, and in the older districts we are still without any new mine of the first magnitude. On the other hand, the labor troubles, which for so many years alarmed the mining industry of the State and disturbed its development, were absent, thanks largely to a watchful attitude on the part of the operators in the principal districts. The main difficulty encountered was a shortage of labor, and above all of efficient skilled labor; but in this respect Colorado was no worse off than the other Western States. At the close of 1907 the fall in the prices of metals,

and to a less extent the depression in the Eastern money markets, led to some curtailment of output.

The production of the Cripple Creek district was in the neighborhood of $11,000,000, of which the Portland contributed nearly one-fifth. The other chief producers were the Golden Cycle, Elkton, Vindicator and (during the first half of the year) Stratton's Independence. The destruction by fire of the Golden Cycle mill on Aug. 7 caused a great reduction of output. Many of the producing mines had contracted to ship their ore to this mill, and as these contracts still remain in force they were unable to make advantageous terms with the United States Reduction Company for the few months that would elapse while the mill was being rebuilt. The reconstruction of the mill was practically completed by the end of the year. Cripple Creek seems more and more to be developing into a leasing camp. On this basis the Little Clara, was a large producer, though hardly so prominent as it was in 1906. The lessees in the Granite produced a considerable amount, especially during the first half of 1907. At the close of the year the drainage tunnel was in about 1200 ft., having been driven for several months by the El Paso company on contract, with only one shift. A new contract has since been concluded, under which more rapid progress is being made. In the meantime the lower workings on the Portland and Strong were unwatered under a joint pumping agreement.

The Leadville output in 1907 was as good as in 1906, although owing to the fall in the values of lead and zinc and the depressed condition of the Eastern markets, which compelled the smelters to shut off many of their customers who were not subject to long-time contracts, the output for the last two months of the year was reduced. The properties of the Western Mining Company, especially the Coronado and Wolftone, continued by far the greatest shippers in the district. The Moyer mine of the Iron Silver company and the Yak Tunnel were the largest independent producers, and a considerable tonnage, aggregating perhaps 200 tons per day, was made by leasers on the Ibex properties. Among the smaller operations the Crescentia on Rock Hill, opened in 1906, was a considerable shipper. The Mammoth mine, reopened by the Wellington Association and very widely advertised in certain journals, was shut down and the pumps were drawn. At the close of the year the properties of the Western Mining Company were shut down, and the pumps were removed. Work was being continued by lessees in the stopes above the water-level, but as these low-grade orebodies were never very profitable even with metals at high prices, it is doubtful whether the mines will ever be reopened.

The San Juan region was free from serious misfortunes during 1907, but unfortunately the list of profitable mines was not extended. The output, however, was greatly in excess of that for 1906. In the vicinity

of Telluride the Tomboy continued to be a large and profitable producer, with the Smuggler-Union and Liberty Bell following it in the second rank. At Silverton the Silver Lake, Gold King and Sunnyside are still the leading mines. The new mill of the Gold Prince at Animas Forks was in operation most of the year, but never to full capacity. The old Aspen mine was reopened and a section of the rebuilt Silver Lake mill was set aside for the concentration of its ore. At Ouray the Camp Bird produced heavily throughout the year. Should the depression in the metal markets continue, the output of San Juan county will be considerably reduced in 1908. That of Ouray and San Miguel counties, on the other hand, will not be very seriously affected.

In Gilpin county the condition of general depression was unchanged; on Quartz Hill in particular there was not a single mine working, and the water now stands at a uniform depth of about 500 ft. from the surface. The Newhouse tunnel, driving of which was suspended in February, 1907, is within 1000 ft. of some of the principal veins, but it is generally understood that it will not be completed until satisfactory arrangements have been made with all of the principal mines ahead. The one bright feature in Gilpin county was the Russell Gulch district, where the Old Town and Pewabic were large producers. During the year the Old Town and Saratoga workings were connected with laterals from the Newhouse tunnel. All the properties on the Calhoun vein were consolidated under one control, and an important program of development was begun, a feature of which will be the sinking of two shafts to the level of the Newhouse tunnel, and the driving of a lateral from the tunnel level to connect with them.

In the lower Clear Creek district the chief producers were the Gem, Little Mattie, Shafter and Specie Payment. The great feature of mining in this district continued to be in connection with the deep crosscut tunnels; of these the Newhouse has already been referred to in connection with Gilpin county; the Central and Lucania made some progress during the year, and also the McClelland tunnel, which is being driven to unwater the Freeland district, and the Rockford, which intersected the Donaldson vein at great depth and will be used as the avenue through which it will be exploited.

At Georgetown there was some revival of activity, but also a notable absence of really important discoveries.

In Boulder county there was but little new development. The Wano mill was successfully run, and a new mill was erected at the Inter-Ocean mine at Sunshine. As a whole, however, I may safely say that the needs of the Boulder county district are rather in the direction of underground exploration than surface improvements.

In the vicinity of Breckenridge a good deal of zinc and lead was shipped,

but this was discouraged considerably by the fall in prices. The Revett dredge on French gulch was successfully operated, and a great deal of drilling was done farther down on the Swan, which is likely to lead to a considerable increase of dredge mining. There was some excitement in the Montezuma district of Summit county, and it is reported that the railroad will be extended from Keystone up the Blue river. This incipient boom mainly manifested itself in the consolidation of properties, and it may be questioned whether sufficient development has been done as yet to establish the future of the district. A large output of silver-bearing lead and zinc concentrates was made during the first half of the year from the Creede United mill at Creede; but this was, at the end of the year, greatly lessened, owing partly to unremunerative prices, and partly perhaps to exhaustion of the deposits. Comparatively little interest was taken in other Creede properties. At Aspen no new discoveries were reported. The Smuggler and Percy La Salle were the chief producing mines; the latter, however, was shut down for the winter pending the erection of a mill.

There were incipient differences between operators and their employees in more than one district, but in each case the difficulty was adjusted or overcome without a strike. At Cripple Creek an attempt was made by a small coterie of local business men and politicians to overthrow the card system, but it was soon apparent that this movement met with no sympathy among either miners or mine operators. The system is now more firmly established and more thoroughly carried out than ever. An uncalled for attempt on the part of the local union in Gilpin county to disturb the harmonious condition which for so many years has distinguished that district, was entirely abortive. At Silverton the contract between the local Mine Operators' Association and Miners' Union, under which wages and working hours had been fixed since 1902, expired. A new schedule was announced by the mine owners, but although they were unwilling in this case to enter into any further agreement with the union leaders, the latter tacitly accepted the new schedule, as it was obviously as favorable as conditions in the district would warrant. At present the only dubious condition of affairs with reference to labor is at Telluride, where the card system was given up and where an attempt was made to revive the sinister influence of the union.

Georgia.—There was no revival of mining in 1907, and work was confined to a few small operations in the Dahlonega district. The attempt to consolidate a number of mines and to work on a large scale has been a failure.

Idaho (By Robert N. Bell).—Owing to the abundance of water enjoyed by the placer miners of Idaho during 1907, and other causes, the gold output of the State showed an increase over that of 1906, and was $1,087,655. An important part of the increase is credited to the old

DeLamar mine, where the new 100-ton mill was in steady operation throughout the year. The ore of this mine averages about $12 per ton, of which about 70 per cent. is gold and 30 per cent. silver. It is treated by cyaniding at a cost of something like $2.50 per ton. A good force of men is kept on development work, with the result that this old mine is in better shape today in the matter of ore reserves than it ever has been, although the ore now being treated is much lower in grade than formerly.

The Golden Sunbeam mine, near Custer City, in Custer county, developed much free-milling gold ore during 1907. It is equipped with a small Elspass mill. The ore at this mine is a soft andesite, containing free gold. The deposit is several hundred feet in width and is exposed for fully a half-mile. Crosscuts indicate that the ore averages $2@4 per ton across a width of 500 ft.; possibly this whole mass can be mined by the "glory hole," or steam-shovel method. Increased milling capacity is now being installed.

In the Atlanta district in Elmore county, the Monarch mine and the Pettit mine have recently been equipped with 200-ton and 100-ton mills, respectively. The Buster mine at Elk City has recently been equipped with a 10-stamp mill by F. W. Bradley and associates, and is now being supplied with $20 ore from a 10-ft. vein.

The silver output of Idaho for 1907 was 8,491,356 oz. of which 7,317,962 oz., including 450,000 oz. from the treatment of Snow Storm copper ores is credited to the Cœur d'Alene lead mines. The Trade Dollar and DeLamar mines in Owyhee county produced 799,873 oz. in 1907, and the remainder was derived from the lead and copper ores mined in Blaine, Custer and Lemhi counties.

Missouri (By E. R. Buckley).—Some silver is obtained in refining the lead concentrates of the disseminated lead district. The St. Louis Smelting and Refining Company has been recovering the silver associated with the galena for several years and the St. Joseph Lead Company contemplates making a similar separation in 1908. The concentrates carry from 1 to 2 oz. of silver to the ton.

Montana.—The gold and silver output of this State is so largely won in connection with copper, that the principal comment upon its mining activities will be found in the article on that metal. Some notes on the precious metals as separate products, however, may be given.

(By Edwin Higgins.) The production of gold in Montana has been on the increase for the past 10 years, but 1907 showed a small decrease from the previous year. The heavy snows of the winter provided an abundance of water for milling purposes and this, coupled with the improved machinery and better methods employed, was a factor attending the success of gold mining in the State for the year. During the last four months of 1907, when the demand for copper properties had practi-

cally ceased, increased activity was noted in gold. The end of the year found an increased number of gold prospects and producers in operation. Placer mining was most extensively carried on in the vicinity of Virginia City, in Madison county, where five electric dredges were in almost continuous operation during the entire year. Sluicing methods were employed in several other districts.

Nevada (By W. H. Shockley).—At the end of 1906 the mining boom was still on in Nevada, but by October, 1907, the wave had subsided and the enthusiasm of the speculators had been thoroughly quenched. The mines, however, did well and 1908 should be a banner year. The business of mine promoting and wildcat stock selling seems to be nearly a lost art, hardly any new companies being advertised and those in a timid way contrasting vividly with the flamboyant advertisements of the past. The chief event for 1907 was the closing of the State Bank and Trust Company, and the Nye & Ormsby County Bank at the end of October. In their various branches at Reno, Carson, Tonopah, Goldfield, Blair, Wonder, and Manhattan, these banks had total deposits of more than $3,000,000. These bank failures were caused by the great shrinkage in values of local mining shares. An added cause in the failure of the State bank was its losses from the Sullivan Trust Company, which went up in 1906 with alleged assets of $3,090,000, mostly of only nominal value. It seems correct to state that the whole population of Nevada was mildly insane at the end of 1906 as regards mining stocks. The Nye & Ormsby County Bank reopened on Jan. 2, 1908.

None of the newer camps developed any notable mines, and no camp of importance was started during 1907 in spite of large sums spent in prospecting. There was, however, a great deal of work done which will add largely to Nevada's future production; indeed, 1907 might well be called the "mill-builders" year, for more mills were completed and planned than at any time during the history of the State. The Las Vegas & Tonopah Railroad was completed to Goldfield, and the Tonopah & Tidewater to Rhyolite. The Western Pacific is pushing ahead rapidly and is now operating 175 miles in the eastern part of Nevada. The Nevada Northern is completed to Ely. Branch roads have been finished to Pioche, Searchlight and Fallon. Among the new railroads for which surveys have been made is the Fallon-Fairview, which is greatly needed.

At Rhyolite the Montgomery-Shoshone mill began work in September, but for a time was bothered by the refusal of the men to use "unfair" electric current generated at the power plant in Bishop Creek, where a strike was in progress. This strike lasted but a short time and the mill ran through October, crushing 3000 tons, worth $25 per ton. Besides this, 1000 tons were sent to the smelters with an estimated value of $100 per ton, the total product for the month being reported as $175,000.

The mill is of 300-ton capacity, crushing with rolls; the pulp passes over amalgamating plates, and is then concentrated and cyanided; no tube mills are used. Outside of the Montgomery-Shoshone property none of the mines of Rhyolite did much. The Tramps Consolidated, owning a large number of claims which were separately supposed to be very valuable, proved a frost; the large quartz ledges are very low-grade and the high-grade ore very small in quantity. Most of the other Rhyolite properties merely present possibilities. The Gold Bar and Homestake King mines, about five miles west of Rhyolite, show wide ledges of low-grade gold ore. Each of these mines began to build a 20-stamp mill to be finished early in 1908. Contradictory reports came from the Tecopa mine near the line of the Tonopah & Tidewater railroad south of Rhyolite, which was a producer of silver-lead ore years ago, when it was in an inaccessible country now opened up by the railway. In January, 1908, the mine was shipping 60 tons of lead ore daily. The Gold Bullfrog Mining Company, nine miles northeast of Rhyolite, completed its 35-ton mill, but milling was delayed owing to lack of water; this was remedied by sinking an 800-ft. well. Judging from results, the Mayflower and other mines near Beatty did not open up very satisfactorily. A 10-stamp mill was running at Bonnie Clare, halfway between Rhyolite and Goldfield, but as the property was closed in December, 1907, the ore was probably poor.

Goldfield is still the banner camp of Nevada, making a large production during 1907 and paying substantial dividends. So far as ascertained, the total was about $1,800,000. Besides this a number of leasing companies paid considerable sums. At the close of the year more people were leaving Goldfield than were coming in; the local trains to Tonopah were taken off and there was but one daily train to San Francisco, instead of two. There was no trouble with the transportation of sufficient supplies.

In November, because of the scarcity of coin, the mine-owners were obliged to pay off in checks; this was not satisfactory to the miners and a strike and shutdown resulted. The mine-owners, fearing a repetition of the troubles of Cripple Creek and the Coeur d'Alene, asked for protection from Governor Sparks who, there being no Nevada militia, requested aid from President Roosevelt, and Federal troops were sent to Goldfield on Dec. 4, 1907. These troops camped in Goldfield but took no part in the government of the town. There was much ordering and countermanding of the troops but they were still in Goldfield at the close of the year. Their presence undoubtedly kept the town quiet, there having been no disturbance during their stay. On Dec. 9, the mine-owners started work under a new wage-scale, miners' wages being reduced from $5 to $4 per day and other wages in proportion; a card system was adopted

by which no member of the Western Federation of Miners was to be employed: this card system was abandoned later, but the reduced wages were enforced and the mines ran nearly full-handed though the strike declared by the Western Federation still remained in force. On Jan. 14, 1908, a special session of the Nevada legislature was called to take measures for preserving peace in the State, President Roosevelt having announced that this was the duty of the State of Nevada and not of the United States, and that he would not allow United States troops to remain in Goldfield indefinitely. On Jan. 24, 1908, a bill was passed creating a constabulary force with a maximum of 250 men. The miners were dissatisfied with this bill and proposed to have it submitted to a *referendum* as provided for by Nevada laws, but the general opinion was that the bill would be sustained by the *referendum*. Outside of Goldfield, Nevada was free from serious labor troubles. One of the Tonopah newspapers made a special fight against foreign miners, and especially against those from southern and eastern Europe; though not meeting with much support in Tonopah this found sympathizers in Round Mountain where strong resolutions were passed about Jan. 18, 1908, by the Round Mountain Miners' Union. The largest producer in Goldfield was the Goldfield Consolidated, which was a shipper through the year, besides working ore in the Combination and other mills and making plans for a large mill to be built in 1908. The property is said to show large deposits of medium-grade ore with some of the very rich ore still remaining. Ore has been found in large amounts on the 450-ft. level and has been cut on the 650-ft. level. One of the main reasons for the closing of some of the Goldfield mines in December, 1907, was that the smelters were unable to make advances against ore shipments, but withheld payment until they received mint returns. This required from 60 to 90 days from the time the ore was shipped. The smelters also penalized Goldfield ores by putting on an extra 10 per cent. charge so as to check shipments. In February, 1908, the smelters were again paying cash for ores as soon as possible after sampling. Though Goldfield is largely overbuilt there is every reason to think it will be a prosperous camp for a long time to come. The Florence mine is thought to be second only to the Goldfield Consolidated and is reported to have wonderful showings of rich ore.

The Tonopah mines were run steadily in 1907, but in spite of the large production were rather disappointing. The cutting of the dividend of the Tonopah Mining Company from 35c. to 25c. per quarter was depressing and led to a stockholders' row which caused a change in the management. The Belmont Mining Company stopped paying dividends, though its 60-stamp mill at Millers was completed, and was paid for by the sale of treasury stock at $3 per share. The ore of the Tonopah Mining Company is treated in the company's 100-stamp mill at Millers, which has a capacity

of 400 tons daily. The ore is concentrated on Wilfley tables and the concentrates shipped. The coarser pulp is ground in Huntington mills and the tailings from the Wilfleys are cyanided; Blaisdell excavators are used. During 1907 the Montana-Tonopah Mining Company finished its 40-stamp mill, designed by F. L. Bosqui. This is well arranged and is doing excellent work. The ore, crushed by stamps, is classified into sands and slimes, and the former concentrated on Wilfley tables. The tailings are ground in two large tube mills and this pulp concentrated by Frue vanners; the tailings from the vanners are cyanided, Hendryx agitators and Butters filters being used. The gold is precipitated by zinc dust. The concentrates and precipitates are shipped to smelters. The operating expense is reported to be not far from $3 per ton with a saving of 90 per cent. This mill will be of great utility in working the low-grade (under $30 per ton) ores of Tonopah of which large quantities exist in a number of the mines. The Butters mill in Virginia City worked several hundred tons of Tonopah ores monthly, charging $6 per ton and paying 85 per cent. At this rate there was a slight advantage over the smelters on ore running from $30 to $40. The smelters pay 95 per cent., but are handicapped by higher freight.

Tonopah stood the panic better than most of the camps. There is, however, an uneasy feeling as to the permanence of the Tonopah mines owing to failure to find good ore on the lower levels, as pointed out by Spurr in his report on the district. In the lower levels, say below 700 ft., are found strong quartz ledges 30 ft., to 60 ft. wide very low in value. One of these ledges is being prospected on the 800-ft. level of the Midway mine and another in the 1000-ft. level of the Tonopah Extension. It is a great pity that prospecting in depth was not started by the Tonopah Mining Company some years ago and pushed to the 1500-ft. level or deeper, for it will have to be done some day.

Manhattan was rather quiet during 1907 but joined the mill-building procession. This camp was especially hard hit by the San Francisco earthquake-fire, for the mines are largely owned in that city. In the last few months of the year there was renewed interest in the camp, some especially good developments being reported from the Thanksgiving mine and by some of the leasers on other properties. Some of the litigation, which retarded the growth of the camp so much, was settled. Both here and at Round Mountain placers formed by the erosion of the quartz veins are worked by dry washers run by gasolene engines treating about 50 tons of gravel daily. The operations are said to be successful. The Manhattan Ore Reduction and Refining Company started its 10-stamp mill which is provided with a tube mill and a cyanide plant and is expected to crush 50 tons daily. The first run on Manhattan Consolidated ore was said to average $20 per ton. The Lennon mill is also a 10-stamp

custom mill. The Drake & Chapman has Huntington mills with 30-tons capacity. Both these mills are nearly completed.

At Round Mountain much work was done; a hydraulic plant costing $100,000 has been put in to work the placers. The Round Mountain Mining Company has a Nissen stamp and Huntington mill working its ores. The Round Mountain Fairview has a Huntington mill, and there is a stamp mill running on custom ore. The Sphinx Mining Company put up a 10-stamp mill for its ores.

At the new town of Blair near Silver Peak, the Pittsburg Silver Peak Mining Company has finished its 100-stamp mill and is now running. This mill will be provided with a cyanide plant and will be one of the best mills in the State. The orebodies in this property are extensive, and there is enough ore running something under $10 to keep the mill going for a long time. This mill is run by power provided by the Nevada & California Power Company which is now enlarging its plant near Bishop creek. The 10-stamp mill of the Silver Peak Valcalda Mining Company is running and working well

A new camp, Stimler, has been started during the last few months and has a number of leasers working; this is in Cottonwood cañon, a few miles from Silver Peak. The Walker Lake rush following the opening of the Walker Lake Reservation in October, 1906, resulted in nothing of moment. The bulk of the claims located during this stampede of 4000 men will probably be abandoned. There was much talk in the summer of the rich silver-gold mines at Regent, 50 miles east of Wabuska, but the prospects did not develop well. Rawhide, near Regent, had a boom, the main street being over a mile long and the population 2000 or more. A number of leasers are working here on rich stringers in porphyry, but expert opinion is that the population of the camp is about three years ahead of developments. Goldyke and Atwood did not make good; the Goldyke Reef Mining Company started its two Huntington mills but was short of water. It is said that the ore is probably too low-grade to pay.

The Comstock mines at Virginia City have levied their usual assessments; prospecting from the Sutro tunnel has given favorable indications. It is proposed to build a large mill at the mouth of the tunnel to be run by the water power from the tunnel.

Rosebud, from whose rich surface showings much was expected, is dull. The ore has given out in depth in most of the mines. Prospecting is still going on with some little encouragement. Seven Troughs is said to have at least one good property in the Fairview claim of the Seven Troughs Mining Company. Fairview made a good production during 1907, the Nevada Hills having paid $225,000 in dividends or $300,000 since its discovery in 1906. None of the other mines produced anything of note though some of them are well thought of. The Ramsey Comstock mine

at Ramsey shows wide ledges of $15 to $20 ore, and is to have a mill in 1908. A great deal of work was done in Searchlight, the best mine being the Quartette, which paid dividends in 1907. Wonder proved disappointing. Some of the chief mines closed down owing to the fact that their money was locked up in the banks, and a number of leasers quit because the ore proved too low-grade to ship. Wonder is heavily handicapped by being 70 miles from Fallon, its nearest railroad station, whence most of the supplies, including lumber and mining timber, come. Wood is scarce and water costs $1.25 per barrel. A large amount of low-grade ore is, however, already developed, and with the bringing in of water, the construction of railroads and the building of mills, the camp should do well. The principal mines are the Nevada Wonder, the Jackpot, Vulture, and the Spider-Wasp, the latter being several miles from the main camp. Aurora supports a 20-stamp mill, the ore being furnished by leasers. There is a large amount of quartz showing in this camp, the croppings being wide; but it is very low in value, perhaps too low even for the largest mills, and certainly too poor for anything short of them. Gold Circle, north of Golconda, had a small boom in September but the rich gold quartz found on the surface did not go down; prospecting is still going on. The railway has been completed to the old camp of Pioche and there is in consequence hope that the old mines on reopening will prove profitable.

New Mexico (By Charles R. Keyes).—The gold production in 1907 was due mainly to the copper, zinc and lead ores, the mining of which showed gains. The production is almost entirely from the southwestern parts of the territory. Although there are about 100 deep gold mines in New Mexico they do not produce what would ordinarily be expected of them. This condition is due partly to the fact that some of the more important and normally the more productive properties are far removed from adequate transportation facilities. Several of the large mines were not in operation in 1907. Placer mining should be much more extensively developed than it is, considering the advantages and richness of the gravels. Gold-bearing gravels of New Mexico are the richest in the United States. They are extensive and are widely distributed, but heretofore it has been difficult to obtain sufficient water to work them properly. Only in a few places, in the northern part of the region, could hydraulic methods be used. The water problem has become less serious of late and it is not unlikely that good supplies of water can be secured in most of the placer districts. Western Socorro county and Sierra county continue to be the principal gold-producers. New and extensive developments in the vicinity of Hillsboro are of a promising character.

The chief silver production is still credited to the Mogollon district in the western part of Socorro county. The silver ores here are associated

with those of gold and copper. Some silver is also obtained from the gold ores of the southwestern districts. Little of the silver in the complex sulphide ores is saved as yet, although some of them contain considerable. None of the older silver mines is operated at the present time, notwithstanding the fact that there have been many improvements in methods of treating this class of ores over the manner of a dozen years ago when these mines were closed down. The ores of this class are mainly horn-silver, and are usually rich, at least down to permanent water-level, or depths of from 300 to 500 ft. The lead ores of New Mexico contain as a rule but little silver. Exceptions are noted in the extreme southwest corner of the Territory and in the Hachita range.

North Carolina.—This State was the only producer in the South of importance. Work continued steadily at the Iola mine in Montgomery county, and at several others. There was a revival of interest, which was shown in the opening of several old mines and the beginning of work at some new ones.

Oregon.—In the Baker city district, in the eastern part of the State, attention was so largely turned to developments on the so-called copper belt that gold mining was rather neglected. In Lane, Douglas and Josephine counties in the western section, operations were carried on, but not on a large scale.

South Dakota.—With a strike of the mine and millmen that lasted four months, and a fire in the Homestake mine that caused an entire cessation of operations in the mills for over a month, and a partial suspension for six weeks longer, the Black Hills showed a decrease of $2,519,000 in the output in 1907. The Homestake produced about four-fifths of the total. Several of the companies, affected by the demand for eight hours, which were compelled to shut down on Jan. 1, 1907, remained closed for seven and eight months, and two of them were still closed at the end of the year, viz., the Gilt-Edge Maid and the Golden Crest. The total amount of bullion produced by the mines other than the Homestake was approximately $1,250,000. The largest producers outside of the Homestake were the Imperial, Mogul and Golden Reward. The monthly shipment of the Homestake at the close of the year was about $600,000 worth of bullion, while that of the other companies was about $130,000. The most important event of the year was, of course, the stubborn and persistent fire in the Homestake mine. After other methods of extinguishing had been tried without success, it was decided to flood the entire mine. This was done, and it was subsequently unwatered by special pumping apparatus, which was described in the *Engineering and Mining Journal*, March 28, 1908.

Utah (By L. H. Beason).—At Mercur, the most interesting gold camp, conditions have improved. The Consolidated Mercur Gold Mines Company,

the principal operator in the camp, installed a new slimes plant and made other changes in its mill equipment resulting in saving a much higher percentage of gold. A feature of the work at the Consolidated Mercur in 1907 was the reopening of the Brickyard mine. This property had been idle for six years, having been shut down, not because it was worked out, but because the ore was of low grade and could not be profitably treated at that time. Now, with reduced expenses and improved extraction, this ore pays well, and a large tonnage of it has been made available. The Brickyard mine supplies the mill with 150 out of the 600 to 800 tons treated daily.

The Sacramento closed down in July, after nearly 15 years' steady operation; a few men were kept on for prospecting purposes. In late years the Sacramento has produced considerable quicksilver, and it bears the distinction of being the only quicksilver-producing mine in the State. The Sacramento produced about $30,000 worth of gold in 1907. The Sunshine mine, near Mercur, was purchased by a syndicate headed by George H. Dern, of the Consolidated Mercur, and plans were made to remodel the mill and begin production.

The newest gold camp in Utah is Gold Springs, situated in the western part of Iron county. The Jennie Gold Mining Company was fairly successful, and added a cyanide department to the mill. In the Gold Mountain district the Annie Laurie and the Sevier Consolidated carried on the principal operations. The former produced close to $200,000 worth of gold in 1907. The mill at the Sevier Consolidated proved to be a failure, and the company was placed in the hands of a receiver. The property is in debt to the extent of about $230,000.

The Ophir district is in a prosperous condition, with two mines producing, the Ophir Hill and Cliff. At Stockton the only active producing property now is the New Stockton. There are indications of a revival in the Stateline mining district in Iron county; but work was done there only in a small way in 1907. The mining developments of Utah during the year were largely in connection with copper and other metals. Further particulars will be found elsewhere, especially in the article on copper.

Washington.—The gold production continues small, and was about the same as in 1906. There were no noteworthy developments during 1907.

Wyoming (By H. C. Beeler).—The year's production of gold in this State is estimated at $30,000, coming principally from the Penn-Wyoming company's works at Encampment, where a small gold content is noted in the copper ores, and from the various small placers throughout the State. The silver is given at 3500 oz., coming also from Encampment, and this will evidently be increased during 1908, as there are a number of properties working on ores showing a higher silver content than before

but whose shipments were made too late to be included in the 1907 report. The principal event in the gold mining in this State during 1907 was the reopening of the Miner's Delight mine, 28 miles southwest of Lander, Fremont county, in the South Pass district, by Senator C. D. Clark and associates. This property has a wonderful record for its size, but has been in litigation for many years, and only recently became available for reopening and development. This property shows a number of veins centering in the vicinity of the present works, and ore conditions indicated, as far as the present works have been reopened, fully warrant the prediction that it will again become a gold producer. The Carissa property at South Pass has been operated only a short time, and there was no return of its production. In southern Albany county there are a number of placers, which are being systematically exploited, before production machinery is installed, and production is anticipated from this district in 1908.

Philippine Islands.—The mint returns probably do not include all the gold produced, although they give all that reached the mints or assay offices. While no large production was made, several mines and at least one dredge were in operation during the year, principally in the Baguio district. Official returns of their production, however, are lacking.

Commercial Movement of Gold and Silver.

The general course of the movement of gold and silver is shown in the accompanying tables. The price of silver fell off sharply in the closing months of the year; chiefly owing to a decreased demand from India.

GOLD IMPORTS AND EXPORTS, UNITED STATES.

	1903	1904	1905	1906	1907
Imports	$65,267,296	$84,803,234	$50,293,405	$155,579,380	$143,398,066
Exports	44,346,834	121,138,415	46,794,467	46,709,158	55,215,681
Balance	I $20,920,462	E $36,335,181	I $3,498,938	I $108,870,222	I $88,182,385

GOLD IMPORTS AND EXPORTS, GREAT BRITAIN.

	1903	1904	1905	1906	1907
Imports	£28,657,393	£33,876,588	£38,567,895	£46,042,590	£57,088,547
Exports	27,766,512	33,039,138	30,829,842	42,617,267	50,866,009
Excess Imports	£890,881	£837,450	£7,738,053	£3,425,323	£6,222,538

GOLD IMPORTS AND EXPORTS, FRANCE.

	1903	1904	1905	1906	1907
	Francs	Francs	Francs	Francs	Francs
Imports	314,259,000	656,063,000	779,648,000	430,473,000	492,336,000
Exports	129,890,000	123,976,000	131,494,000	165,087,000	154,572,000
Excess Imports	184,369,000	532,087,000	648,154,000	265,386,000	337,764,000

GOLD HOLDINGS OF THE LEADING EUROPEAN BANKS.

	1903	1904	1905	1906	1907
Bank of England...........	$153,536,320	$149,980,465	$142,651,255	$145,322,390	$165,383,645
Bank of France............	502,500,480	469,307,905	575,671,135	541,150,235	538,787,750
Bank of Germany..........	145,434,000	154,370,000	166,300,000	137,940,000	137,140,000
Austro-Hungarian Bank.....	230,700,000	231,165,000	224,325,000	233,045,000	228,795,000
Bank of Russia............	382,865,000	426,375,000	576,215,000	589,520 000	607,125,000
Bank of the Netherlands....	23,493,000	21,036,000	33,019,500	27.680,000	38,239,100
Belgian National Bank......	14,983,335	15,613,335	16,233,335	17,076,665	17,610,000
Bank of Italy..............	84,345,000	108,520,000	134,345,000	159,440,000	193,320,000
Bank of Spain.............	71,925,000	72,780,000	75,117,000	76,840,000	78,210,000
Bank of Sweden...........	....	....	18,900,000	19,780,000	20.325,000
Total.................	$1,609,778,135	$1,649,147,705	$1,962,775,225	$1,947,794,290	$2,024,935,295

AVERAGE PRICE OF BAR SILVER IN LONDON, 1833–1902.

In pence per standard ounce, 0.925 fine.

Year.	Pence.	Year.	Pence.	Year	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.
1833	59.1875	1843	59.1875	1853	61.5000	1863	61.3750	1873	59.2500	1883	50.5625	1893	35.6250
1834	59.9375	1844	59.5000	1854	61.5000	1864	61.3750	1874	58.3125	1884	50.6250	1894	28.9375
1835	59.6875	1845	59.2500	1855	61.3125	1865	61.0625	1875	56.8750	1885	48.6250	1895	29.8750
1836	60.0000	1846	59.3125	1856	61.3125	1866	61.1250	1876	52.7500	1886	45.3750	1896	30.7500
1837	59.5625	1847	59.6875	1857	61.7500	1867	60.5625	1877	54.8125	1887	44.6250	1897	27.5625
1838	59.5000	1848	59.5000	1858	61.3125	1868	60.5000	1878	52.5625	1888	42.8750	1898	26.4375
1839	60.3750	1849	59.7500	1859	62.0625	1869	60.4375	1879	51.2500	1889	42.6875	1899	27.4375
1840	60.3750	1850	60.0625	1860	61.6875	1870	60.5625	1880	52.2500	1890	47.6875	1900	28.2500
1841	60.0625	1851	61.0000	1861	60.8125	1871	60.5000	1881	51.6875	1891	45.0625	1901	27.1875
1842	59.4375	1852	60.5000	1862	61.4375	1872	60.3125	1882	51.6250	1892	39.8125	1902	24.0900

AVERAGE PRICE OF SILVER.

New York, in cents per fine ounce.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct	Nov.	Dec.	Year.
1902....	55.56	55.09	54.23	52.72	51.31	52.36	52.88	52.52	51.52	50.57	49.07	48.03	52.16
1903....	47.57	47.89	48.72	50.56	54.11	52.86	53.92	55.36	58.00	60.36	58.11	55.375	53.57
1904....	57.055	57.592	56.741	54.202	55.430	55.673	58.095	57.806	57.120	57.923	58.453	60.563	57.221
1905....	60.690	61.023	58.046	56.600	57.832	58.428	58.915	60.259	61.695	62.034	63.849	64.850	60.352
1906....	65.288	66.108	64.597	64.765	66.976	65.394	65.105	65.949	67.927	69.523	70.813	69.050	66.791
1907....	68.673	68.835	67.579	65.462	65.981	67.090	68.144	68.745	67.792	62.435	58.677	54.565	65.327

London, in pence per standard ounce, 0.925 fine.

1902....	25.62	25.41	25.00	24.34	23.71	24.17	24.38	24.23	23.88	23.40	22.70	22.21	24.09
1903....	21.98	22.11	22.49	23.38	24.89	24.29	24.86	25.63	26.75	27.89	27.01	25.73	24.75
1904....	26.423	26.665	26.164	24.974	25.578	25.644	26.760	26.591	26.349	26.760	26.952	27.930	26.399
1905....	27.930	28.047	26.794	26.108	26.664	26.910	27.163	27.822	28.528	28.637	29.493	29.977	27.839
1906....	30.113	30.464	29.854	29.984	30.968	30.185	30.113	30.529	31.483	32.148	32.671	32.003	30.868
1907...	31.769	31.852	31.325	30.253	30.471	30.893	31.366	31.637	31.313	28.863	27.154	25.362	30.188

UNITED STATES: EXPORTS AND IMPORTS OF SILVER.

	1903	1904	1905	1906	1907
Exports...................	$40,610,342	$50,312,745	$57,513,102	$60,957,091	$61,619,653
Imports...................	23,974,508	26,087,042	35,939,135	44,227,841	46,005,776
Excess, exports	$16,635,834	$24,225,703	$21,573,967	$16,729,250	$15,613,877

GREAT BRITAIN: EXPORTS AND IMPORTS OF SILVER.

	1903	1904	1905	1906	1907
Exports	£11,466,726	£13,263,694	£14,561,677	£18,865,285	£16.920,349
Imports	10,310,330	11,687,339	12,992,014	17,288,063	15,983,892
Excess, exports	£1,156,396	£1,576,355	£1,569,663	£1,577,222	£936,457

EXPORTS OF SILVER FROM LONDON TO THE EAST (a)

	1903	1904	1905	1906	1907
India	£7,423,330	£9,527,618	£7,230,421	£15,129,627	£11,163 424
China	310,060	512,792	886,847	433,957	559,450
The Straits	821,879	79,268	38,299	1,750	1,029,356
Total	£8,555,269	£10,119,678	£8,155,567	£15,565,334	£12,752,230

(a) As reported by Pixley & Abell.

UNITED STATES MINT PURCHASES OF SILVER IN 1907.

	January.	February.	March.	April.	May.	June.	July.
Ounces	800,000	800,000	950,000	1,517,000	900,000	1,005,000	1,187,500
Av. price cents per oz.	69.2675	69.1776	67.9978	65.8510	66.5316	67.4328	69.4277

	August.	September.	October.	November.	December.	Year.
Ounces	351,000	100,000	1,070,000	2,540,000	3,556,000	14,776,500
Av. price cents per oz.	69.0650	68.6010	60.8767	58.9813	55.3353	62.7335

Production of Gold and Silver in Foreign Countries.

Africa.

The Transvaal, Rhodesia and West Africa are the three chief gold producing regions of the African Continent. The attempted revival of the old mines of Egypt has not succeeded, while the possibilities of the Sudan and the Congo Free State are still almost unknown.

The Transvaal (By W. Fischer Wilkinson).—The value of the gold won from the Transvaal mines during 1907 amounted to £27,403,738, which is the largest output yet recorded. The statistics of this and previous years are shown in the accompanying table.

The number of companies making returns in 1907 was much the same as in 1906, and the increase in the gold production was due to the larger tonnage handled. The grade of the ore treated again showed a reduction as compared with that of previous years, due to the increased scale of operations and the progress made in the methods of working.

There is still a considerable margin between the average working costs

and the average recovery, and a still further lowering of the grade may be expected, carrying with it a corresponding increase in the tonnage available for profitable mining. Low-grade ores of a content of 5 cwt. per ton or so, which were formerly considered worthless or of little value, are now becoming a valuable asset, and the prospective lives of the working mines are constantly having to be lengthened to meet the altered conditions. Undeveloped properties or those partially developed, whose prospects

TRANSVAAL GOLD PRODUCTION BY YEARS.
(Chamber of Mines Returns.)

Year.	Witwatersrand District.			Outside Mines Value.	Transvaal Total.
	Tons. Milled.	Value.	Value per Ton Milled.		
		£	s.	£	£
1884–9	1,000,000	2,440,000	48.8	238,231	2,678,231
1890	730,350	1,735,491	47.4	134,154	1,869,645
1891	1,154,144	2,556,328	44.2	367,977	2,924,305
1892	1,979,354	4,297,610	43.4	243,461	4,541,071
1893	2,203.704	5,187,206	47.0	293,292	5,480,498
1894	2,830,885	6,963,100	49.2	704,052	7,667,152
1895	3,456,575	7,840,770	45.2	728,776	8,569,555
1896	4,011,697	7,864,341	39.2	739,480	8,603,821
1897	5,325,355	10,583,616	39.74	1,070,109	11,653,725
1898	7,331,446	15,141,376	41.3	1,099,254	16,240,630
1899	6,872,750	15,067,473	43.84	661,220	15,728,693
1900	459,018	1,510,131	65.82	....	1,510,131
1901	412,006	1,014,687	49.25	81,364	1,096,051
1902	3,416,813	7,179,074	42.00	74,591	7,253,665
1903	6,105,016	12,146,307	39.79	442,941	12,589,248
1904	8,058,295	15,539,219	38.46	515,590	16,054,809
1905	11,160,422	19,991,658	35.82	810,416	20,802,074
1906	13,571,554	23,615,400	34.80	964,587	24,579,987
1907	15,523,229	26,421,837	34.04	981,901	27,403,738

with costs at 30s. per ton or thereabouts were not bright, can look forward to a profitable existence now that costs have been in many cases reduced to 20s. or less. The possibility of working low-grade ores profitably has now been amply demonstrated, and there is no danger of the Transvaal failing to contribute heavily to the world's supply of gold for many years after the exhaustion of the richer mines.

The chief producers are in the Witwatersrand district, the "outside" mines making comparatively unimportant returns. Of these, the Nigel mine in the Heidelberg district, the Sheba in the Barberton district, and Glynn's Lydenburg and the Transvaal Gold Mining Estates of the Lydenburg district were the large producers.

In the Witwatersrand district there were, according to the December returns of the Chamber of Mines, 66 producing mines. The total number of stamps running was 8383 or an average of 127 stamps per company. The Simmer & Jack and the Robinson are the largest producing mines. The Simmer & Jack, for the year ending August 31, 1907, milled 726,654 tons of ore at the rate of 6.448 tons per stamp per day, The company has a

battery of 320 stamps and four tube mills. The gold production per ton milled was as follows: Amalgamation, 19s. 8.032d.; cyaniding sands, 10s. 9.473d; cyaniding slimes, 1s. 7.706d; by-products, 0.300d.; total, 32s. 1.511d. The working costs were 18s. 4.745d. per ton milled, and the total value of gold won during the 12 months was £1,167,222. The Robinson mine has 210 stamps and milled 410,927 tons in 1907, the gold recovery being valued at £1,118,478, equal to 54s. 5d. per ton, or, adding the gold declared from reserve, an average return of close to 62s. per ton. The working costs during this period were 18s. 2d. per ton milled. The costs for the month of November were as low as 14s. 7d. per ton milled.

The mining companies publish as a rule elaborate statistics of their working results. There is, however, in the majority of cases, one important omission, and that is any information as to the probable life of the mine. The mines, having limited ore resources, become every year depreciated in value, and as it is not customary to provide a reserve fund for the redemption of capital it is of importance to the shareholder to have some guide as to how long he may expect dividends. It is only when the mine is getting near to the end of its life that full information on this important point is usually given. There are, however, companies that try to inform their shareholders what the life of their investment is likely to be, and a few examples taken from the past year's reports may be recorded. For instance, the Ferreira expects a life of six or seven years; New Kleinfontein 27 to 30 years; Crown Reef, 2½ years, exclusive, however, of 50 deep-level detached claims; Robinson Deep, 21 years and New Heriot, 17 years. The Geldenhuis Estate gives information as to the life by estimating the total tonnage available, which is perhaps as good a way as any, as it avoids the factor of stamps.

It is, of course, only possible to give an approximate estimate of what the life will be, as the tonnage to be treated depends on the cost of working and any reduction in that cost at once increases the ore tonnage. Some information, however, on this subject is wanted, so that shareholders may make provision for redemption of capital. Knowing the life, the value of the share based on the rate of interest required can be easily calculated with the help of actuarial tables.

Costs and recoveries vary considerably at the different mines according to the special conditions. On an average, the mines showed improved results over 1906, increased profits having been earned by treating larger tonnages. Reduced costs are, however, frequently accompanied by a reduction in the grade and bring no increase to the profits, which after all is the main consideration. But, as a rule, the increased output resulted in increased profits. The reduction that has taken place in working costs in recent years was given by a committee of engineers during the year to the Mining Commission appointed by the government to inquire into the

working of the mines. The figures for cost per ton milled are as follows: 1902, 25s. 4.69d.; 1903, 12s. 6.74d.; 1904, 24s. 3.93d.; 1905, 23s. 2.11d.; 1906, 22s. 1.69d. The grade of ore has, as shown in the table of annual gold production, been steadily falling and, of course, the reduction in costs is partly due to the increased tonnage handled. The figure for development redemption, which is a book entry calculated from the average cost of the total tonnage developed and charged to working costs, is specially affected by the estimated increase in the tonnage developed due to the lowering of the pay grade.

DIVIDENDS OF GOLD MINES OF THE TRANSVAAL.

Year.	£	Year.	£	Year.	£	Year.	£
1887......	12,976	1892......	879,320	1897......	2,707,181	1902(a)...	2,121,126
1888......	112,802	1893......	955,358	1898......	4,848,238	1903......	3,345,502
1889......	432,541	1894......	1,527,284	1899(a)...	2,946,358	1904......	3,927,830
1890......	254,551	1895......	2,046,852	1900(a)...	....	1905......	4,857,539
1891......	334,698	1896......	1,513,682	1901(a)...	415,813	1906......	5,735,161
						1907......	7,131,612

(a) War period, Oct. 11, 1899, to May, 31, 1902.

The figures given in the table showing total dividends indicate that the mining business is being conducted on a large and expanding scale with every outward appearance of prosperity, and many people must find it hard to understand the complaint of business depression in Johannesburg and South Africa and why shares are standing at prices considerably lower than those reached in former years. In order to explain this remarkable phenomenon, it is necessary to study the past history of the Rand.

Before the war and especially in the boom year of 1895 exaggerated estimates were made as to the value of the gold field, and companies were formed without sufficient thought being given as to whether the labor supply was sufficient to work them. Deep-level ground was sold at high figures, and in many cases working capital was raised at high premiums. Nor was speculation confined to mining ground, but real estate and buildings were bought and sold at inflated values, and generally prices were put up to figures above what the mines could pay interest on. Then came the war, which loaded the mines with further capital, additional funds having to be provided to meet war losses, expenses of maintenance, debenture interest, etc.

After the war in 1902 a second boom occurred founded on too optimistic notions of what the mines could perform under the new conditions, and again the limitation of the labor supply was not sufficiently reckoned with. Further the time required to restore the labor force that had been employed before the war was miscalculated. The recovery of this labor force was made especially difficult owing to the competition of the military

authorities and the public works department, who started work on roads and railways on an ambitious scale. The mining companies had, therefore, mines which could not be worked for want of labor, and the capital invested in the country received no return.

Thereupon came the cry for foreign labor, and the introduction of the Chinese. That experiment, which is now to be terminated, assisted the mines in getting to work more quickly than if they had to wait for the normal native supply. At the present time the number of natives employed exceeds what it was before the war, and, though perhaps not sufficient without the Chinese to meet all the requirements of the mines, is, at all events, large enough to support the mining industry on a considerable scale.

This historical review will help to explain why Johannesburg has been depressed. The depression is the natural consequence of the wild speculation of 1895 and 1902 and is traceable to the disregard of the quantity of unskilled labor that the country could supply. No doubt with imported Asiatic or other labor the mines could be worked on a larger scale, but the people of the colony have decided against this policy and the mining industry must accept the conditions that exist, and that existed when many of the present companies were formed, and by introducing every possible economy must try to make the best return possible on the huge sums that have been invested in the country.

During 1907 these facts were realized, and attention is now being devoted to seeing how expenses can be reduced and profits increased. To restore confidence in the South African mines profits and dividends adequate for this class of investment must be shown. Unfortunately the return in the past on the capital invested, taking an average of all the dividend-paying companies, is lamentable. It was stated before the Mining Commission, which sat during 1907 in Johannesburg, that 39 companies in 1906 paid £5,565,972 in dividends on a market value of £56,305,227 (April, 1907). The estimated average life of these companies was 12.66 years, and to redeem the capital at 3 per cent. compound interest would require £3,728,363. The balance of the dividends was only sufficient to pay 3 per cent. on the capital. Evidence was also given to show at the end of 1904 there were 163 mining and finance companies with a market valuation of £276,000,000. The dividends were £3,821,846 equal to 1.3 per cent. without any amortization. In 1905, these 163 companies had a market valuation of £170,000,000 and the interest earned was 3.8 per cent. In 1906, the market valuation was £135,000,000 and the interest 4.1 per cent. No allowance is made in each case for redemption of capital.

The labor requirements of the mines both as regards white and colored labor was the subject of constant discussion and debate. The decision

of the government to stop the importation of Chinese labor and to repatriate those in the country on the termination of their engagements made it necessary for the mine owners to look to the Kafir supply for their future unskilled labor force. The recruiting of Kafirs was fortunately attended with success, and the numbers employed at the close of the year were in excess of any previous period. The increase in the numbers is partly, however, explained in that there was very little fresh development going on in the country, and the fear was expressed that there would again be a shrinkage in the labor supply whenever public confidence was restored and fresh capital came into the country. Anyhow the demand for unskilled labor was satisfied toward the end of the year. It is as regards the future that there is anxiety.

PERSONS EMPLOYED IN RAND GOLD MINES AT END OF MONTH.

	White.	Colored.	Chinese.		White.	Colored.	Chinese.
1902—July	8,162	32,616	....	1905—June	16,939	104,902	41,340
December	10,292	45,698	....	December	18,159	93,831	47,267
1903—June	11,825	66,221	....	1906—June	17,959	90,882	52,352
December	12,695	73,558	....	December	17,495	98,156	52,917
1904—June	13,413	74,632	1,004	1907—June	17,166	111,862	57,517
December	15,023	83,639	20,885	December	17,697	129,618	37,118

It is estimated that in June, 1908, there will be only 21,000 Chinese left, in December, 1908, only 12,250, and that the whole lot will have gone early in 1910.

The withdrawal of this labor force can hardly take place without disturbing the gold production of the Rand. The non-producing or the poorer mines are, however, those which are most likely to suffer, and increased efficiency both of white and Kafir labor, together with the substitution of machine for hand drilling, may help to modify the severity of the decline in the output which the situation appears to demand. Those who have had experience in recruiting Kafir labor for the mines say that the supply is exhausted. Possibly that is so in the districts wherein recruiting has been carried on, but Africa is a big place, and with the opening of new country which is constantly going on, especially that north of Zambesi, it does not seem unreasonable to hope that fresh recruiting districts may be found.

Special light was thrown on the condition of white labor owing to a strike which took place in May. The strike originated at the Knights Deep mine over the question whether a white miner should supervise more than two machine drills. The management of that mine decided to reduce the price of contract stoping and proposed that the miners should supervise three drills instead of two, an arrangement which would

have resulted in an increase in their individual earnings. The strike extended to other mines, but it never seriously crippled the companies and ended in a victory for the employers. The strike led to the publication of figures showing that many of the miners were receiving wages amounting to £600 and to £800 per annum. At the Knights Deep for the month of April the average amount paid to machine stopers was £2 4s. per shift of eight hours, the maximum wage being £3 19s. 7d. and the minimum 14s. 11d. per shift. In hand stoping, the average wage was £1 19s. 6d. per shift of eight hours, the maximum wage being £3 3s. 3d. and the minimum 19s. per shift.

The conditions under which the machine men work make them deserving of good pay. But the investigation caused by the strike showed that white miners were receiving pay out of all proportion to the cost of living and far in excess of the carpenters, fitters, engine drivers, or shift bosses, and further, that much of the hard work of running the drill was done by natives or Chinamen. Consequently they received but little sympathy from the public. Their places were filled without difficulty. Many of those who took work on the mines at this time were the Afrikanders or Colonial-born who, accustomed to handle natives, were especially useful recruits. In order to encourage young men to study mining and fit themselves for work underground, mining classes were arranged for, in the hope that in this way a supply of efficient miners ready to work at reasonable pay would be obtained.

The average rates of pay on the Rand for the period July to December, 1906, are given in an accompanying table compiled from the statistics of the government mining engineer.

NUMBER AND WAGES OF MINERS ON THE RAND PER SHIFT.

Number employed.	Class.	Average Pay	
1392	Machine men, stoping	32s	9d
938	Machine men, developing	35	5
1524	Hand men, stoping	21	3
599	Engine drivers (hauling, surface)	20	2
177	Engine drivers (mill engines)	19	3
972	Fitters	20	2
610	Carpenters	20	2
294	Amalgamators	18	3
43	Cyanide Foremen	22	11
301	Cyaniders	15	9
273	Shift bosses	24	6

Much attention was given in 1907 to the study of working costs, and numerous suggestions were made for securing greater economy. There are some who maintain that stopes are carried too wide and that an unnecessarily high amount of waste rock is put through the mill. Closer

sorting is advocated by others, so as to raise the grade and increase the gold production. The more extended use of machine drills has its advocates and especially of the small stoping drills. Much time and money was spent on experiments with these small drills. Then the question of air losses came in for criticism, and it was pointed out that the air supplied to the mines is very often below 60 lb. The poor work done by the Kafir, namely, a 36-in. hole per shift, also caused remark. This is the task established by custom and it is one which many of the natives can perform long before the shift is up.

On the milling and metallurgical side several new departures from the ordinary practice were proposed. In the new battery being erected for the Simmer Deep, the bin framing and roof trusses will be of steel. The cast-iron anvil blocks are to be done away with and the mortar boxes placed directly on a concrete foundation. The mill is to be electrically driven, each ten 1670-lb. stamps having a separate motor. The gold thrown away in the residue did not escape the attention of critics, and finer grinding is advocated to recover something of the £2,500,000 gold that now goes over the dump. The costly and clumsy tailing wheels are condemned by some who advocate the use of some form of scraper to separate sands from slimes, as is the practice at some mines in the United States. A sand pump working on the injector principle invented by Mr. Robeson, the mechanical engineer of the Rand Mines, has also been suggested as an improvement on the tailings wheel. A new method of treating ores by cyanide was introduced called after its patentees, the Adair-Usher process. The process was introduced on several mines and has been well spoken of.

While there are many directions in which costs can be brought down, it is generally admitted that the most important economy to be effected is in the expenditure on white labor. Perhaps the most important evidence as to the value of the white miner is that given by the well-known American engineer, Ross E. Browne, who published during the year, an important report on the subject of working costs on the Rand, which he had made a special study of during a residence of 20 months. He compared the work done on the Rand with that done at a Californian mine, where the conditions were more or less similar as regards depth of mining and width of stoping. The results he obtained went to show that labor on the Rand was far less efficient than in California, and that, provided that both the Kafir unskilled labor and the white skilled labor were made more efficient—which he considered could be done—the cost of working could be brought down from a present cost of 22s. per ton milled to 15s. per ton. The results of his calculations are as follows: Present working cost per ton milled, 22s.; substituting Californian labor, wages, cost of supplies, economy of management, etc., 17s.; under the most favorable conditions,

with efficient skilled white labor and white direction of colored labor, reduced cost of supplies, etc., 15s.; with efficient labor, exclusively white, present white wages, present cost of supplies, etc., 26s.; with present white labor and substitution of available white for the present colored labor—maintaining the present average white wage, cost of supplies, etc.—a reliable estimate cannot be made, but the figure would probably exceed 35s. This report is very encouraging, and while the reduction from 22s. to 15s. may be too optimistic, there is such a wide margin that considerable improvement may be confidently expected.

Rhodesia (By W. Fischer Wilkinson).—There was much activity in mining during 1907 and the gold production exceeded all previous records. The total output was 612,052 oz. bullion, having a value of about £2,246,000. A table compiled by the Chamber of Mines shows the annual production since 1898. The figures differ somewhat from the table presented last year, the statistics having been adjusted to embrace the mines of Southern Rhodesia only.

GOLD PRODUCTION OF RHODESIA.

Year.	Oz. crude.	Year.	Oz. crude.	Year.	Oz. crude.
To Sept. 1, 1898	6,471	1901	172,062	1905	407,096
Sept. 1–Dec. 31, 1898	18,085	1902	194,169	1906	551,894
1899	65,304	1903	231,872	1907	612,652
1900	91,940	1904	267,737	Total to end of 1907	2,618,082

An important alteration in the mining law to take effect on Jan. 1, 1908, has been arranged. Hitherto, except in the case of small properties, mining companies had on flotation to give to the Chartered Company a certain percentage, originally 50 per cent., which was reduced later to 30 per cent., of the vendor capital. This system was objectionable because it tended to make the capitalization inflated and further because it put the Chartered Company in a position to make money out of the mines before an ounce of gold was produced. Under the new arrangement the share interest is to be abolished and a royalty on the gold produced substituted. The royalty is fixed on a sliding scale, but will average 5 per cent. of the gross output. The richest mines, exceeding 1 oz. per ton, will pay 7½ per cent., and the low-grade mines 2½ per cent.

West Africa (By W. Fischer Wilkinson).—The gold production of Ashanti and Gold Coast Colony in 1907 shows a considerable increase over that of 1906, due principally to the returns of the Prestea Block A mine, which commenced crushing in November, 1906, and is now the leading producer. The working mines and their production are shown in the accompanying table.

GOLD PRODUCTION OF THE GOLD COAST COLONY AND ASHANTI, 1907.
(West African Chamber of Mines.)

Mines.	Tons milled (2000 lb.)	No. of Stamps or equivalent.	Fine Gold recovered, ounces	Value, £.	Value per Ton, shillings.
Prestea Block A	78,164	50	45,623	194,047	49.6
Ashanti Gold fields	48,645	40	45,560	194,007	79.8
Abbontiakoon Block 1	71,776	50	28,791	122,304	34.1
Abosso	39,435	30	(a)27,541	117,006	56.3
Wassau	50,210	30	25,332	107,641	42.9
Bibiani	45,580	45	21,354	90,588	39.7
Broomassie	15,638	20	17,352	76,077	97.3
Taquah	22,168	50	17,254	73,259	66.1
Akrokerri	25,970	40	14,341	61,373	47.3
Sansu	31,291	20	7,702	32,619	20.8
Attasi	36,982	50	6,842	28,930	15.6
			257,692	1,097,851	....
Dredging companies	....	....	13,734	57,034	....
Total	465,859	425	271,426	1,154,885	49.6

(a) Includes 1394 oz., value £5926, from old tailings.

GOLD PRODUCTION OF WEST AFRICA.

Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.
1896	23,940	1898	17,733	1900	10,557	1902	29,880	1904	94,815	1906	199,432
1897	23,555	1899	14,250	1901	6,088	1903	70,763	1905	165,844	1907	271,426

Some improvements were made by various companies in milling plants. There is not much new development going on. Costs continue high, reports of different companies, growing totals varying from 42s. to 50s. per ton. The bad climate and high wages for skilled labor are important factors.

Asia.

Outside of Siberia—which is included with Russia under the head of Europe—the chief gold-producing countries in Asia are British India, China and Korea.

British India.—As in previous years the main production of gold came from the Kolar goldfield, in Mysore. The production in 1906 was somewhat reduced by the fact that one of the chief mines had reached a lean zone, where the ore proved much poorer than in the upper levels. Development work was continued, however, and in 1907 better ore was reached, and the output gradually increased. The Kolar district has been a producer for many years, and has been notable for its steady yield. The greater part of the output is from the four large mines—Champion Reef, Mysore, Nundydroog and Ooregum—which are owned in London and

are all under the management of John Taylor & Sons, of that city. The production of the other mines has not been large.

Outside of Mysore, operations are being carried on in the Deccan, where one mine has reached the producing stage and two or three others are being developed. In the southern part of India, in the Bombay and the Madras presidencies, there are several places where gold is washed out by the natives from river-beds; but the amount of this production is small, and is not exactly ascertained.

China.—The actual gold production of China is still an unsolved problem. It is variously stated by different authorities, whose estimates differ widely. There are no authoritative statistics collected. The existing guesses at both gold and silver production are based largely upon the figures of exports and imports. In 1907 the exploitation of gold mines in Manchuria was carried on extensively; to some extent this was done by Russian companies, but about the middle of the year an arrangement was made by which several large placers, which had been worked under Russian direction, were turned over to the Chinese authorities. Some Japanese companies are also exploiting mines in Manchuria.

With the object of stimulating gold production and coinage in China, the board of finance and the director of the Chinese Imperial Mint passed the following order in April, 1907. 1. That more mines be prospected for, so as to insure a sufficient supply of gold. 2. That the provinces be ordered to make purchases of the metal and quickly transport same to Tien-Tsin for minting. Rewards will be granted to the provinces which make the largest purchases. 3. That experimental coinage be made first at Tien-Tsin and extended to other provinces, should it prove to be satisfactory. 4. That a uniform rate of exchange be fixed. 5. That one-tenth of the pay of officials of all grades above 100 taels be in gold. 6. That gold coins be accepted for payment of customs and *likin.* 7. That gold, either in bullion or coin, is prohibited from being exported. 8. That the metal is prohibited from being used for the coating of idols. 9. That a law be passed to prevent the destruction of gold coins for any purpose.

(By Thomas T. Read.) The exact relative importance of the gold mining industry cannot be stated, for it is impossible to obtain reliable data. In The Mineral Industry for 1906 the production of gold for the whole of China was estimated at $4,500,000; while the same volume also quoted a Japanese estimate that the annual production of Manchuria alone was $10,000,000. This illustrates the degree of accuracy at present possible. As previously stated, the production is the aggregate of innumerable small producers whose interest it is to conceal the actual amount as much as possible. Knowledge of the deposits is almost equally meager. In Manchuria gold is widespread, both in alluvial deposits, and numerous small quartz veins, from which the former have been derived. These are

worked, the former by panning, the latter by heating and quenching the ore to make it brittle, grinding in stone mills and panning the ground product. The methods used bear striking similarity to those used in Europe in the fifteenth century. The industry does not flourish as much as it might for the country is infested with bandits, who relieve the miner of his winnings, and not infrequently of his life. Since the close of the war policing of the country is rapidly progressing, and the production should rapidly increase also.

In Shantung gold exists at various places, the best known being at P'ing-tu where there is a large and valuable deposit of gold-bearing pyrite. This was formerly worked by stamp-milling and amalgamation, but on reaching the zone of unaltered sulphides, this did not give a satisfactory extraction and the mine is now closed down. What "improved methods" in China have to contend with is shown by the fact that the concentrates after having been re-treated could still be sold to the native farmers, who carried them home and occupied their time in the winter by regrinding and panning them.

Gold is known to occur in Kansu, and there are quartz veins in the upper valley of the Han-ho, in Shensi. There is a small amount of gold washing along the Han-ho in Hupeh. Mines were formerly worked in the southern part of Anhwei. There are deposits in Hunan and Kuangsi, but they are not much worked. There are said to be valuable mines in Kuang-tung. Gold occurs in Ssüchuan, but knowledge of this very valuable mineral province is extremely scanty.

Japan.—A considerable gold production is reported from Japan, partly in connection with copper and silver. Some gold comes also from Formosa, now owned by Japan. Several mines in that island are worked by Japanese companies. A consular report gives the production of the Formosan gold mines in 1907 at approximately 75,000 oz., or $1,550,000, chiefly from the Keeling district.

In Japan itself, gold mining is carried on chiefly in the province of Satsuma where, according to a Japanese authority, there are 197 mines. Many of these have been worked for a long time. Though the precise output can not be ascertained, as the greater number of these are worked after the old style by hand digging and on a small scale with limited capital, the annual yield is estimated at the value of 3,000,000 yen. Besides this no small amount of ore is pocketed by miners who, taking advantage of incomplete surveillance, secretly take the best ore and sell it to buyers of stolen minerals. The important mines are those possessed by the Shimazu family, Mitsui family, Ushiwo Joint Stock Company, Otani partnership, Kichibei Murai, Soichiro Asano and others. The Ushiwo Mining Company, with a capital of 1,000,000 yen, possesses two mines at Isagori, and is doing a good business, declaring a dividend of 30 per cent. last term. At Aira-

gori another mining company has been recently started with a capital of 800,000 yen. This new enterprise is working a modern mining system, using water power electricity for the operation of its machinery.

Korea.—The most important gold mining operation in this country is carried on by the Oriental Consolidated Mining Company, an American concern. Several German companies were organized to operate in Korea, but none of them have been successful. Some new mines are being opened by Japanese companies.

Australasia.

By F. S. Mance.

The mineral industry of the Commonwealth of Australia and New Zealand for the year 1907, while affording grounds for general satisfaction was materially influenced by the vicissitudes occasioned by the fluctuation in the price of the industrial metals. Thus the yield for 1907, which gave such promise during the earlier months of the year, shows a comparatively small gain when compared with the preceding 12 months. At the close of the third quarter of 1907 substantial increases were shown in the value of the output of the principal metals, but these were largely counterbalanced by the lessened values during the remaining quarter. However the aggregate yield makes a fine showing and it can be said, that, with the exception of gold mining, operations were conducted with greater energy and persistency than during any previous period in the history of this Continent.

PRODUCTION OF GOLD IN AUSTRALASIA.
(In ounces fine.)

State.	1901	1902	1903	1904	1905	1906	1907
Western Australia.....	1,669,072	1,819,308	2,064,801	1,983,230	1,955,316	1,794,547	1,697,554
Eastern States:							
Victoria..........	730,449	720,866	767,351	765,596	747,166	772,290	695,576
Queensland......	598,382	640,463	668,546	639,151	592,620	544,636	466,476
New South Wales	216,888	254,435	254,260	269,817	274,267	253,987	247,363
Tasmania........	69,491	70,996	59,891	65,821	73,541	60,023	65,355
South Australia(*a*)	21,946	24,082	21,195	29,177	20,330	24,439	20,000
Total Commonwealth..	3,306,228	3,530,150	3,836,044	3,752,792	3,663,240	3,449,922	3,192,324
New Zealand.........	412,875	458,933	479,715	467,898	492,954	534,616	477,312
Total Australasia.	3,719,103	3,989,083	4,315,759	4,220,690	4,156,194	3,984,538	3,669,636

(*a*) Northern Territory is included with South Australia.

The steady decline in the yield of gold, which has been characteristic of the operations of recent years, continued during 1907. The total value of the gold production was $82,358,207 in 1906, and $75,849,349 in 1907. The yield contributed by Western Australia is an evidence of the magnitude of operations in that State. However, the consistent drop in the

average value of the ore won, the consequent decrease in the profits earned, and the fact that no new mines of importance are being opened up, are matters of great portent when the future of the industry comes to be regarded. In the Eastern States 1907 was devoid of any really important results. The activity in the other branches of the industry thinned the ranks of the gold miners, while investors have been tardy in furnishing the capital essential to the continuance of operations. Prospecting has been comparatively neglected, and, as in Western Australia, the lower grade of the orebodies now being worked in the established mines means that the margin between profit and loss is being gradually lessened. Unless, with the decline in the value of the industrial metals, renewed attention is given to the gold-mining industry, the outlook is anything but hopeful. It may be mentioned that dredging operations have been attended with considerable success, and this branch of the industry is an important factor in the gold production. The returns from New Zealand also show a falling off, which is principally due to the diminished output from hydraulic mining in the South Island. The quartz mines in the Ohinemuri country have furnished an increased output, the principal contributors being the Waihi and Talisman Consolidated. These mines promise to make good headway and the output from the Waihi mine, which reached £875,000 in 1907, is expected, as the result of the improved plant recently installed, to exceed £1,000,000 during 1908. The gold exported from the Commonwealth during 1907, amounted to but £9,442,238, as compared with £14,807,158 in 1906, and £9,416,976 in 1905.

The Broken Hill field was the scene of unprecedented activity during 1907 and the results achieved are possibly the most gratifying in its history. Some idea may be formed of the progress made, when it is stated that the value of the output during 1907 was £5,111,815, as compared with £3,539,596 for the previous year. The dividends paid during 1907 totalled £1,055,500, which brings the amount returned to shareholders by these mines up to £15,333,435. Developments at depth have disclosed the fact that the orebodies continue in great size; and not only are the metallic contents fully maintained, but in several of the mines they show a substantial improvement. As evidencing the possibilities of this field, it may be mentioned that the lode at the 970-ft. level in the South mine has been proved for a length of 1000 ft., and has an average width of 117 ft., the ore assaying silver, 7.5 oz.; lead, 13 per cent., and zinc, 11.8 per cent. All the mines were able to report additions to their ore reserves, and this notwithstanding the heavy output maintained during the year. It is becoming more noticeable, however, that with depth the ore is increasing in density, so that the difficulty and the cost of treatment are enhanced. The production of the Broken Hill Proprietary Company during 1907 was: Silver, 5,481,335 oz. fine, and lead, 69,158 tons. The dividends paid

during this period amounted to £528,000. The extent of the operations of the other companies at the end of 1907 is illustrated by the weekly mill returns, shown in the following statement, the first figure being tons of crude ore treated during the week and the figure in parentheses representing the tons of concentrates produced: South, 4392 (704); Junction North, 1905 (271); North, 2380 (400); Central, 3898 (706); Block 14, 1679 (206); South Blocks, 1850 (325); British, 2703 (468); Junction, 1404 (158); Block 10, 2959 (503); total, 22,170 (3741). Assays of the crude ore varied from 11 to 16 per cent. lead and 10 to 20 zinc, with 6 to 12 oz. silver per ton. The concentrates assayed from 57 to 71 per cent. lead, 3 to 11 zinc and 18 to 31 oz. silver per ton.

The Yerranderie field in New South Wales continues to open up well, and a good tonnage of silver-lead ore was disposed of to the custom smelting works.

PRODUCTION OF SILVER IN AUSTRALASIA.

	1906			1907		
	Ounces.	Kilograms.	Commercial Value.	Ounces.	Kilograms.	Commercial Value.
New South Wales........	8,686,423	270,184	$5,801,749	12,149,672	377,906	$7,937,016
Queensland............	703,087	21,869	469,604	921,497	28,662	601,986
South Australia..........	801	25	535	1,000	31	653
Tasmania..............	2,705,563	84,154	1,807,060	2,850,000	88,647	1,861,820
Victoria..............	33,000	1,027	22,046	31,661	954	20,683
Western Australia.......	....	....	....	....	....	....
Commonwealth....	12,128,874	377,259	$8,100,994	15,953,830	496,200	$10,422,158
New Zealand...........	1,390,536	43,251	928,755	1,562,603	48,603	1,020,802
Total............	13,519,410	420,510	$9,029,749	17,516,433	544,803	$11,442,960

The Chillagoe Company, Queensland, dealt with ore in some quantity from the Mungana mines, but the output had necessarily to be restricted owing to the inability of the company to treat the sulphide ore; but this, it was expected, would be remedied when the Huntington-Heberlein plant is in full swing. Good supplies of ore have been drawn from the other mines in this State and the production for the year amounted to 921,497 oz. silver and 5158 tons of lead of a total value of £187,870.

In Tasmania, the returns from the Zeehan field show a decided increase on the output for 1906, and the capabilities of the mines to maintain a steady production under favorable market conditions has been amply demonstrated. The production of this State for the year is valued at £572,560.

The accompanying table shows the production of silver in Australia. In several cases it is necessarily estimated, as the value of silver and lead is not separated. Western Australia reports no silver, though a small quantity probably occurs in connection with the gold.

Europe.

The only important gold producer in Europe is the Russian Empire, and its output is chiefly from the Siberian mines. In other countries the output is small.

France.—According to *Echo des Mines*, there were in 1907 four producing gold mines in France. The most important is the Carachet, in the department of Creuse, near the foot of the Puy de Dome. The other mines are the Carcassonne, the Lucette and the Montrevault, all in the same mountainous region. The total production is estimated at 1440 Rg. gold for the year.

Russia (By I. I. Rogovin).—According to Russian law, all gold produced in the mines has to be recorded in special official registers; the entries are controlled by the district inspector. But this is a formality, because in practice the inspector is unable to carry out the law. Official statistics are figured from these books, but as the industrial taxes, which are rather heavy in Russia, are fixed in proportion to the yield of gold, it is in the interest of gold producers to conceal the true figures. Therefore, official statistics are always under the real figures. The gold produced is sent, with few exceptions, to the Government assay offices, which, after testing it, purchase it on account of the Imperial Bank. It is then sent to the Imperial Mint, where it is once more tested and then transferred into the metallic fund of the Bank as security for paper currency.

GOLD PRODUCTION IN RUSSIA IN 1907.

District.	Poods.	Fine ounces	Value.
Ural Region	63.70	33,555	$ 693,582
West Siberia	130.98	68,996	1,426,147
Irkutsk Province, including Bodaibo	874.06	460,412	9,516,716
Transbaikal and Amur	695.53	366,366	7,572,785
Total Government assay offices	1,764.27	929,329	$19,209,230
Authorized private banks	449.88	236,975	4,898,273
Total official	2,214.15	1,166,304	$24,107,503
Allowance for gold not reported	221.42	116,630	2,410,750
Total	2,435.57	1,282,934	$26,518,253

It has been customary to make an allowance of 10 per cent., as in the table, in addition to the official returns, for gold not reported. Some authorities think this too low; but it is at least conservative.

The gold yield of 1907 is to be considered a satisfactory one, but the industry, generally speaking, was less profitable, and many prominent mines suffered losses. This was due to increase in prices of labor, supplies, machinery, tools, etc. In order to compensate for the increased cost of

winning the gold producers were obliged to look for richer areas and cheaper methods of working. This explains the extensive development of quartz mining in the southern part of the Ural region, in the Semipalatinsk region of Western Siberia, and the starting of gold mining in the Amur region. In the Ural tailings were also worked. The number of dredges in use increased, there being about 60 in all, chiefly in the Yenisseisk Mining district in Siberia, and in the Ural.

Gold dredges are of recent date in Russia and were put in use only five or six years ago. Bearing in mind that the gold-mining industry has existed in Russia over 150 years and remembering the primitive method of washing auriferous sands, the tailings of old plants alone ought to present a large opportunity for the work of dredges. Moreover, there are in Russia a great many rivers and lakes with auriferous channels, where dredges might be used.

North America.

Canada.—The production of gold in 1907, according to the preliminary reports of the Mines Branch, was $8,264,765, of which $3,150,000 was from the Yukon. The total showed a decrease of $3,759,167 from the preceding year. The production of silver was 12,750,044 oz., an increase of 4,181,389 oz., the larger part of the gain coming from the Cobalt district in Ontario.

British Columbia (By E. Jacobs).—The accompanying table gives the production of gold and silver in this Province for the last four years.

GOLD AND SILVER PRODUCTION OF BRITISH COLUMBIA.

	1904		1905		1906		1907	
	Oz.	Value. (*a*)	Oz.	Value. (*a*)	Oz.	Value. (*a*)	Oz.	Value. (*a*)
Gold, placer....	55,765	$1,115,300	48,465	$ 969,300	47,420	$ 948,400	41,400	$ 828,000
Gold, lode......	222,042	4,589,608	238,660	4,933,102	224,027	4,630,638	196,179	4,055,020
Total gold....	277,807	$5,704,908	287,125	$5,902,402	271,447	$5,579,038	237,579	$4,883,020
Silver...........	3,222,481	1,719,516	3,439,417	1,971,818	2,990,262	1,997,226	2,745,448	1,793,519

(*a*) Placer gold is valued at $20 per oz.; lode gold at $20.67 per oz.; silver at average market quotations.

The production of placer gold shows a smaller total than for any other year since 1898. The decrease in production was general throughout the placer gold mining districts of the Province; the larger losses having been Cariboo about $150,000 and the Cassiar—chiefly Atlin—about $100,000. In lode gold the more important decreases were in the Boundary district—about $500,000—and in West Kootenay—about $250,000.

In the Cariboo, the announced decision of the Guggenheim companies not to proceed further with construction of their water supply system

caused general disappointment throughout the district. The Guggenheims' engineers are stated to have reported unfavorably regarding average gold values obtained when testing the gravel, so expenditure on contruction has been stopped. Total gold recovery from this property from 1897 to date is $1,062,700.

The Cassiar district includes the Atlin, Liard and Skeena divisions. In the Atlin gold dredging has been abandoned; placer mining by individuals has steadily decreased; hydraulicking by the larger companies in 1907 resulted in a smaller recovery of gold than in 1906. In Liard division, the Berry Creek Mining Company's results from hydraulicking on Thibert creek were much below expectations. In Skeena division, a few thousand tons of ore were shipped to the smelter at Hadley, southeast Alaska, from the Outsiders' group, on Portland canal, and developments on two or three other properties are promising.

The three chief silver-producing districts for the year were West Kootenay, East Kootenay, and Slocan, and they stand in that order as regards relative quantities produced. West Kootenay's proportion was about 1,200,000 oz.; East Kootenay's 900,000 oz.; Boundary about 675,000 oz., and the Coast about 80,000 oz.

In West Kootenay Ainsworth camp had several mines at work; some 1100 tons of silver-lead ore were shipped and important development work was done. In the Slocan—at Whitewater—there was milled a lot of ore from which silver, lead and zinc concentrate were produced. Around Silverton, the Hewitt, Vancouver group, and Standard each made good progress both in development and production. In Slocan City division several mines were active, notably the Arlington and Ottawa.

Of the mines in Nelson division La Plata made the best record, its output of silver and lead having been appreciably large. The Silver King (copper-silver) and Queen Victoria (copper) were both shippers in quantity. Rossland mines made a production about equal to that of 1906—about 280,000 tons, practically all from the Le Roi, Centre Star-War Eagle group, and Le Roi No. 2. In Revelstoke and Lardeau sections mining was not active. The largest producers were the Eva (gold) at Camborne and Silver Cup (silver-lead) in Ferguson camp. The Silver Dollar, near Camborne, completed a small stamp mill and commenced crushing ore.

A considerable quantity of silver came from the copper mines of the Boundary district, where mining and smelting were active during 10 months of the year.

Nova Scotia.—The production of gold in this Province was steady during 1907. There were no important changes made. At the Richardson mine preparations were in progress to enlarge the mill, and to carry on work at a greater depth.

Ontario.—The important silver production of the Cobalt district is

treated under the head "Nickel and Cobalt," where a review of the district will be found.

Quebec.—There was some further prospecting in the Chibogamo district, with promising results. Prospecting was also carried on in the region of lake Temagami, on the Ontario line. The commencement of work in Northern Quebec, however, depends upon the building of a railroad through that section.

Yukon Territory (By John Power Hutchins).—The gold production of the Klondike district in 1907 was little more than half that of 1906. The accompanying table shows the gold output of the Yukon Territory for a period of years. It includes other districts besides the Klondike, but their yield is comparatively small.

GOLD PRODUCTION OF THE CANADIAN YUKON.

Year.	Amount.	Year.	Amount.	Year.	Amount.	Year.	Amount.
1896......	$300,000	1899.......	$16,000,000	1902.......	$14,500,000	1905.......	$7,000,000
1897......	2,500,000	1900.......	22,250,000	1903.......	12,250,000	1906.......	5,000,000
1898......	10,000,000	1901.......	13,000,000	1904.......	10,350,000	1907.......	3,150,000

These figures reveal a marked similarity between this and other placer districts not only in respect to the rapid increase of the annual output to a maximum a few years after the discovery of the placers, but also in the rapid decrease in the output after the maximum figure had been reached. It is of interest to note that in both California and the Klondike the annual production reached a maximum the fourth year after discovery. These figures were more than $80,000,000 for California and more than $22,000,000 for the Klondike.

The Klondike is essentially a placer district. As it has no paying lode mines, the exhaustion of the placer areas results in smaller annual output. Because of this fact and others stated later it is not likely that there will be a material increase in the gold output. In the Klondike the probability of finding extensive new deposits of pay-gravel is remote but there is the possibility that developments will extend the boundaries of the deposits already exploited.

The "White Channel" is what is left of the ancient creek beds; it has a course approximately parallel to the present creeks but at an elevation from 100 to 300 ft. greater. Where it has not been entirely eroded, this channel usually appears with its rim completely removed; locally it is called a bench deposit. There is one notable exception in the occurrence of the "White Channel"; this is the deposit between Lovett gulch and the Klondike river, for there the two rims of the "White Channel" are intact for several thousand feet. In the Klondike there are no "back channels," "deep channels" or "break-outs" such as occur in California.

Another way in which dying placer regions may be revived is by the introduction of machinery and large-scale operations to work gravel too poor to be profitably worked by hand. Most of the Klondike claims have been bought by a powerful corporation, and preparations are now under way to work these claims on a large scale, using mechanical devices to supplant most of the hand labor. A part of the decrease in gold output is due to the fact that the Klondike district is now in a transition period and much ground is now idle while these preparations are going forward.

The snowfall of 1906-7 was light, only about 18 in., but, as it thawed slowly, it was possible to make full use of the small amount of water available; considerable of this water was used in ground-sluicing overburden. The season was a very dry one, in all months except September. The time of the freeze-up was normal. During the winter only a little drifting was done on Dominion, Sulphur and Quartz creeks. None was done on Bonanza, Hunker and El Dorado creeks. A small amount of rich ground was mined on Hunker creek, the average content of which is said to have been about 25c. per pan.

There was little in the way of novelty in mining methods to note during 1907. The Klondike from the time of its discovery, has been notable for the remarkable variety of its mining devices. At first, when all were engaged in solving the tremendous obstacles concomitant with the rigors of a climate having a minimum temperature of 60 deg. F. to 70 deg. F. below zero, the variety of mining machines used was bewildering. Even in 1907 in mining creek-bed gravel by the open-cut method, the following means of transporting pay-gravel to the sluices were commonly used: Shoveling to platforms, then to sluice; shoveling into wheelbarrows, wheeling to bucket, raising on inclined cableway to sluice; shoveling into buckets, hoisting by derrick to sluice; shoveling into cars, hauling on inclined track to sluice. But shoveling into wheelbarrows, wheeling bucket and hoisting on inclined cableway to sluice was the method of transportation most generally used. Less frequently the following methods are or have been used: Steam-shoveling direct into sluice; steam-shoveling into cars, hauling on inclined cableway to sluice; steam-shoveling into skip, hauling on cableway to sluice; steam-shoveling into sluice, sustained on same car with shovel; steam-shoveling into sluice, sustained on another car; hand-shoveling upon a troughed belt, conveying to sluice; excavating by ground-sluicing or with water under pressure from gravity or from pump; sluicing to centrifugal dredging pump; elevating to sluice; and steam scraping to sluice. In fact with the exception of hydraulic elevating, which a dearth of water prevents, there is hardly a mechanical device for placer mining that has not been used in Klondike.

Dredging in the Klondike district has been successful only where unfrozen or very rich frozen gravel was worked. Three powerful dredges

began operation on lower Bonanza creek, but the experience there has been most discouraging. An attempt was made to dig the frozen bottom gravel, which must be mined to recover the major part of the gold content; at least such is the case in the Klondike district, where a great vertical concentration has taken place; consequently most of the gold occurs either near, on, or in the bedrock. This procedure resulted in tremendous wear, tear, and damage to the excavating apparatus, although only a small amount of excavating was done; bucket lips had to be ordered by the carload by express. Steam thawing was then undertaken, but this was not rapid enough to prepare sufficient ground to keep two dredges going (the third having been partially dismantled to furnish parts to replace those worn out and broken by the other two while trying to excavate the frozen ground). The dredging outcome was altogether discouraging.

As a general rule it may be said that frozen ground cannot be dredged. This does not mean that it is impossible to dredge it, but that it is usually more economical to exploit it in some other way. Dredges have been applied in the Klondike by merged interests where the less spectacular and more humble methods of the one-man enterprise would probably be much more suitable and profitable.

Were the alluvium of the Klondike creeks unfrozen, dredging could be carried on at a large profit, for the other conditions are generally favorable to dredging. The gravel is fine, clean and of moderate depth. It generally rests on a favorable bedrock, though there are areas where blocky schists carry considerable gold in their crevices to such a depth that it cannot be recovered by dredge buckets.

There is probably less than 3000 acres of so-called dredging ground in the Klondike; of this more than 50 per cent. is partially or totally frozen, and therefore of dubious value for dredging. In 1907 there was about as much ground dredged in the Klondike as during the season of 1906.

The cost accounts of the three new 5-cu.ft. dredges that worked spasmodically in the frozen gravel during 1907 are not available for publication, but it is known that they are very high. As in previous years it was found that on a large dredge working mostly in unfrozen ground the operating cost was about 15 to 20c. per cubic yard.

During the year none other than narrow and shallow pay-streaks were found, for, although the vertical and horizontal concentration has been great, areas of local enrichment such as those at the confluence of a small rich creek with a larger stream have not been found. This is an interesting feature, but likely to lead to misleading conclusions if not given proper weight in considerations of the area of local enrichment.

There were no modifications of dredge design, construction and manipulation during 1907. It was observed that the stacker dredge stood cold weather better than the sluice dredge; also that the steam dredge had a

similar advantage over the electric dredge. Electric dredging will probably show a saving of 30 to 40 per cent. over steam operated dredges, particularly in case there be a central hydro-electric generating station furnishing power to several dredges; besides, it takes about 20 per cent. less labor to operate electric dredges.

Eight- and 12-hour shifts are worked; the approximate gross wages per day are: Foreman, \$10; winchmen, \$7.50; firemen, \$6.50; deckhands, \$5.50; roustabouts, \$5.50.

Considerable work was done upon the large ditch, flume, and pipe-line which, when completed, will furnish 125 sec.-ft. of water; much work was also done at some of the claims preparing for hydraulicking with this water. Pits about 20x30 ft. and 10 ft. deep have been excavated in bedrock at No. 3 above Discovery and at No. 24 and No. 30 below Discovery on Bonanza creek. Electric bucket-elevators will operate in these pits. Material sluiced to them by water under gravity head will be elevated together with the water to sluices set at such a hight as to give sufficient grade and dump room. In 1904 this type of plant was tried and discarded. This type of machine is very immobile and therefore not well suited to working in the shallow creek beds of Klondike.

The plant installed at No. 60 below Discovery on Bonanza and consisting in part of a steam-shovel and belt-conveyer was moved to Poverty bar, a bench deposit of Bonanza creek, opposite No. 14 below Discovery, but it failed there also.

A dam about 60 ft. high and planned to hold about 158,000,000 gal. was finished in 1907. This structure was begun in the summer of 1906, but its foundation was not placed deep enough. The core was built of 8x8-in. timber and a No. 14-gage iron pipe was placed through the core for an outlet. No masonry work was built around this pipe. Loose dirt and broken rock from the adjacent hills were tamped with water around the core. This work continued with several hundred men until about Dec. 1 of 1906. Much of the material dumped into the dam after Nov. 1 was badly frozen, and so could not be properly tamped. Work was resumed in the following spring, but when the spring thaw came the frozen material in the dam settled, crushing the outlet pipe, and twisting the timbers of the core. These difficulties are stated in order to show some of the problems of earthwork construction in the far North.

As the mining season of 1907 was very dry, only about 1,250,000 cu.yd. were sluiced, or only about 50 per cent. of the yardage washed in 1906. A duty of more than 5 cu.yd. per miners' inch per 24 hours was attained at several properties; operating cost was more than 20c. per cu.yd. No hydraulicking with water under pumping pressure was done this year.

Wood hauled on runners in 16-ft. lengths cost \$10 to \$16 per cord deliv-

ered at the mine; that hauled on wheels cost $16 to $20 per cord. This cost is largely affected by the length of haul. Coal was used but in small quantity; it came from mines outside of the Klondike district.

No new Government wagon roads of importance were built in 1907. Existing roads were well maintained and it is possible to haul about one ton per animal on them. Freight rates from Seattle to Dawson were about $60 per ton, the same as in 1906. Passengers' rates in the Klondike for railway, river steamer and stage were, as in 1906, about 20c., 11c., and 25c. per mile, respectively.

In the early days of the Klondike most of the gold was produced by drift mining. Shafts were sunk and drifts from them were run in the pay-gravel, which was hoisted and then trammed to the pay-dump. During 1907 there was very little of such mining done, as nearly all the ground, formerly thought suitable for exploitation by this method, is now being reserved for future working by hydraulic or mechanical methods.

Central America.—In the Central American countries there was some increase in mining in 1907, largely due to the introduction of capital from the United States. This movement, however, is still on a moderate scale. In Costa Rica the mines worked were the Abangarez Goldfields, the Esperanca (late the Boston) mine in the Abangarez district, and the Colburn mine at Pozo Azul near Chomes. Development work is proceeding at the Montezuma mine in the Barranca district and at the Machuca mine in the Aguacate district. The amount of bullion exported from Costa Rica during 1906 was valued at $538,472; in 1907 there was an increase of about 10 per cent.

In Guatemala, the railroad extensions completed in 1907 are expected to open up several new mining districts. In Salvador there was a production of about $1,300,000 gold in 1906 and 1907.

Mexico (By A. Van Zwaluwenburg).—The production of gold in Mexico for 1907 showed a small increase, while in silver there was a decreased output. There was reduced activity in mining throughout the year, owing in the early months to the abnormally high cost of fuel and supplies and in later months to the rapid fall in the prices of metals. During the first half of the year the congestion of traffic on the railroads rendered transportation difficulties so serious as to interfere with development and in many cases to cripple production. When later activity in the production of ore began to fall off, supplies began to move more freely, and at the beginning of 1908 the railroads were able to handle normal tonnages promptly and without difficulty.

The stringency in the money market began to be felt earlier in Mexico than in the United States. The prolonged drouth during the months which in normal seasons furnish rains for the growth of crops, transportation difficulties and uncertainty as to the Government's attitude in regard

to freight rates on ore, all seemed to intensify the feeling of apprehension and to retard and to check the enormous development of which 1906 gave promise.

The end of 1907 found the industry in a state of suspense. In a large proportion of the mines production was curtailed if not stopped altogether, for at the then current prices of copper, silver and lead, many of the large companies, which operate on a small margin at best, were unable to work at a profit. Among the stronger corporations energies are directed toward improvement of equipment and getting ready for renewed activity when the expected return in the market brings the normal demand.

For years the gold output of Mexico has been obtained chiefly as a by-product in the production of silver, copper and lead. Of late years there have been a greater number of operations carried on primarily for the gold content of the ore than in former years. Among the more notable gold producers were the mines of El Oro district and the Dolores. New discoveries were reported in western Chihuahua, Sonora, Oaxaca and other out-of-the-way districts. The proportion of gold in the ore which reached the large smelting centers experienced a normal increase, and it is possible that the reopening of the old Peñoles mine near Mapimi will in 1908 add materially to the gold production of the country.

Silver is still, as it has been for many years, the chief object of mining in Mexico. All the old camps—Pachuca, Guanajuato, Zacatecas, Parral and others—continued to produce freely, except during the last month when the mines dependent upon the custom smelting works for reduction of their ores almost ceased operations. Pachuca and Guanajuato, in which a large proportion of the ore is treated in local mills, suffered no apparent loss of activity, but in December Parral experienced an almost complete shut-down.

There were a number of new mines which approached or reached the producing stage during the year, notably several mines in the Taviche district in Oaxaca, and La Republica mine on the Kansas City, Mexico & Orient Railroad in Chihuahua.

Mexico is wonderfully rich in streams which may be utilized for the production of electric energy to be distributed to mining districts. Guanajuato, Pachuca, El Oro, Etzatlan, Ocotlan and Torreon are all within the radius of easy transmission from power stations on neighboring streams. Chihuahua is situated too completely upon the central plateau to offer much inducement for enterprise of this sort, but practically all the other mining States are potential producers of abundant water power. Many concessions for hydro-electric power production were granted during 1907, and several are in process of development. Torreon is to be supplied from an installation on the Nazas river; Guanajuato and El Oro have already drawn power from transmission lines for months; and corpora-

tions have been formed to furnish electric current to the mines of Durango, Jalisco and Oaxaca.

At the close of 1907 labor was plentiful, the failure of crops having driven many agricultural laborers to the mines.

West Indies.—Attention has been called by F. Lynwood Garrison (*Eng. and Min. Journ.* Sept. 14, 1907) to the probable existence of rich gold placers in Santo Domingo; but it is hardly possible that these can be profitably worked until a stable government is established in the island.

South America (By John Power Hutchins).—The year 1907 was not a particularly notable one for the mining industry of South America, and this enormous area, almost equal as it is to North America, still lies undeveloped and awaiting investigation. There are several good reasons for this condition, the principal ones being bad climate, poor transportation and unstable government. Bad transportation and unstable government are largely a result of bad climate. Parts of South America lying in the south temperate zone, and those sections located at such elevation in the tropic zone as to have temperate climates, are in a much more prosperous condition.

South America with the Cordilleras and their numerous ramifications is extremely mountainous. This makes transportation a serious problem, for mule-back freighting of mining machinery is costly and hazardous.

Tropic South America is largely a region of one-man operations. A fair example may be found in the mining operations of Dutch Guiana. The only operation carried on successfully by a merged interest was one in which no machinery at all was used. This was simply a number of one-man enterprises combined, and the fact that the ground mined was rich had much to do with its success.

The year witnessed no great progress in placer mining. The placers of South America are credited with a total gold product since 1492 of nearly $2,000,000,000. This large sum is about one-fifth of the total gold production of the world. The total annual gold production of South America is at present only about $10,000,000, most of which comes from placers worked in a desultory way. There have been about 20 failures in dredging in tropical South America and not one success. Although the bad climate has had considerable to do with these failures, it is also true that a first-class modern dredge has never been operated in this region.

The question suggests itself as to the means by which the gold production of South America has been brought to such a large total during the history of the region. The answer is briefly as follows: In the early days the conquering Spaniards enslaved the natives and forced them to work the placer mines. The food for these miners was raised and transported to them by other slaves. Labor cost was thus very small and it was possible to work very low-grade material. At present these low-grade placers could

be exploited profitably only by hydraulicking or dredging on a large scale.

There are about 15 dredges in Terra del Fuego, and of these about half are said to be failures. Conditions are not favorable for work of this kind on the extreme end of South America; the ground is rocky and it is necessary to import coal at a cost of more than $5 per ton, and work may be prosecuted during about nine months of the year. Dredges operating in Colombia, Dutch Guiana, Ecuador, British Guiana and Brazil have generally not been profitable. This statement seems discouraging; but when it is recalled that most of these dredges would have been failures had they been installed under ideal conditions, owing to faulty design and weak construction, the outlook is not without promise.

The continued search for ground suitable for exploitation by the dredging method resulted in more extensive investigation of the placer deposits of South American countries. Drill prospecting is carried on with particular vigor in Colombia, Dutch Guiana, Peru, and Ecuador. A large volume of material which is said to have a maximum thickness of nearly 100 ft. and to carry gold in paying quantity, is located near Zaragoza, in the bed and on the banks of the Nechi, a tributary of the Magdalena river.

The placers of Peru are attracting considerable attention. It is thought that some of the large volumes of auriferous gravel can be exploited successfully by hydraulic mining. These deposits occur at an elevation of from 2000 to 4000 ft. Some of the placers of Brazil carry both gold and diamonds. A dredge has been installed on the Jequitinhonha river to handle such material.

Lode mining is being carried on in Peru, Colombia, Ecuador, British Guiana, Brazil, Venezuela, Chile, Uruguay, Argentina and Bolivia. Peru is the center of this industry, the chief products from its mines being copper, silver and lead, named in the order of their importance. The recent drop in the prices of these metals has curtailed production, the cost of which is high, largely because of adverse transport conditions. Colombia has a few profitable gold mines. The most notable of these is the Frontina & Bolivia, near Remedios in the department of Antioquia.

The only lode mining in Ecuador is in Zaruma. Since the introduction of the cyanide process, about three years ago, operations in this locality are said to have been profitable. Only in the extreme northern and southern parts of Ecuador is any mining done, the intermediate section being covered by lava flows. The eastern slopes of the Andes which constitute the headwaters of the Amazon river are not so obscured and gold is mined from placers by natives. This region is a vast unexplored wilderness, penetrated only by rubber hunters and traders. It is almost inaccessible and will probably remain unexploited for a long time.

But one lode mine of consequence, the Peters mine, is operated in British Guiana. This property is owned by American capitalists. High-grade ore makes it possible to operate successfully even though cost of mining is, on account of climatic conditions, high.

Progress in Gold-ore Treatment During 1907.

By Alfred James.

Fine sliming and the treatment of slimes are again the most interesting features of the year's progress—even former exponents of coarse crushing are now converts to, and profit-takers of, modern slimes practice. From Eastern Asia to Western Australia and from South Africa to North America sliming methods are preëminent and have modified the other methods or processes employed in the industry, such as crushing to tube mills and roasting ores ground to very fine particles.

Crushing.—We heard less in 1907 of "Pans vs. Tube Mills." Experiments in the Transvaal appear to have resulted in favor of the latter, and many more tube mills have been installed. With the heavy stamps previously referred to in these notes the Luipaards Vlei appears to have been able to output 8.5 tons per head per diem, but in Rhodesia I am advised that 10 tons per head is being obtained in more than one instance. In Rhodesia the practice differs from that on the Rand; pans are found to be more suitable, as they are cheaper to install, make more efficient, compact and convenient units, and do not require so much power; consequently we hear of comparatively few tube mills operating in Rhodesia. The pans thus used are of the West Australian type.

But tube mills have unquestionably been improved in their *modus operandi.* Linings now last much longer and instead of having to lay off a mill every three weeks or so for a week for a new lining a mill can be run for three to six months with a lining placed in position in from two to three days. "Automatic" linings such as those of El Oro, where pebbles are supposed to fix themselves in rebates cast in iron liners, do not yet appear to have proved themselves entirely satisfactory, and changes have recently been made in the shape of the recesses, which now have their sides chamfered to each other.

The use of quartz or other reef matter to be crushed in place of imported pebbles has much increased, and this and the utilization of reef matter and local chert or other hard rock in linings of the Barry or Brown type have very materially reduced the cost of operating tube mills. In Africa a mill is started with a charge of pebbles containing say 10 per cent. of reef matter (stones), then in regular work it is found possible to reduce the daily feed pebbles by 5 per cent., adding reef matter (stones) to balance until the daily feed consists of reef matter only.

Slime Treatment.—It is in the treatment of the slimes produced by tube mills and otherwise that the progress of 1907 was most manifest. It may be regarded that the decantation process, which originated in Africa, and has so long survived there, has now had its day. Designed to treat material of low value, at that time not amenable to any method of treatment known to the industry, it resulted in profits being realized from material which otherwise must have run to waste. But the huge and expensive plant and low extraction had to give place to more modern automatic methods capable of cheaper equipment and higher saving, and already such well-known workers as J. R. Williams, W. A. Caldecott, E. J. Way, and H. S. Denny have been looking into methods employed elsewhere, notably the Ridgway, Butters, Burt and Merrill.

In my notes of last year reference was made to the new Denny plants at the Meyer & Charlton and the New Goch. The Denny brothers severed their connection with these two mines shortly afterward, but in spite of their absence filter-pressing appears to be regarded as successful by the heads of the group, although much doubt has been expressed as to the wisdom of running cyanide solution through the mortar boxes. George Albu has criticized this as making it difficult to obtain the assay value of the mortar-box product, but I assume this point would not be regarded as serious if Mr. Albu were convinced he were obtaining a higher recovery of his gold. This discussion has been fully dealt with in another place.

On the Rand, however, the Denny methods have not been followed. Rhodesia on the contrary, having investigated the results, has had quite a Dehne filter-press boom, plant after plant having been installed during 1907.

Vacuum Filters.—But undoubtedly filter-pressing even with Dehne presses must give way before the suction filters of which so much was heard in 1907. Cheap and efficient, the daily tonnage handled by vacuum filters is increasing by leaps and bounds.

Of the various vacuum filters on the market I must unhesitatingly refer first to the Ridgway. This differs from the filters of the basket type in being more rapid, working with thinner cakes, giving better washing and emptying itself (automatically) in more cleanly fashion, than the basket filters. Working as it does with the whole cycle of operation *complete in sixty seconds* on normal slime pulp, it will be seen that difficulties of keeping material on cloths during transit of frames or during emptying and filling of tanks, which arise in other methods, are entirely avoided, while the washing of so thin a layer ($1\frac{1}{4}$ to $\frac{3}{8}$ in. thick) is much more rapid and complete than the washing of a cake of double or treble the thickness. The Ridgway is thus able to work with a much smaller area of filter cloth for any given capacity. On the other hand this small area of filter cloth per machine may be a serious limitation in the treatment of slime con-

taining a high percentage of moisture, for which, as for "puggy" slime, gray filtering surface is necessary.

During 1907 a 500-ton-a-day plant was put into successful operation at the Great Boulder in West Australia, and I understand another plant of similar capacity has already been started. Plants of similar and smaller capacities have been erected (or are in course of erection) in Mexico, India, South Africa, Eastern Asia, etc., and altogether we hear much of the Ridgway filter at the present moment. The official figures of the West Australian Chamber of Mines show the Ridgway to be working under expensive local conditions (purchased power, dear labor and supplies) for under 4d. per ton treated; in Africa the cost should be little more than half this.

Butters-Cassel vs. Moore.—The Butters-Cassel filter has been installed very largely in Mexico as well as in the United States, not perhaps because it was the best type of vacuum filter—the recently published correspondence shows the pioneer Moore to be at least the equal and probably the superior—but because it was pushed by a man of repute and of great energy who was recommending the best thing he could get hold of. But in the recent correspondence in the technical papers it appeared to me that the only persons writing in favor of the Butters-Cassel filter were those who were, or had formerly been, associated with Butters, while the Moore was recommended time and again by persons in no way associated with Moore, who had installed it after investigation and who were apparently in no way connected—not even by paying royalties—to Moore or his associates. Certainly if the Moore process had been run with half the energy, experience, knowledge and skill of the Butters I cannot imagine the latter type of filter to be employed at all. For it is obvious that the hoisting of a basket of frames is a much neater and better and quicker expedient than the pumping in and out of pulp and solution or wash, each successive charge being mixed with the residue of the previous charge whether of pulp or of solution—if used—or of water, and it is within the knowledge of men experienced in slimes filtration that the maintenance of a vacuum in a basket half immersed, or partially immersed—for say four periods of 10 minutes each during each cycle of operations—does not make for good results. The lower portion of the cake is being thickened by pulp deposit while the upper is cracking from the inrush of air, an unequal cake is formed and the washing must be unequal and there seems to be a very great waste of water in discharging the cake after filtration. From the published correspondence it seems to be laid down without serious contradiction that the Moore is more accessible and open to inspection and that it does more work for a plant of given cost or in a given time.

Further modifications of the basket filter have appeared in the form

of enclosed filters worked by pressure or vacuum, such as the Blaisdell, Burt, and Kelly. This form of filter is scarcely new and when tried some years ago proved itself unhandy, liable to freeze or choke, difficult to wash thoroughly and difficult to discharge—in a word it seems to be an attempt to work in the dark. Of course modifications of the standard immovable inclosed type are made; one runs the frame into a cylinder or runs the cylinder away from the frame; this makes the operation more accessible, but so far does not appear to be as successful as the pioneer basket type.

Merrill's Plant.—Reverting, however, to filter presses, Merrill has at last got his big plant at work after three years' labor. It appears now to be working to nearly its full capacity—according to my recent information—and to be treating slime at a remarkably low cost. Advantage has been taken of every natural condition to make a successful working plant, and using static pressure to fill the presses and wash their charges, the Merrill costs are probably as low as any slime treatment method yet put in practice. But my information is that with such a method the satisfactory treatment and hydraulic discharge of thick cakes in huge filter-presses are possible only with crystalline or granular slime pulp and with a great waste of water. In a word it appears that successful results may not be anticipated by the process on ordinary slimes pulp in ordinary goldfields under ordinary economic conditions; the thorough washing of 4-in. cakes is feasible only with crystalline, granular or readily permeable slimes pulp.

The successful handling of slimes pulp has directed more attention to the solution of the gold contained therein. Last year I referred to the system of treatment based on feeding solution or wash water at the bottom of a tank, the slime contents of which were in state of gentle agitation, the idea being an overflow of clear solution containing the gold content of the charge. W. L. Holms of Mexico City (late of Consolidated Gold Fields of South Africa) brought this to my notice some years ago, but it did not appear to make headway. Bewick, Moreing & Co. were working it in 1906 in West Australia, but evidently without commercial success; in 1907 Adair Usher boomed an apparently similar process in South Africa. Possibly some metallurgist at the Ferreira Geldenhuis Estate or elsewhere may make something tangible out of the idea, but it looks as though the only result would be to call attention to the extra extraction obtained by the increased agitation of the pulp, and in this connection the work of the Brown agitator ought to be investigated carefully.

The Brown Agitator.—This apparatus, introduced into New Zealand at the Komata Reefs, depends on the principle of the lessened specific gravity of a center column into which air has been introduced at just such a pressure as will overcome the weight of the column of water at the point of

introduction. There is no question of a jet of air circulating solution on the principle of the injector, but merely the physical lightening of the weight of a column by the displacement of a small proportion of the water by air. Brown uses long narrow vertical (cylindrical) tanks of say 40x10 ft. or 55x15 ft. with a center column of 1-in. diameter per 1 ft. of tank diameter. Into this central tube or column the air is introduced and agitation is so effective as to lift stones at perhaps the smallest horse power per ton agitated of any efficient mechanical agitator. The power taken appears to be from $2\frac{1}{2}$ to 5 h.p. per 50 tons (slime) pulp charge, which is the slime content of a 40x10-ft. tank. By this method the advantage of the accelerating action of air agitation in certain ores is obtained at a small cost. The use of these tanks has spread to the Waihi—the manager of which speaks very highly of these agitators—and Waihi Grand Junction in New Zealand, and a number have been installed in Mexico under the name of the Pachuca agitator.

Cyanidation of Silver-Gold Ores.—The progress of cyanide treatment for the silver-gold ores of Mexico has been one of the features of the slime-boom. Chihuahua has been adopting cyanide with avidity, and Pachuca appears to be now coming into line. Of course El Oro has done pioneer work and the MacDonalds of Guanajuato have been in the forefront from the first. In almost all instances the practice is similar, tube-mill sliming and treatment of the slime pulp at first by decantation, but now by basket pressure or Ridgway filter. Very high extractions are claimed, but having regard to the natural refractoriness of silver sulphide ore—in some instances as at Chinacates associated with pyrites, chalcopyrite, galena and blende—one is impressed by the necessity of having recourse to the most modern practice, whether for getting the silver and gold into solution or for recovering the solution from the pulp for precipitation of the precious metal.

Cyanidation During 1907.

By Charles H. Fulton.

Introduction.

The cyanide process was prominently in the eyes of metallurgists generally, and decided progress was made, notably in the increased application of fine grinding and slimes filtration. The number of slime filters now on the market—filter-presses, pressure filters, and vacuum filters, both of the intermittent and continuous type—is large and some possess much merit. It is still to be decided which is the best type of vacuum filter, and the struggle will go on during the present year. It would seem as if the continuous type of vacuum filter as against the intermittent type of basket vacuum filter should have a field. A noticeable

feature in cyanide practice during the year was the introduction, wherever it was at all feasible, of crushing in dilute cyanide solution as offering many advantages which have been outlined in other volumes of The Mineral Industry. Metallurgists are still in doubt of the advisability of the method when practiced in conjunction with plate amalgamation, but the difficulties arising in this connection will probably be solved. Remarkable progress was made in the cyanidation of silver ores in Mexico and in Nevada, and the older processes have been practically displaced. The practice is very uniform, generally considered, and is described under milling practice. While in these descriptions the decantation method of slime treatment is apparently in vogue, this has been displaced recently in many cases by some method of pressure or vacuum filtration. The great success of the Merrill filter-press plant of the Homestake company during the year put the press before the metallurgical world as the coming filter-press wherever water conditions will permit it. It is of course recognized that its use is confined to certain types of slimes only.

Tube milling is taking on the features of standard practice and is the preëminent fine-grinding method, although pans have a field, notably in Rhodesia and Kalgoorlie. Blaisdell apparatus, for charging and discharging sands has a wide use and proves efficient wherever tried. It does not compete with cheap labor on the Rand, but this seems to be due in part to conservatism. The apparatus is widely used in Mexico, where it has special application in connection with the double and triple treatment of sands practiced there as a necessity to gain extraction on silver ores. The Dorr scraping classifier is being quite extensively employed in mills in this country and Mexico, and gives a very good classification.

General Milling Practice.

The New Metallurgy on the Rand.—G. A. and H. S. Denny[1] during 1906 and 1907 introduced several decidedly new features into the cyanide practice of the Rand but briefly referred to in the former volume of The Mineral Industry.[2] They were the first to introduce tube mills on the Rand for the regrinding of coarse sands and concentrates (the spigot product of the first classifiers), sending this product to be treated with the slimes, instead of cyaniding the same by long leaching treatment; or concentrating the pulp, after amalgamation, on vanners or Wilfley tables, and chlorinating the concentrates. There are at the present writing about 64 tube mills on the Rand, of which usually two go with 100 stamps or three with 200 stamps, thus accounting for from 3600 to 4000 stamps out of the 7000 total on the Rand. All of the more recent installations are provided with tube mills. The Denny brothers are also largely respon-

[1] "Recent Innovations in Rand Metallurgical Practice." *Eng. and Min. Journ.*, Vol. LXXXII, 1217. *Journal* South African Assn. of Engineers, June, 1906.

[2] Alfred James. *The Mineral Industry*, XV, 407.

sible for the introduction of the Dehne filter press, beginning to replace the decantation process. At the present writing five large mines are putting in filter-presses, aside from three plants already using them. (Meyer & Charlton, New Goch, etc.) Another prominent feature of the "New Metallurgy" is the circulation of the cyanide solution, e.g., crushing in cyanide solution, practiced largely in Australia, Mexico, and in the United States. The essentials of the new process may be stated as follows: 1. Regrinding the coarse concentrates and sands in tube mills. 2. A second amalgamation of the ground product. 3. Treatment of the reground product with the slimes. 4. Filter-pressing of the slimes. 5. Automatic return of all solutions to battery storage tanks and crushing in the solution. 6. Handling all slimes in conical tanks by gravity alone.

The accompanying plan shows the scheme of operations at the Meyer & Charlton Gold Mining Company, Ltd., treating 400 tons per day:

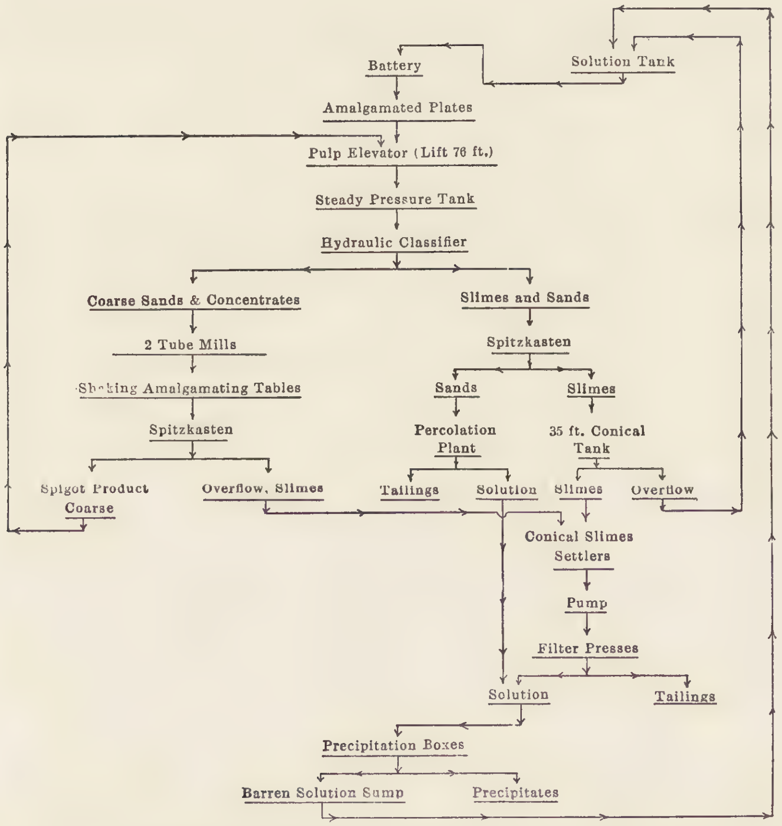


FIG. 1.—SCHEME OF TREATMENT AT MEYER & CHARLTON CYANIDE MILL.

The amount of solution per ton of ore is 6.6 tons of 0.03 per cent. KCN and 0.004 per cent. protective alkali. By means of crushing in solution the gold is readily dissolved, as much as 98 per cent. of the gold recovered from the slimes being in solution before they are settled, and 70 per cent. of the gold in the sands before percolation starts. When the pulp issues from the mortars 12.65 per cent. of the total gold in the ore is in solution.

This is similar to the experience of crushing in cyanide solution in the Black Hills. The gold in the Rand ores is, however, much more readily soluble than in some other ores, and frequently extensive agitation and long treatment are essential in order to recover the last 20 per cent. of the total extraction. Taking the figures of one month (10,740 tons) with an ore value of $10.90 per ton, the amount extracted on plates was 43.85 per cent.; on shaking amalgamated tables (after tube milling) 3.01 per cent.; in sand treatment, 9.76 per cent.; from slimes and sands in transit, 38.67 per cent.; a total of 95.29 per cent. The tailings contained $0.51 gold, or 4.698 per cent. of the ore contents. Of the total ore milled 52.3 per cent. is reground in the tube mills. Approximately 70 per cent. is treated as sands and 30 per cent. as slimes.

A feature of the plant is a large 35-ft., conical steel tank for the settling and thickening of the slimes and automatic decantation of solutions, and eight smaller slimes tanks arranged for gravity transfer of the slimes in two series of four, without any provision for agitation except that of transference. Practically no solution of gold takes place in the filter-presses, these serving the purpose only of separating solution from solids.

Messrs. Denny advocate the abolition of sands percolation, and a total sliming of the Rand ores, crushing in cyanide solution, and of the continuous method of slime treatment by having the solution of the gold occur during the flow of the pulp, instead of obtaining it by special agitation. They outline the following plan: 1. Crushing by rolls. 2. Chilean roller mills. 3. Shaking copper plates. 4. Wheeler pans. 5. Second set of shaking copper plates. 6. Large conical settling tank. 7. Settled slimes to filter-presses, decanted solution to gold sump. 8. Discharge of slime cakes and return of barren solution (precipitated solution) to Chilean roller mills for crushing in cyanide solution. Their warrant for rejecting sand percolation is based on the fact that at the Meyer & Charlton and other mills, only 9.76 per cent. of the gold in the ore is recovered by the sand treatment, at a cost of 42c. per ton, with tailings of 3½ times the value of the slime tailings. Calculations made on the cost of sliming the sands, allowing 72c. per ton for the grinding, are much in favor of fine-grinding the sands. The abolition of sand percolation, and the sliming of all the ore is, however, an unsettled question, and as is pointed out[1] it is

[1] *Min. and Sci. Press*, Vol. XCV, p. 78.

still debatable whether as high an extraction can be obtained from reground sands, as such, in distinction to slimes.

On the question of crushing in cyanide solution in connection with plate amalgamation, metallurgical opinion is at present very much divided. There is little doubt that difficulties are experienced, such as hard plates, corrosion of plates, etc., but with the very dilute solution usually employed, 0.03 to 0.15 per cent., and the many advantages derived from the method, such as the cutting down of the amount of water employed, and the avoidance of throwing away accumulations of "waste" solution, containing cyanide and often some gold, these difficulties are not so great as to warrant the abandonment of the method. Alloy plates, such as Muntz metal, have been suggested for trial and the present year will probably see the difficulty practically solved. As to the continuous process of slime treatment by simple transference from vat to vat by gravity, it is evident that this is applicable only to those ores in which the gold is very readily soluble, such as the Rand ores. Its application must be very limited. Some form of mechanical agitation and aëration will be necessary with most ores.

Some data of general practice on the Rand during the past year are of interest. The Simmer East and Knights Deep[1] joint mill erected during 1907 is the largest unit mill on the Rand[2], containing 400 stamps, of 1670 ℔. weight, with provision to increase to 1800 ℔. The mortars are set on concrete foundations, but no anvil blocks are used. Each 10 stamps are driven by an electric motor, a departure from ordinary practice. The mill cost approximately $2,500,000. The capacity is very nearly eight tons per stamp.

The capacity of stamps in the newer mills is large, ranging from 6½ to 8 tons. With 7000 stamps, and 64 tube mills on the Rand, the average capacity per stamp is 5.79 tons, the screen mesh varying between 20- and 35-mesh and the amount of water between 7 and 9 tons per ton of ore crushed. The stamps vary between 1150 and 1600 ℔. in weight. The coarser screens are used in the plants employing tube mills for regrinding. The average yield per ton of ore crushed during December, 1907, was $8.05 per ton. The average extraction ranges between 85 and 95 per cent., of which 50 to 60 per cent. is recovered by amalgamation and the remainder by cyanidation. The decantation process of slimes treatment is still preëminent on the Rand, though filter-pressing is making decisive progress. It would seem as if the Merrill press should have a fair field.

The Desert Mill[3].—The plant of the Tonopah Mining Company at Millers, Nevada, is a new 100-stamp mill treating Tonopah ore, highly quartzose in character with small percentages of lime, magnesia, iron, and

[1] E. M. Weston., *Eng. and Min. Jour.*, Vol. LXXXV, pp. 350, 355.
[2] J. B. Pitchford, *Min. and Sci. Press*, Vol. XCIV, p. 337.
[3] A. R. Parsons, *Min. and Sci. Press*, Vol. XCV, p. 494.

manganese carbonates. The ore is essentially a silver ore, the ratio of silver to gold by weight being 90 to 1. The silver exists mostly as argentite, with some stephanite, and a small amount of cerargyrite. Small amounts of pyrite and chalcopyrite occur. Analysis also shows traces of lead, zinc, arsenic, selenium, etc. Fig. 2 shows the method of treatment.

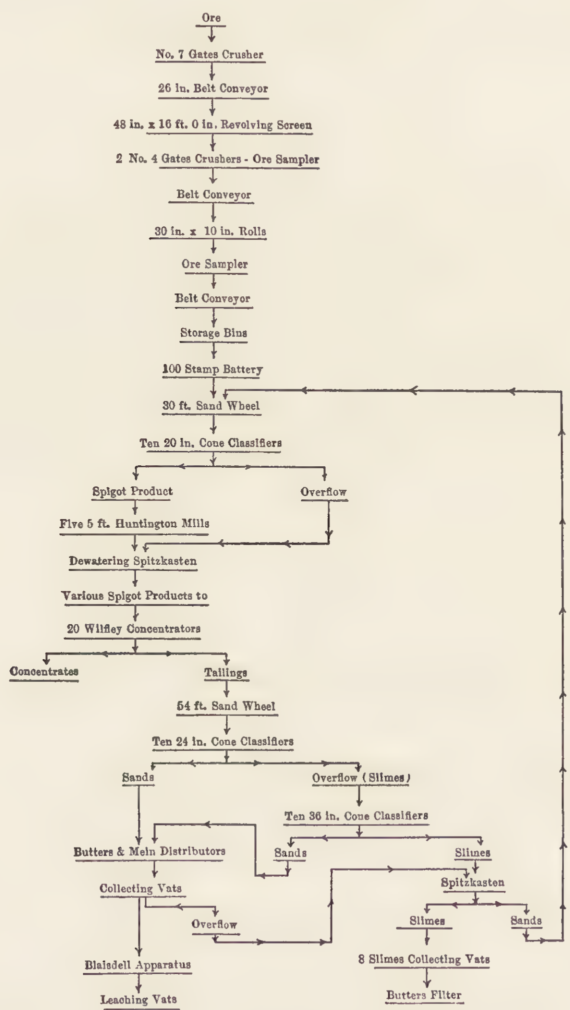


FIG. 2.—SCHEME OF TREATMENT AT THE DESERT MILL.

Each battery of 20 stamps is driven by a 50-h.p. motor alternating current, 25 cycle, 440 volts, the current being generated by steam from Babcock & Wilcox boilers, burning either coal or oil, driving three

14x28x30 in. horizontal cross-compound condensing McIntosh & Seymour engines direct connected to 250 kw., 25 cycle, 2200 volt alternators.

The falling weight of the stamp is 1067 ℔.; there are 104 six and one-half inch drops per minute; the hight of discharge is $3\frac{1}{2}$ in., crushing through 12-mesh wire screen. The capacity per stamp is 4.34 tons. The mortars are narrow and set on concrete foundations, without anvil blocks. Chrome steel shoes and cast-iron dies containing a percentage of chrome steel are used. Chrome steel dies crush 315 tons each and shoes 285 tons each. The consumption of metal per ton of ore crushed is 14.5 oz. The ore is crushed in 0.15 per cent. KCN solution using 7 tons of solution per ton of ore.

At present 90 tons of ore produce one ton of concentrates, containing 4.5 oz. gold and 815 oz. Ag. The recovery by concentration on Wilfley tables is 15.63 per cent. of the gold and 30.62 per cent. of the silver in the ore milled.

Classification is carried on by means of cone hydraulic classifiers with an upward current of cyanide solution, followed by spitzkasten; a noteworthy feature of the plant is that the overflow from the first cones is re-classified by the second set of cones. It is more usual to re-classify the spigot product or sands from the first cones. The reason is probably that it is essential to have nothing but slimes (free from fine sands as possible) go to the Butters filters, as this is particularly liable to trouble from sands getting in with the slimes. With the Blaisdell sand distributor to load the leaching vats, and double treatment, some slimes in the sands are not a serious matter.

The sands are collected in vats, drained and excavated by Blaisdell apparatus, and conveyed by a series of Robins conveyers to the leaching vats, where they are charged by a Blaisdell disk sand-distributor. Each vat holds 250 tons. While transferring the sands from the receiving vats to the leaching vats, lead acetate solution is allowed to drip on the sands (0.5 ℔. per ton) and 4 ℔. of lime per ton is also added in the receiving vats before excavation. The first leaching of the 250 tons of sands is by 30 tons of 0.25 per cent. KCN solution. This is followed by repeated pumpings of weak solution, 0.15 to 0.20 per cent. KCN. The first treatment occupies five days, including time of transfer. The sands are then transferred to another leaching vat and treated in a similar manner for five days. This is followed by a third transfer and treatment for three to five days. The sand thus undergoes three treatments totaling 12 to 15 days. It is discharged with 15 per cent. moisture and contains 0.03 oz. gold and 3.10 oz. silver per ton. The sand tailings show the following screen analysis:

Remaining on 20-mesh, 0.15; on 30-mesh, 11.64; on 40-mesh, 13.98;

on 50-mesh, 12.31; on 60-mesh, 10.48; on 80-mesh, 17.54; on 100-mesh, 12.77; through 100-mesh, 21.05 per cent.

The slimes are collected in redwood vats, 36 ft. diameter and 20 ft. deep. These vats are provided with rim overflow launders, the clean overflow being returned at once to the battery storage tank. Eight vats are provided with mechanical arm agitators, driven by a 30-h.p. motor. There are two sets of four arm agitators, quartering, and making 5 r.p.m. The lower set of arms is 2 ft. from the bottom and the upper one 8 ft. The thickened slime is drawn from the collecting vats to the agitators, which have previously been filled with 150 tons of barren solution, to which is added 1000 ℔. of slacked lime and 600 ℔. of KCN. The slime ready for agitation has a specific gravity of 1.144, or 21 parts of solid to 79 parts of solution. It is agitated for 30 hours, the agitation being assisted by aëration by compressed air, and pumping from the bottom to over the top by centrifugal pump. After agitation the slime is settled for 6 hours and the clear solution decanted to the precipitation storage vats. The settled slime is pumped to a second agitation vat, fresh solution added and the agitation continued for 24 hours, after which the contents go to the Butters filter plant. The Butters plant consists of two filter vats containing 96 frames each. The time of a treatment cycle in the filter is 3 hours 5 min. About 125 tons of dry slime are treated per 24 hours.

Precipitation is carried on in 14 seven-compartment zinc boxes. Each box holds 105 cu.ft. of shavings or 1600 ℔. of zinc. The boxes precipitate 1200 tons of solution per 24 hours. Four clean-ups are made per month. The consumption of zinc is 0.32 ℔. per ton of solution or 1.10 ℔. per ton of ore. The precipitates are not treated with acid but are thoroughly roasted, then pulverized and fluxed with 20 ℔. borax and 16 ℔. bicarbonate of soda per 100 ℔. of precipitate. The melting is done in six Faber du Faur tilting furnaces, 2800 ℔. of precipitate being melted in 24 hours. The bullion produced is 978.5 fine, of which 965 is silver and 13.5 gold. The extraction by cyanidation is 86.06 per cent. gold and 80.14 per cent. silver. Including recovery by concentration the extraction is 88.24 per cent. gold and 86.20 per cent. silver. The consumption of cyanide is 3.24 ℔. per ton; of lime, 7.20 ℔. per ton; of lead acetate, 0.38 ℔. per ton.

The Montana-Tonopah Mill[1].—This mill at Tonopah, Nev., treats ore of the same nature as the Desert Mill. The ore is crushed in a series of Gates crushers, to a $\frac{3}{4}$-in. ring and is fed to 40 stamps, crushing in cyanide solution through 12-mesh screens. The product from the stamps is classified, the coarse passing to eight Wilfley tables while the slimes and fine sands and the tailings from the tables go to two Dorr classifiers which precede tube mills, of which there are two of 5x22-ft. size. The reground product from the tube mills joins the overflow from the Dorr classifiers

[1] F. L. Bosqui, *Eng. and Min. Journ.*, Vol. LXXXIII, p. 805.

and the whole product passes to two large cone classifiers. The spigot product from these goes to 16 Frue vanners and the slimes overflow joins the tailings from the vanners and goes to the settling tanks of the cyanide plant. The thickened pulp is agitated in six Hendryx agitators from which the pulp passes to the stock tanks of the Butters filter plant. The solutions are precipitated by zinc dust with the Merrill triangular filter-press.

The Bullfrog Cyanide Mill.[1]—This mill of the Bullfrog Water and Reduction Company is at Bullfrog, Nev. The process is outlined in Fig. 3.

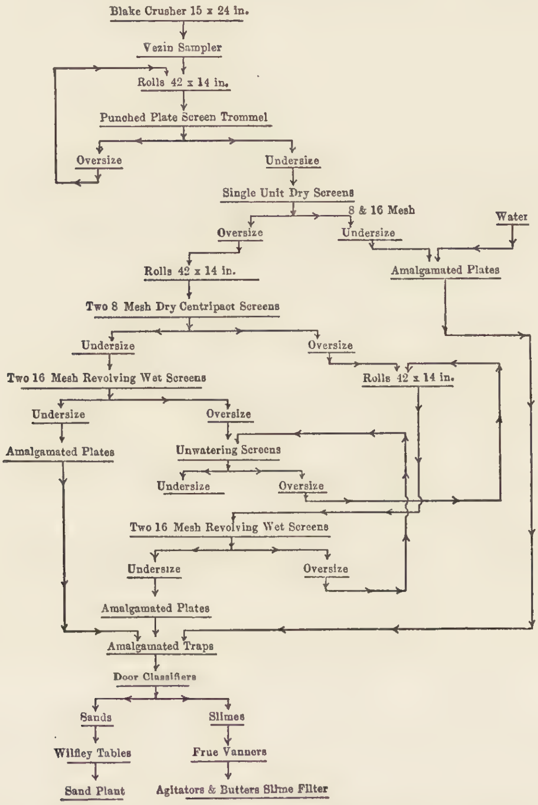


FIG. 3.—SCHEME OF TREATMENT AT THE BULLFROG CYANIDE MILL.

The sands from the Wilfley tables go by elevator to two receiving sand tanks, provided with roller gates to take care of the overflow water. The sands are excavated from the receiving tanks by Blaisdell excavators, and conveyed by belt conveyers to the Blaisdell sand distributors over the leaching vats. There are five leaching tanks 36 x 6.5 ft. The sands are leached in the usual manner. The slimes from the vanners combine with

[1] Enos R. Ayres, *Eng. and Min. Journ.*, Vol. LXXXIII, p. 376.

the overflow from the sand receiving tanks and pass through a spitzkasten to settle out sands and then to two large settling or thickening cones 20 ft. diameter and 18 ft. high. The thickened slime goes to four agitator tanks, 28 x 16 ft., in which they are agitated with solution, 0.5 lb. KCN per ton, by Blaisdell agitators. The decanted solution goes to an intermediate settling vat for clearing. The slimes are filtered in a Butters filter. Precipitation takes place by zinc thread. This mill is certainly of novel design, and marks a wide departure from the present standard, even though the recovery is largely by concentration. Its working will be watched with interest.

The Golden Cycle Mill.—This mill, at Colorado City, Colo., replacing the old Telluride plant, was built on the design of Philip Argall in 1906, was partially destroyed by fire in 1907, but resumed operations after reconstruction late in January, 1908. The plant when complete will have cost $1,250,000, and has a capacity of 750 tons per day. The ore is rough crushed, and without drying passes to ball mills provided with 0.5x 0.25-in. aperture, slotted inner screens, and 0.14-in. aperture outer screens, crushing as coarse as is consistent with good roasting. The ore then passes to eight Edwards furnaces, whence it goes to Chilean roller mills in which it is crushed in cyanide solution and amalgamated.[1] The product from the Chilean mills is passed over blankets and then is classified into sands and slimes, the sands are treated by percolation and the slimes by agitation and the Argall vacuum filter.[2] Complete data on the mill are still lacking. Its advent marks the introduction of the Kalgoorlie roasting method into the United States.

Treatment at Cripple Creek.—There are at present at Cripple Creek, 12 mills treating chiefly low-grade oxidized ores with a capacity of 1400 tons daily.[3] The practice varies considerably. The Isabella mill, with a capacity of 125 tons daily, treats ore averaging $6 per ton. The ore is rough crushed in a 9x15-in. Blake crusher, 5 lb. of lime per ton being added at this point. It then passes to 16x36-in. Davis rolls, followed by 14x27-in. rolls, and over 0.25-in. aperture bumping screens, the oversize returning to the last rolls and the undersize by conveyer to the storage bins. The six leaching tanks (5x30-ft.) are filled from belt conveyers. The leaching solutions employed are respectively 4, 3, and 2 lb. KCN per ton. Precipitation is carried on by zinc shavings in the ordinary way. The total cost of treatment per ton is 90c. The Wishbone mill crushes the oxidized ores by stamps in cyanide solution.

The Portland Mill.—This cyanide plant at Colorado City, Colo., treats chiefly the tailings from the Portland chlorination mill. The tailings are

---

[1] According to recent information amalgamation has been discontinued.
[2] *Eng. and Min. Journ.*, Vol. LXXXV, p. 219.
[3] F. L. Barker. *Mines and Minerals*, April, 1908.

nominally 14-mesh but 15 to 20 per cent. passes a 200-mesh screen. The average value is $1.40 in gold.[1]

Two 22x5-ft. tube mills each crush 200 tons with one ton of 1.2 lb. KCN solution per 1.5 tons of tailings, 5.2 tons of lime being added per ton of tailings. The pulp from the mills, 80 per cent. of which passes an 80-mesh screen, is diluted with "barren solution" containing 0.6 lb. KCN per ton, and passes to the decantation and agitation plant. The decanted solution from this passes through sand filters before going to the zinc thread precipitation. It was found that the tailings from decantation contained about 60c. of gold, a considerable portion of which was "free gold" of a whitish or gray color. (Probably imperfectly roasted telluride mineral.) To save this material, blanket launders, 70 ft. long and 2.5 ft. wide, now convey the tube mill product to the decantation plant, with the result that the tailings average 40c. in place of 60c. The decantation process will probably be replaced by a filtration process in the near future. The present cost of treatment is 50c. per ton.

The Dos Estrellas Mill.[2]—The plant of the Compañia Minera Las Dos Estrellas is in the Tlalpujahua district, in Michoacan, within three miles of El Oro, Mexico. The ore is quartz, carrying from 3 to 8 oz. of silver per ton in the form of sulphide, chloride, and metallic silver, and from 0.3 to 0.5 oz. gold per ton, chiefly free. In mill No. 2, recently constructed, and operated by electric motors, the ore is crushed by 120 stamps in cyanide solution. The stamp duty is 3.55 tons per 24 hours through 20-mesh screens. The pulp passes to classifiers, whence the sands go to five Allis-Chalmers tube mills, 24x5 ft. The tube mill product is classified into sands and slimes, the final sands, 36 per cent. of the total pulp, going to the leaching vats. Of the sands, 57 per cent. passes an 80-mesh screen. The slimes are treated by agitation and Burt slime filters. Lead acetate, 0.25 lb. per ton, is used in sand treatment as already described for the Desert mill. The cyanide consumption is 1.33 lb. per ton of ore. Ninety per cent. of the gold is extracted and 53.4 per cent. of the silver.

Guanajuato Reduction and Mines Plant.[3]—This cyanide plant at Guanajuato, Mex., is a new 80-stamp mill treating 7500 tons per month. The ore is crushed in series by two gyratory crushers to a 1½-in. ring. Rich ore is sorted out on a belt. The stamps weigh 1050 lb. and make 100 seven and one-half inch drops per minute. The mortars weigh 9000 lb. and are placed directly on concrete battery blocks. The screen is 26 mesh, No. 28 steel wire. The battery pulp is classified by spitzkasten into coarse sand, fine sand and overflow. The coarse sands go to one set Wilfley tables, the tailings from which are reground in an Abbé tube mill, this reground product being again concentrated over Johnson vanners.

---

[1] Regis Chauvenet and John M. Tippet, *Min. Rep.*, Oct. 24, 31, 1907.
[2] *Min. and Sci. Press*, Feb. 8, 1908.
[3] Carlos VanLaw, *Eng. and Min. Journ.*, Vol. LXXXIII, p. 649, and T. A. Rickard, *Min. and Sci. Press*, Vol. XCIV, p. 824.

The tailings from the second set of Wilfley tables join those from the other concentrators. The overflow from the first spitzkasten is taken to a large second one, the spigot product of which is concentrated on Johnson vanners. The overflow from this second spitzkasten joins the tailings from all tables and passes to two 20-ft. steel dewatering cones, where about one-half the water is removed and returned to the mill. The thickened pulp (4 to 1) is conveyed by an 8-in. cast-iron pipe line, one mile long, to the cyanide plant. The pulp is here classified into sands and slimes by the Homestake double system of cone classifiers.[1] The sands are fed into two receiving tanks by Butters distributors, and after draining, are discharged through bottom gates on belt conveyers which feed into a set of 40x8-ft. leaching tanks. The time of leaching is 15 days, with 0.5-per cent. KCN solution. The slimes are treated by agitation and decantation. Precipitation is by zinc thread in the usual manner.

The Guanajuato Consolidated Plant.[2]—This company of Guanajuato, Mex., has a new 80-stamp mill treating 230 tons per day. Stamps of 1050-℔. weight crush through a No. 10 diagonal slot equal to 50-mesh screen in 0.025 per cent. cyanide solution, using 7.2 tons solution per ton of ore. The capacity per stamp is 3.6 tons. The ore contains 85 per cent. insoluble matter; 6 per cent. Fe; 2.5 per cent. S; 2 per cent. CaO; 14 oz. Ag; and 0.08 oz. Au per ton. The silver occurs as argentite, stephanite and pyrargyrite. The pulp is classified by double-compartment spitzkasten, the spigot product passing to Wilfley and Standard tables, the tailings from which pass to Frue vanners. Concentration extracts 50 per cent. of the ore value. The vanner tailings to go 10 cone classifiers for the separation of sand and slime, with an extra classification of the sands. The overflow from the spitzkasten passes to settling vats. The slimes amount to 53 per cent. and the sands to 47 per cent. of the ore after concentration. The slimes are treated by agitation and decantation with eight washes of 0.1-per cent. KCN solution and two washes of water, but Ridgway filters are being experimented with. The sands are leached by double treatment, the first treatment of 78 hours with a 0.3-per cent. KCN solution extracts 35 to 40 per cent. of the gold and silver in the sands. The second treatment after transference takes 11 days, 6.5 days with 0.3-per cent. KCN solution and 4 days with 0.15-per cent. KCN solution, followed by about 18 hours of wash water. Lead acetate is used in both slime and sand treatment to precipitate soluble sulphide. The solutions are precipitated by zinc thread, the consumption of zinc amounting to 1.3 ℔. per ton of ore treated or 0.08 ℔. per ton of solution precipitated. The amount of solution precipitated per day is 4030 tons, the amount of zinc provided is 2.2 ℔. for every ton. The author of the

---

[1] *The Mineral Industry*, Vol. XV, p. 419.
[2] Bernard McDonald, *Eng. and Min. Journ.*, April 4, 1908.

paper, Bernard McDonald, states that if the plant were to be rebuilt decided modifications would be installed. The decantation process is obsolete and vacuum filters are rapidly replacing it in Mexico. The mill is peculiar in not having regrinding apparatus, but using very fine battery screens.

The Butters Copala Mines[1].—This plant at Sinaloa, Mexico, cyanides silver ores containing 15.8 oz. silver and $1.96 gold per ton. The ore is crushed by twenty 850-lb. stamps, through 12-mesh No. 15 wire screens, in 0.07-per cent. NaCN solution, with an alkalinity of 0.135 per cent. NaOH, and using 16 tons of solution per ton of ore. The stamp duty is 5.3 tons. The battery product passes to cone classifiers, the spigot product of which goes to two tube mills. The product from the tube mill joins the overflow from the first classifiers and goes to a second classifier, the spigot product of which is returned to the tube mills, and the overflow of which goes to the third classifiers, which make the final product of sands and slimes. The sands, constituting 42 per cent. of the total, contain 98c. in gold and 15.33 oz. in silver per ton and are leached with a 0.3-per cent. NaCN solution. The slimes amounting to 58 per cent. of the ore contain 70c. in gold and 15.2 oz. of silver. They go to agitation vats and thence to Butters filters. The sand residues contain gold, 0.05 oz.; silver, 1.6 oz. The slime residues contain gold, 0.038 oz.; silver, 0.34 oz. Lead acetate is used in the treatment of sands. The zinc consumption is 1.1 lb. per ton of ore; of lime, 6.5 lb., and of cyanide 1.5 lb. The precipitates are dried and fluxed, and not acid treated.

The Blaisdell Coscotitlan Syndicate[2].—This concern has constructed at Pachuca, Mexico, a plant to re-treat the great tailing dumps from the patio process. The plant has a capacity of 500 tons per day. It is expected that the tailings will be passed over Callow screens to take out vegetable matter, and the mixed pulp then classified by spitzkasten, the coarse being reserved for future treatment, and the fines reground in tube mills. The reground pulp will be treated by agitation and Blaisdell filters; zinc thread precipitation will be used. The Usher-Adair process is being experimented with. (See under slimes treatment.)

The Real del Monte[3].—This company has erected two cyanide plants at Pachuca. One, the Loreta mill, crushes with 40 stamps, classifies the product and concentrates over Wilfley tables and Frue vanners. The tailings from these are reclassified, the sands being reground in Chilean mills, and the reground product classified. Slimes are treated by agitation and decantation. The Butters filter and tube mills (to replace the Chilean mills) are being added.

Metallurgical Practice at Kalgoorlie, Australia[4].—The methods in vogue

---

[1] L. N. Bullock, *Min. and Sci. Press*, Vol. XCIV, pp. 221, 335, 719.
[2] M. R. Lamb, *Eng. and Min. Journ.* April 4, 1908. *Min. and Sci. Press*, Feb. 8, 1908.
[3] M. R. Lamb, *loc. cit.*
[4] "Metallurgy of the Kalgoorlie Gold Field." G. W. Williams, *Eng. and Min. Journ.*, Feb. 15. 1908.

on these telluride ores ranging from $6.50@17 per ton in value, fall into two divisions. First, wet crushing, classifying and concentrating plants, with or without the use of bromo-cyanogen. Second, roasting, dry crushing plants, with auxiliary fine grinding to slimes and filter-pressing. One of the more recent mills employing the first method is the New Golden Horseshoe. Here one hundred 1270-lb. stamps, making 104 eight inch drops per minute, crush through 15-mesh screens, in 0.07 per cent. KCN solution. The stamp duty is 5.5 tons per 24 hours. The mill pulp is elevated to 10 classifiers, the coarse sands, 27 per cent. of the total product, pass over 12 Wilfley tables and the fine sands over 11 Wilfley tables. From the 23 tables three products are obtained. First, concentrates, which go to roasters; second, middlings, which pass over four more tables and then to the tailings wheel; and third, fine sands and slimes which pass to the sand traps (classifiers). The spigot product from these is reground in 10 tube mills, 16.3x4 ft. size, and then passes back to the classifiers. The slimes from all sources pass to the agitation vats, where the cyanide strength is made up to 0.06 per cent., and agitation is carried on for 18 hours. The consumption of lime is 1.5 lb., and of cyanide 1.5 lb. per ton of ore treated. The concentrates are roasted in Edwards furnaces, crushed fine in grinding pans, agitated. and filter-pressed. At the older mill of the Golden Horseshoe, sand percolation is still practiced. The Ivanhoe Gold Corporation and the Oroya Brownhill use bromo-cyanogen in the treatment of slimes.

The practice at the roasting plants, which include the Great Boulder, Perseverance, Kalgurlie, South Kalgurlie, Associated, and Associated Northern is very uniform. The ore after rough crushing is crushed dry in Krupp ball mills or Griffin mills, chiefly by No. 5 Krupp ball mills. Eight Krupp mills at the Kalgurlie maintain a duty of 45 tons each per 24 hours, crushing through a 37-mesh screen, with a consumption of 25 h.p. each. The product from these mills shows the following screen analysis: On 40-mesh, 1.3; on 60-mesh, 14.9; on 80-mesh, 15.3; on 100-mesh, 9.5; on 120-mesh, 2.9; on 150-mesh, 4.3; on 200-mesh, 6.2; through 200-mesh, 45.6 per cent. From the mills the ore passes to Merton or Edwards roasting furnaces, and after roasting is thoroughly cooled. It is then mixed with 0.06 per cent. cyanide solution and at the South Kalgurlie passes to Wheeler grinding pans, of respectively 8 ft. and 5 ft. diameter. The system of fine grinding is generally as follows: The pulp from the mixer is delivered to one pan, the overflow from which is divided between two following pans, each of which in turn overflows to another pan. The final product then passes to two spitzkasten, the spigot product of which is returned to two more pans. The first five pans are of 8 ft. diameter and the last two of 5 ft. diameter. Amalgamation takes place in the pans, about 25 per cent. of the gold in the ore being recovered in this manner. At

most of the other mines enumerated, the ore from the furnaces is first classified before passing to the pans for the regrinding of the coarse material. The aim is to produce a final product, 90 per cent. of which will pass a 120-mesh screen. The final product at the Kalgurlie after regrinding shows the following screen analysis: On 100-mesh, 2.1; on 120-mesh, 2.1; on 150-mesh, 6; on 200-mesh, 13.4; through 200-mesh, 76.4 per cent. After fine grinding the pulp passes to the settlers and thence to the agitating tanks, where the solution strength is brought up to 0.10 per cent. KCN. The consistency of the pulp for agitation is two of the slimes to three of solution. Both air and mechanical agitation are employed, chiefly the latter. After agitation the pulp goes to Dehne presses, hydraulically closed, both Montejus and plunger pumps being used for filling. The cakes are discharged with 15 to 20 per cent. moisture. The consumption of cyanide at the roasting plants averages about 0.6 ℔. KCN per ton. Lead acetate is added in the mixers, 1 ℔. per 5 tons of ore, to precipitate the soluble sulphides. Ordinary zinc shaving precipitation is employed.

Slime Treatment.

This question received great attention during 1907, in connection with fine grinding. The so-called "sliming" of an ore ill defines what takes place, and the term "fine grinding" is much to be preferred as more truly expressing the nature of the operation. A true slime exists in an ore, independent of the method of grinding and cannot be produced by it, but fine sands are the product of fine grinding, although slime may be present inherently. Metallurgists generally are becoming somewhat more cautious in their enthusiasm for grinding everything fine, so as to be able to treat it by "slime methods," and the percolation of fine sands will hold its own.

Vacuum filtration for fine sands in place of percolation is to be considered with caution, in fact the cases where it can be applied are rare. Ordinarily fine sands are a source of serious trouble to the vacuum filters, especially when present with true slime. When very fine sands only are to be treated, with no slimes present, vacuum filtration is a possibility. In this case, however, the filter press has a field all of its own.

During 1907 considerable litigation and much discussion went on in reference to the submerged suction or vacuum filters, and submerged pressure filters. Filters of these types increased greatly in number during the year, especially in the United States and Mexico. While the Moore Filter Company won several actions against the owners of other patents, litigation on this important matter is not over and just what are the master patents is not yet decided. Aside from the Moore and Butters-Cassel filters, quite a number of other filters of the same general type, such as

the Ridgway, Kelly, Burt, and Blaisdell, are being used. The more important are described.

The Ridgway Filter.—The machine consists of 12 flat, cast-iron filtering frames which are in the form of sectors of a circle. These frames are corrugated on their under surfaces to which screens are attached and over which ordinary filter cloth is fixed. The frames are suspended horizontally from radially arranged hollow arms, projecting from a central hollow revolving column provided with internal compartments. Each frame is connected by three radial pipes and rubber hose with these compartments. The outer edges of the frames form the periphery of a 12-ft. circle, each frame being supported on a 4-in. roller running on a circular track at the periphery of the frames. The under or filter side of the frames dips into an annular trough, divided into three parts, with spaces between. The circular track is elevated opposite these spaces, so that the frames in passing are lifted, so as to clear the formed slimes cake, when going from one part of the trough to another. A nest of three valves in the pipes leading from the filters to the column are operated automatically by rollers as the frames pass over the elevated sections of the trough. One pipe connects with the solution compartment of the central column, one with the wash-water compartment, both connected with a vacuum pump, and the third with the compressed air compartment. The mode of operation is as follows: Two parts of the annular trough are filled respectively with slimes pulp to be filtered, and solution wash, while the third serves as the discharge chamber. The frames revolve, with suction on, pass through the slime pulp and take on the $\frac{3}{8}$-in. filter cake, rise over the elevated portion and pass to the solution part of the trough, the valve being automatically changed; here they suck solution through the slime cakes, pass over the second elevated portion, the valve changing automatically to compressed air, and discharge the slime cakes into the third part of the trough, then again making the cycle as described. The total cycle requires 60 sec., of which 13 are in pulp, 30 in wash solution, the balance, 17 sec., on the raised portions of the track and in discharging. The machine itself requires 0.5 h.p. The total power, including vacuum, etc., is 5 h.p. The pulp and solution in the compartments is regulated as to level by float valves. Each frame has 4 sq. ft. of filtering area. This machine has been in operation continuously at the Great Boulder mine at Kalgoorlie since January, 1906, treating 50 tons of dry slime per 24 hours, of the clean quartz kind, with a 20-in. vacuum; a new 500-ton plant is now in operation. The consistency of the pulp is 44 per cent. solids, 92 per cent. of it passing a 200-mesh screen. The discharged slimes contain 33 per cent. moisture. The operation of the machine has yielded satisfactory results, with low costs. The filter is also being installed in Guanajuato, Mexico.

The Blaisdell Filter.—This is a pressure filter and consists of three

parts, a series of filter frames, a vertical pressure cylinder, and pumps for vacuum, slimes, and solution. The filter leaves are suspended in the cylinder, and consist of a series of non-porous columns, vertically grooved with narrow drainage channels, and covered with canvas. The outlet from each leaf is at its lowest point, all uniting into one common pipe and passing out of the cylinder. The cylinder is charged by gravity from slimes storage tanks, pressure (25 to 50 ℔. per sq. in.) within the cylinder being maintained by a small pump. The gold-bearing solution is forced into the interior of the filter frames, and the slime cake forms on the outside. When a 2-in. cake has formed, the supply of slimes is cut off and the surplus returned through a bottom valve to the storage tank. The cylinder is then filled with solution or wash water as desired, which in turn is forced into the filter leaves. While transfers of liquids are made the slime cake is held in place by vacuum. To remove the slime cake from the filter leaves, clear water is admitted at low pressure to the interior of the filter leaves, when the slime cake drops off, and is discharged at the bottom. An agitator in the bottom of the cylinder keeps the pulp in suspension.

The Burt Rapid Cyanide Filter[1].—This is a pressure filter, similar in principle to the Blaisdell, just described. It is in successful operation at mill No. 2 of the El Oro Mining and Railway Company. It consists of a pressure cylinder inclined at 45 deg., with a large discharge door operated by toggles at the lower end. Within it are loosely suspended a series of 28 filter mats, each of 8 sq. ft. area. They consist of a core of cocoa matting, held by a rectangle of perforated pipe, and entirely enclosed in canvas. The ends of the pipe nearly meet in a special T, held rigid to the interior of the cylinder, and are all connected to the main solution pipe outside the cylinder. On the bottom side of the cylinder near the door is placed the one connection for admitting slimes, solution or wash water, and for displacing surplus slimes, or solution, etc., all properly manipulated by a series of valves. In operating the apparatus, the slimes are forced into the cylinder under 40- to 60-℔. pressure, until a slime cake of 1.5-in. thickness has been formed. The slime is then shut off, the compressed air valve turned on and the slimes discharge valve opened. The low air pressure forces out all the surplus slime to the storage vats and keeps the slime cakes on the filters at the same time. The discharge valve is then closed and wash water admitted, which, after a sufficient time, is again displaced by low air pressure (a low air pressure of less than 10 ℔. will not cause cracking in the slime cake), thus displacing the wash water and drying out the cakes somewhat. The discharge door at the end of the cylinder is then opened and air is admitted to the interior of the filter leaves, causing the slime cakes to drop off. These slide out of the

[1] E. Burt, *Min. and Sci. Press*, Vol. XCV, p. 717.

cylinder by gravity. The capacity of the press is 120 tons per 24 hours, one charge of 1.75 tons of slime taking 27 to 30 min. The amount of surplus slime to be repumped at each operation is 39 per cent. of the charge.

Other Filters.—A probably important patent in slime filters is that of Askin M. Nicholas, No. 619,211, granted in United States Feb. 7, 1899; this may have a bearing on many of the later filters.[1] The Kelly filter[2] is similar to the Burt pressure filter, except that the filter mats are on a frame which can be slid in and out of the pressure cylinder, to discharge the cakes. Other patents of interest in connection with slime filters are: the Wade continuous vacuum filter, U.S. pat. No. 854,972;[3] the L. C. Trent continuous filtering machine;[4] the Argall gravity suction filter, U. S. pat. No. 803,827, June 5, 1906; the Fairchild continuous vacuum filter[5]; and the Hendryx filter.[6]

Comparison of Moore and Butters Filters.—There was much discussion during 1907 concerning the relative merits of the Moore and the Butters filter. Each filter possesses such definite merits that neither can be decided to have the advantage at present. The Moore company has won important litigation, but the Butters company seems to be installing the greater number of plants, particularly in Mexico. However, its field is becoming invaded by the pressure filters, and it would not be surprising to see the continuous type of vacuum filter, like the Ridgeway, come to the front in this country and Mexico. The situation in reference to the Moore and the Butters filter may be summed up as follows:[7] 1. The operating costs of both systems are about equal, as are also the maintenance and repairs, the latter possibly somewhat higher in the Moore system. 2. The installation cost for the Moore system is 35 per cent. higher than for the Butters system of equal filtering area. 3. The capacity of the Moore system is 50 per cent. greater than that of the Butters system without gravity discharge, for an equal filtering area, owing to the shorter cycle of the Moore system. 4. The Moore system has higher efficiency for the recovery of gold-bearing solution, and less loss, as the loss due to the mixing of residues in the pipe lines and vats, occurring in the Butters system, is absent in the Moore.

The Butters System.—Recent practice of the Butters system is described by E. M. Hamilton.[8] As at present constructed the filter leaf is made on a frame, the upper side of which is formed by a bar of wood 1.75 in. thick and 5.5 in. wide, and 10 ft. long placed on edge, sufficiently long

---

[1] *Min. and Sci. Press*, Vol. XCV, p. 715.
[2] "Recent Improvements in The Art of Slime Treatment." D. J. Kelly. *West. Chem. and Met.*, Sept., 1907.
[3] *West. Chem. and Met.*, Sept., 1907.
[4] *Min. and Sci. Press*, Feb. 22, 1908.
[5] *Min. and Sci. Press*, Vol. XCV, p. 279.
[6] *West. Chem. and Met.*, Vol. III, p. 188.
[7] A. G. Kirby, *Min. and Sci. Press*, Vol. XCV, p. 48.
[8] "Filtration of Slimes by the Butters Method." *Min. and Sci. Press*, Vol. XCIV, pp. 785, 818.

for its ends to rest on the containing box. The remaining three sides of the frame are made of 0.5-in. pipe, one end flattened, turned at a right angle and bolted to the under side of the wooden bar; the other extremity projects through the end of the same bar and is connected with the vacuum apparatus. The lower member of the pipe frame is perforated on its upper side with $\frac{5}{16}$-in. holes at intervals of 4 in. The space in the interior of the frame is exactly filled with cocoa matting. On each side of the frame is then placed a sheet of 16-oz. canvas, large enough to overlap the frame. This is then sewed by machine in vertical parallel rows 1 in. apart, and the overlapping ends are sewed around the pipe. The upper edges are fastened into the wooden bar by being sunk into grooves, a lath being driven in over it, wedging it firmly in position. This lath projects outward for 0.5 in. from the bar and being grooved on the upper side serves the purpose of deflecting drippings of solution from the bar when the pulp is withdrawn, preventing them from trickling over the surface of the newly formed cake and cutting channels in the slime, which channels were found to be starting points for cracks.

Each frame has six vertical strips of wood on each side to stiffen it and to prevent the adjoining frames from sticking together. Frames are also made in which the pipe is made continuous around the four sides, the upper horizontal member being perforated for the admission of water into the interior, while the lower one admits air. This type is used where water is scarce, and the cakes are not submerged in wash water to discharge them. Compressed air can be used for a dry discharge but is not very satisfactory. The filter leaves are placed in the containing box 4 in. apart, center to center. Each filter frame is connected by union and rubber hose to the 4-in. pipe running along the side of the containing box and connecting with the vacuum pump. At each end of the 4-in. pipe is a valve, one connecting with the vacuum, and the other with water at low pressure. All transfers of pulp and solution are performed by a single centrifugal pump with valves and levers so arranged that one man can perform all operations without leaving the spot. The thickness of cake varies from 0.75 to 1 in., and the best specific gravity of the pulp for good working is from 1.3 to 1.4.

To get the highest efficiency from the filter it is essential to make the transfer of pulp and solution as rapid as possible. For this rapid transfer the gravity system of filling and emptying gives good results. Wherever topographic conditions allow it, the filter box can be rapidly filled from the pulp storage tank by two or more 6-in. pipe lines, and after the cakes are formed, the surplus pulp is discharged by large gates into a sump vat below, which has a light stirring gear to keep the pulp in suspension while it is being transferred back to the storage vat by the centrifugal pump. In this way the filter vat may be filled in 4 or 5 min. and emptied in

3 min., so that the cake is exposed to the air only 8 min., thus materially reducing the time of a cycle and avoiding the cracking of the slime cakes.

The Agitation of Slimes.—It is essential to recognize the fact that filtering devices, in order to attain the highest efficiency, must act solely as filters and not as extractors of gold and silver, notwithstanding the opinion of some metallurgists to the contrary. In cases of extremely easy solubility of the gold, agitation tanks for effecting the solution of the gold may be unnecessary, but in most cases, particularly in the case of silver ores, extensive agitation is essential. An interesting development in this direction is the Brown agitation vat[1], known also as the "Pachuca" and "Groth" vat. It is a steel cylindrical vat with a conical bottom 30 ft. high and 15 to 20 ft. in diameter. At its central axis it has a 15-in. pipe, reaching nearly to the bottom, and extending almost to the top of the tank. Into the bottom of this 15-in. pipe leads a 1.5-in. compressed air pipe. The tank is filled with slimes pulp, and compressed air turned on into the interior 15-in. pipe, causing an air lift action through this pipe which serves very effectively for agitation and aëration. Fine sands can be handled efficiently by this method.

The Usher process[2], so-called, is agitation by air. The air is forced through perforated pipes in the bottom of the slime vats for aëration and agitation. It seems, however, that this method has long been practiced in the United States and presents nothing particularly new. The Adair process[3] consists in adding certain amounts (0.25 per cent.) of wad and umber or natural oxides of manganese and iron (found in the Transvaal near Johannesburg) to the slimes pulp. The addition seems to aid slimes settlement and is said greatly to increase the solution of the gold. The two processes are commonly used together, in decantation and agitation of slimes, and by their employment it is possible to obtain clear solution continuously from decantation vats while filling and even agitating.

The Hendryx Agitation Tank[4].—This consists of a cylindrical tank with a conical bottom. In the center of the tank is a circular well which extends nearly to the top and bottom and is supported by braces from the side of the tank. It has a circular apron at the top which slopes gently toward the circumference, receiving the pulp discharge through the central column or well and permitting it to spread out in a thin sheet and absorb air. In the well is a hollow shaft, carrying a number of screw propellers and driven from the top by a driving pulley. A coil of steam pipe in the vat serves to raise the temperature of the charge if that be desirable, the loss of oxygen from heating being compensated for by

[1] Francisco Narvaez, *Min. and Sci. Press*, Vol. XCV, p. 689.
[2] E. M. Weston, *Eng. and Min. Journ.*, Vol. LXXXIV, p. 65.
[3] *Ibid.*
[4] L. D. Bishop, *West. Chem. and Met.*, Vol. III, p. 187.

the cooling and reabsorbing of oxygen while the pulp flows over the apron. The pulp is continuously agitated by being lifted through the central well by the action of the screw propellers.

The Merrill Filter Press.—The great slimes plant of the Homestake company at Deadwood has been a marked success since its installation, and the Merrill press[1] will undoubtedly play an important part in slimes treatment, wherever filter-pressing is applicable. The plant has been treating 40,000 to 50,000 tons per month. During November, 1907, the recovery was 91 per cent. by bullion returns, the average value of the slime heads being 90c. The cyanide consumption is 0.33 ℔. per ton; of zinc dust, 0.2 ℔. per ton, and of lime 4.5 ℔. per ton, at a total cost of treatment of 32c. per ton.

At the plant there are two receiving tanks, one for slimes from the Lead mills, and one for those from the Gayville mills. These tanks are deep ones, provided with a central pipe reaching nearly to the bottom of the tank. Into this the slime enters at the top, while at the same time lime sludge is conveyed to nearly its bottom by a pipe line. These vats have the appearance of a "Brown" vat, except for the air agitation device. The method of charging keeps the contents in continuous agitation and serves efficiently to mix the lime with the pulp. The discharge from the tanks is at the bottom, the flows uniting and passing to the charging tank for the presses. This tank has an equal-pressure device, a constant level being maintained by a 12-in. valve operated by a float. The variation of level, when charging presses, is never more than 4 in. and usually is less. The charging pressure is 30 ℔., obtained solely by gravity. The specific gravity of the pulp is determined by the adjustment of the spigot discharge of the cones in the clarifying houses at Lead and Gayville, and no thickening of the pulp is carried out at the slimes plant.

The filtering cloths in the presses are first No. 11 or 12 twilled cloth on the metal plate, followed by No. 8 twilled cloth, the first cloth serving to protect the second or real filtering cloth from any unevenness of the corrugations on the filter plate. In this way cloths last over nine months. As the presses need not be opened except to change cloth, this arrangement has obvious advantages. Once a month the cloths are freed from lime by filling the press with a 2-per cent. hydrochloric acid solution, succeeded by a water wash. This takes from six to eight hours.

Crushing and Grinding.

Recent Tendencies.—Stamps are preëminent for crushing after rough crushers; the general tendency is to increase the weight, to use coarse screens, 12- to 20-mesh, with a regrinding of the coarse product. The weight of stamps on the Rand and in Rhodesia has increased up to nearly

[1] *The Mineral Industry*, Vols. XIV and XV.

1800 lb. as well as in Australia, with an increase of capacity. The capacity of the recent stamps on the Rand is between 7 and 8½ tons each, and it is stated that in Rhodesia it is as high as 9 and 10 tons. Some of the Kalgoorlie mills crush as much as 7 tons per stamp. In the United States and Mexico but very little change has been made from old practice, and gravity stamps of more than 1250 lb. are rare, and capacities above 4.5 or 5 tons are not known, except in isolated cases with very soft ores. Chilean roller mills find application in place of stamps in some instances, notably in the Black Hills, and will probably have an increased field. For dry crushing, ball mills are largely used. For fine grinding and regrinding, tube mills are most largely used although the question of grinding pans of the Wheeler type *versus* tube mills is being freely discussed. However, the use of pans at present is practically confined to regrinding the soft friable roasted ore in Kalgoorlie although a number of plants there are using pans for regrinding raw sands.[1] The method of grinding in pans in Australian milling practice is briefly described in the reference. The Homestake Mining Company, of South Dakota, is now experimenting on regrinding coarse product with pans and tube mill, side by side, and the results will be studied with interest. Pans are also used in some of the Rhodesian mills.

Tube-mill Practice.—An important invention in tube-mill practice is that of the El Oro or Brown tube-mill lining. It consists of bar plates of cast iron, in segments with a circumference to fit the interior of the tube mill, to which it is firmly bolted. Each bar has two longitudinal channels, 1½ in. deep, 3¼ in. wide at the top and 3⅛ in. at the bottom, in which the grinding pebbles wedge themselves firmly, thus forming what is practically a silex lining of the pebbles themselves. The lining by far outlasts any other form, lasting over eight months against three to six months of the silex lining, or practically until the segments wear down to the one inch web. The lining requires no attention; as the pebbles wedged in the riffles become worn out they are automatically replaced by new ones. Silex linings are used almost entirely on the Rand, lasting from two to three months. In Australia hard iron liners are mostly in use, lasting from four to six months. The difference is due to the nature of the material ground. Most tube mills on the Rand are of 5½ ft. diameter and 20 to 22 ft. length. In Australian practice the mills are rarely over 13 to 16 ft. long and 4½ to 5 ft. in diameter. United States and Mexico follow the Rand in this respect, although short mills are also employed. The Abbé spiral-feed mill has received considerable application in this country and in Mexico.

C. W. Van Law describes tube mill practice at Guanajuato.[2] There

[1] H. T. Brett. *Min. and Sci. Press*, Vol. XCIII, p. 745.
[2] *Min. and Sci. Press*, Vol. XCV, p. 205.

are two Abbé mills, 4.5x20 ft. crushing 80 tons per day. The pulp going to and coming from the mill gives the screen analysis shown in the accompanying table.

EFFICIENCY OF TUBE MILLS AT GUANAJUATO.

Size.	To the Mill. Per Cent.	From the Mill. Per Cent.
On 40-mesh	11.2	0.5
On 50-mesh	11.2	1.7
On 60-mesh	8.9	2.9
On 80-mesh	16.6	6.0
On 100-mesh	16.3	16.2
On 120-mesh	26.1	21.8
Through 120-mesh	9.7	51.2

The mills are of the tire type, the tire running on friction rollers, which rollers can be adjusted by canting so as to cause the mill to take any position desired, longitudinally. In mills without this adjustment a travel of the tube, in one or another direction which ordinarily could not be prevented, will cause the rims of the friction rollers to be unduly worn in a short time, requiring their renewal frequently, or, if guide rollers on vertical axis bearing against the side of the front tire are used, these are subjected to this wear. Silex linings are used, lasting eight months without renewal. It is essential to have the tires on which the mill runs absolutely true, and of homogeneous material, so as to have them wear evenly, otherwise bumping results, causing poor work and ultimate destruction of the mill. The tubes are kept filled slightly above the center with pebbles, the pebble wear being 0.75 lb. per ton of ore treated. It requires 60 h.p. to start the mill and 43 h.p. to keep it going.

Costs.

Plant No. 2 of the Homestake Company.—Plant treats 25,000 tons per month. Figures cover a period of six months. Classification, 1.6c.; cyanide treatment, 9c.; precipitation, zinc dust, 1.9c.; power, 3.5c.; assaying and refining, 1c.; total per ton of sands, 17.1 cents.

Rand.—Average cost of milling and cyaniding on the basis of a 110-stamp mill, $1.33 per ton.

Kalgoorlie, Australia.—Cost of reduction at two wet mills: Ivanhoe Gold Corporation, $2.20 per ton; Oroya Brownhill, $3.30 per ton. At the following dry crushing and roasting mills: Associated Northern, $2.85; Associated, $2.76; Great Boulder Perseverance, $2.66; South Kalgoorlie, $2.65; Kalgurlie, $2.65; Great Boulder Proprietary, $2.48. At the Associated Northern, treating 3700 tons per month. Cost of slime treatment and filter-pressing: cyanide treatment, 34c.; filter pressing, 41c.; precipitation, 12c.; disposal of tailings, 4c.; total, 91c. per ton.

Mexico.—At Los Dos Estrellas mill, El Oro; Cyanide treatment, 85c.; assaying and refining, 3.5c.; milling, 72c.; total, \$1.805 per ton. The El Oro Mining and Railway Company, El Oro, 1907: Cyanide treatment, \$1.10; milling, 43c.; total, \$1.53 per ton. At the Guanajuato Consolidated Mining and Milling Company's plant at Guanajuato: Milling, \$1.10; chemicals, \$2.06; labor, 41c.; general expense, 38c.; assaying and refining, 12c.; power, 26c.; crushing and sorting ore, 16.48c.; total, \$4.496 per ton. At the Butters Copala Mines Company, the cost of Butters slime filtration was: power, 1.3c.; maintenance, 6.8c.; labor, 2.9c.; total, 11.05c. per ton of slime. Cost of precipitation by zinc thread and refining without acid treatment, 1.87c. per oz. of bullion.

United States.—At the Combination mine, Nevada, the cost of regrinding sands in tube mills was 70c. per ton of sands. At the Isabella mill, Cripple Creek, dry crushing: Cyanide treatment, 38c.; tramming ore to mill, 22c.; milling, 14c.; disposal of tailings, 9c.; maintenance, 2c.; assaying and refining, 2c.; total 87c. per ton. The great battle between the cyanide and the chlorination process is in progress on Cripple Creek ores, and a reduction of rates has been made. The accompanying table gives rates of treatment by cyanidation and by chlorination.

RATES PUT INTO EFFECT MARCH 1, 1908.

Class of Ore.	Frt. Rate to Mills.	Treatment Charge. Cyanide. Golden Cycle. 3 to 6 yr. contract.	Treatment Charge. U. S. R. & R. Co. Chlorination. No contracts.
Up to \$8 incl.....	\$0.75	\$3.25	\$2.75
\$ 8 @ 10........	1.00	3.50	3.50
10 @ 15........	1.00	4.25	4.00
15 @ 20........	1.00	5.00	4.50
20 @ 25........	1.25	5.25	4.75
25 @ 30........	1.25	5.75	4.75
30 @ 40........	2.00	5.50	5.00
40 @ 60........	3.50	5.50	4.50
60 @100........	3.50	6.00	5.00

The mills make the charge on the basis of freight plus treatment so that these two items added show a progressively increasing cost with the grade of ore.

Precipitation and Refining of Precipitate.

San Sebastian Mine.—Electrolytic precipitation of cyanide solution[1] is successfully carried on at the San Sebastian mine, of the Butters Salvadors Mines Company, at Santa Rosa, Republic of Salvador. The ore is a very complex silicious one, containing telluride of gold and free gold and but very little silver; pyrite, and marcasite; telluride of copper (rickardite), and a mineral similar to enargite but containing antimony. The quantity of copper extracted by cyanide ranges between 1.5 and 2.5

[1] Chas. P. Richmond., *Eng. and Min. Journ.*, Vol. LXXXIII, p. 512. See also Vol. XIII and XV of *The Mineral Industry.*

lb. per ton of ore treated. The ore is crushed in Blake breakers; dried in a rotary dryer; crushed in No. 5 Krupp ball mills to 40 mesh; roasted in Jackling furnaces, reducing the sulphur from 3.5 to 0.25 per cent.; and after roasting, elevated with weak cyanide solution to collecting tanks. The sands and slimes are then classified, the slimes treated by agitation and Butters filters and the sands by percolation. Combined electrical and zinc precipitation is practiced to recover the gold and copper.

The electrical precipitation box is 30 ft. long, 10 ft. wide and 4 ft. 8 in. deep, with a bottom slope of 1 in. per foot. It is divided by weir partitions into 12 compartments, 10 of which are used for precipitation and the first two for settling slimes out of solution before entering the precipitation compartments. Each compartment contains 25 anodes and 24 cathodes. The anodes are rolled lead plates $\frac{1}{8}$ in. thick, 4 ft. long, and 22 in. wide. The lead plates are peroxidized in a 1 per cent. solution of permanganate of potash with or without $2\frac{1}{2}$ per cent. sulphuric acid, and a current of 2.5 amperes per sq.ft. for 6 hours. The life of an anode is from eight to 12 months. They finally disintegrate completely, forming PbO_2, which is resmelted to lead and again rolled into plates. The cathodes are lead plates $\frac{1}{16}$ in. thick and of the same size as the anodes but not peroxidized. Electrical connection to the plates is made by cast lead lugs to which is soldered the copper conducting wire. The plates are then inserted in a wooden head and secured with wooden pegs. The distance between plates is 3 in. The 10 compartments are connected in series and the current strength is one ampere per square foot of anode surface with a drop of potential of 4 to 4.5 volts for each compartment. The gold and copper precipitate as a dense hard coating. There is a gradual accumulation of low-grade precipitate on the anode and in the bottom of the box, having a gold content of from 5 to 50 oz. per ton. The solution has an average value of $16 per ton and flows through the boxes at the rate of 150 tons per 24 hours. After passing the electrical precipitation boxes the solution goes to two zinc boxes of the usual type. The electrical precipitation recovers from 80 to 90 per cent. of the gold, nearly all of the copper that is precipitated and regenerates a high percentage of cyanide. A

PRECIPITATION AT SAN SEBASTIAN MINE.

Month. 1906.	Solution to Electrical box.		Solution leaving Zinc boxes.		Lb. of Cu. precipit.	Lb. of KCN regenerated	Per Cent. of Regeneration of KCN	Fine oz. of gold precipit.	Ratio of Au. to Cu. Au=1.
	Per Cent. Cu.	Per Cent. KCN	Per Cent. Cu.	Per Cent. KCN					
Feb.	0.079	0.163	0.061	0.198	1,965	3,843	21.4	3942.1	1 to 7.27
March.	0.088	0.132	0.057	0.166	2,915	3,226	25.7	3923.3	1 to 10.83
April.	0.075	0.135	0.047	0.156	1,888	1,483	15.5	3356.4	1 to 8.20

curious feature is that the tailings solution from the zinc boxes still

carries a considerable percentage of copper which escapes both the electrical and zinc precipitation.

The deposit of gold and copper on the cathodes of the electrical precipitation is separated as follows: The lead cathodes remain in the boxes for 20 to 30 days increasing in weight from 8 to 12 lb. They are then removed, drained and placed in an acid box, with 2 to 3 per cent. sulphuric acid, as an electrolyte, where their function is reversed to that of an anode. They are hung in a wooden frame, with closed bottom and open sides, over which is placed a cotton cloth sack. The cathodes are again lead plates $\frac{1}{16}$ in. thick. The box contains four compartments, each holding five anodes and six cathodes. The distance between plates is 4 in. The compartments are connected in series and receive 450 amperes, equivalent to five amperes per square foot of anode surface. The average drop of potential of the box is eight volts. As an anode, the copper on the plate dissolves and precipitates on the cathode where it slimes and falls to the bottom of the box. The gold released from the copper falls to the bottom of the anode frame. When the plates are cleaned down to the lead, they are lifted out, washed and replaced as cathodes in the first electrical box, to be used over again. The time required to clean a plate varies from 48 to 72 hours. Both the cyanide electrical box and the acid refining box are on the same circuit.

The gold precipitate melts up into bullion, 750 to 850 fine; the copper is shipped as cement copper to smelters and brings prevailing prices and also a return for the small amount of gold it contains. With the system of precipitation some of the difficulties of cyaniding cupriferous ores are eliminated. With the constant removal of most of the copper from the solution the extractive power of the latter is maintained, and the high consumption of cyanide is partly offset by this regeneration, and by the sale of the recovered copper.

Reliance Mill.—Douglas Lay[1] discusses the electrolysis of cyanide solutions at this mill at Nelson, B. C. He used sheet iron anodes of No. 10 plate, 5 ft. 1 in. x 2 ft. 8 in. The cathodes are of sheet lead, the same size as the anodes. The circulation of solution is over one anode and under the next, in the style of zinc box compartments. The current density employed is very low, 0.02 to 0.03 ampere per square foot of cathode surface. The electrodes are connected in multiple. The electrolyte is maintained at a high alkalinity to prevent the formation of prussian blue. The anode product is ferric hydrate, which is removed by floating it off the top of the solution. The cathode is sold to lead smelters on the lead bullion basis. The method presents nothing particularly new.

The Tavener Process.—This method of refining precipitates as prac-

---

[1] *Eng. and Min. Journ.*, Vol. LXXXIII, p. 801.

ticed at present is described by L. A. E. Swinney[1] for one of the largest Rand mines. The cakes of precipitate from the filter-presses are dried in cast-iron pans 3 ft. long, 2 ft. wide, 5½ in. deep in a special drying furnace. A low heat is employed. The dried slimes are roughly sampled and weighed. To 23,000 oz. of precipitate were added 32,000 oz. PbO, of which 3000 oz. were reserved for a cover; carbon, in the form of coal, 3200 oz. or 10 per cent. of the PbO; 3248 oz. of iron in the form of tin plate scrap; 12,647 oz. of assay slag; 6748 oz. silica in the form of clean quartz sand tailings. This gives the charge the following percentage composition; Precipitates, 30; litharge, 37.4; coal, 4.1; assay slag, 16.0; iron, 4.2; silica, 8.3. This mixture is smelted in pan reverberatory furnaces, described in The Mineral Industry, Vol. XI. The mixed product is placed on the pan of the furnace, upon which had been placed the lead bullion obtained from the last by-product smelt. Only about ⅓ of the iron is added with the mixed product, the remainder being added when the smelt is nearly complete. The smelting then proceeds until the charge is liquid, when the remainder of the iron is added. The temperature is then maintained at a high point and the charge rabbled from time to time. Finally the slag is drawn off to pots, and pots assaying under 1 oz. are sent to the slag dump, while those over this figure are resmelted. The furnace is then cooled somewhat and the last of the slag skimmed from the surface of the lead bullion and the bullion tapped into molds; 28,204 oz. of lead bullion were obtained. Any matte formed is treated in the by-product smelt.

Cupellation of the lead bullion is carried on in an English cupelling furnace; the test is made of bone ash of 80- and 40-mesh sizes, put through a 64-mesh screen; 3000 oz. of bone ash are used for one test, mixed with 9 per cent. water containing 14 lb. of caustic potash. A carefully made test lasts through two cupellations. Cupellation is carried on in the usual manner; when the final litharge is skimmed off, a flux of sodium carbonate, borax and silica is thrown on the gold, melted to clean it, then skimmed off; the gold is then cooled, but while still plastic is broken up into pieces; it is then melted in crucibles and cast into bullion bars. The fineness of bullion produced averages 860. About 25 per cent. of the litharge is lost in the slag. The accumulations of sweepings, rich slag, old tests, etc., are smelted in a by-product smelt from time to time, with approximately the same charge as that given above. The cost of the process is 7.65c. per fine ounce of bullion produced, distributed as follows, for a smelt of 2959.35 oz.: Coal, $16.27; borax, $16.55; litharge, $5.32; coke, $3.52; labor, $172.80; total, $214.46.

Refining Precipitates.[2]—In the method employed by the Homestake

[1] *Trans.* I. M. M., Nov., 1906. Jan., 1907.
[2] Private communication from Allan J. Clark.

Mining Company almost all acid treatment has been eliminated except that of very low-grade precipitate ($3 per lb.) of which only little is produced. Most of the material has a value of approximately $27 per lb. The precipitate comes from the triangular-frame precipitating presses of the three cyanide plants. At each of the plants the presses are cleaned up in the usual manner, the precipitate dried in a rectangular zinc-lined box, level with the floor, heated from beneath by steam coils, and covered by a hood provided with an exhaust pipe connected with a fan. When dry, the precipitates are mixed in this box with the fluxes, in the following proportion: Precipitates, 100 parts; litharge, 125 parts; borax, 20 parts; assay slag, 12 parts; and some reducing agent. The mixed product is carried in square boxes 18x18x15 in., provided with handles, to the "assay office" or refining works. Here they are briquetted, practically dry, in a simple piston briquetting press actuated by air at 850 lb. The briquets are 5 in. diameter and 4 in. high and weigh 5 lb.; they are put into large sheet-iron trays, holding 50 briquets and placed in a special baking furnace. This furnace is 20 ft. long and 5 ft. wide. It consists practically of a large muffle, the bottom being constructed of ribbed cast-iron plates, under which pass the products of combustion from the grate, heating the space above. The trays are put in at the back of the furnace, being slid in endwise at a side door, while they are discharged at the front end, broadwise. As one pan is taken out at the front the others are moved forward by special hook tool and a fresh pan put on at the rear. In this way the briquets are very gradually heated, being finally discharged at a dull red heat. During the heating, some mercury is driven off. The hot briquets are placed under a hood to cool off. Coke is used as fuel in the baking furnace.

The smelting of the briquets and the subsequent cupellation are both carried on in a cupelling furnace fired with wood. The test is water jacketed at the sides, is of 4 ft. diameter, and is made of crushed limestone, maximum size 8-mesh, and Portland cement, the ratio being 30 per cent. limestone to 70 per cent. cement. The furnace is heated, some lead bullion is put into the test, and when hot enough the briquets are charged on this bath of lead. When the briquets have thoroughly fused down and the slag become liquid, the slag, with which comes some matte and a little lead, is drawn off into slag pots and a fresh charge of briquets is added. The lead is recharged. This procedure is repeated until all the briquets have been fused down, when the heat is raised, all slag closely skimmed, the air blast turned on and the lead bullion cupelled. Just before the "set" of the gold, a heavy sheet-iron divider, in the form of rectangles is pressed deeply into the bullion, dividing it into cubes and affording a ready means of breaking it up into pieces for the melting pots. The bullion is cast into bars of 975 to 980 fineness. The litharge from

the cupellation contains 0.6 per cent. of the value of the original precipitates and is ground and used for the fluxing of the next lot of precipitates. The matte and slag from the first smelting of precipitates, together with the old tests, are resmelted in a 36-in. blast furnace, some metallic iron being added to precipitate the lead from the matte, which is chiefly lead sulphide. The products of this smelting are slag, still containing considerable lead, high in zinc, but low enough in gold to be discarded; lead bullion, containing 4.5 per cent. of the original values in the precipitate, and used as the lead bath to start the smelting of the next lot of precipitates; and matte, containing 0.03 per cent. of the value in the precipitates, which is held for shipment.

The slags produced in the smelting are interesting metallurgically. That from smelting the briquets varies, but has the following approximate composition: SiO_2, 8 to 12; Pb, 30 to 40; Zn, 15 to 20; CaO, 5 to 8; FeO, very little; B_2O_3 and Na_2O, the balance. They are thinly fluid. The slag produced finally in the blast furnace, small in amount, has this approximate composition: SiO_2, 24; Zn, 24 to 27; Pb, 20 to 22; Fe, 3; CaO, low; B_2O_3 and Na_2O, the balance. The high percentage of zinc can be carried only in the presence of lead silicate and considerable alkali and boric acid. The matte produced in the smelting of the briquets is chiefly lead sulphide and, as no sulphuric acid treatment is employed, small in amount.

The cost of refining per ounce of fine gold is 12.64c.; the cost per ounce of gold and silver is 9.25c. The amount of bullion handled by this method is about $100,000 per month.

Progress in Gold Milling in 1907.

By Robert H. Richards and Charles E. Locke.

Mill Design, Construction and Equipment.

Light Stamp-Battery Frame.[1]—Under conditions where the largest lumber available was 8x8-in. x20 ft., mountain pine cross grained and knotty, a satisfactory frame was constructed which stood the test of time perfectly. The mortar block, 28-in.x54-in.x12-ft., was built up of 2x12-in. and 2x6-in. planks, using 4x6-in. buckstaves, $\frac{3}{4}$-in. bolts and 40-d. nails. This was set on 24-in. of concrete resting on solid clay, and the spaces between the mortar block and the sides of the hole were filled with concrete. The frame was of the front knee type. Each post was built up of three 8x8-in. timbers bolted with $\frac{3}{4}$-in. bolts and keyed with 3-in. oak blocks. Where the posts were notched for the cam shaft they were reinforced by doubled 2x12-in. planks bolted and spiked inside the posts. The posts had a 12x2-in. tenon mortised 6 in. deep into

[1] F. A. Thompson. *Bulletin* Colorado School of Mines, Vol. IV (1907), p. 7.

the 8x8-in. cross sills which in turn rested upon the concrete around the mortar block and were secured to a 6x12-in. mud sill at each end. The guide girts were 12x12-in., built up of 2x12-in. planks. All braces were wedged tightly into place and tie rods were inserted parallel to every member of the frame except the battery posts. In the following comparison of the timbers used for this frame and for a California frame of the same type, using the same weight of stamp, the dimensions of the California frame are given first, in each instance, as against corresponding dimensions of the light frame: Battery posts, 14x24-in. against 8x24-in.; mudsills, 14x16-in. against 6x12-in.; cross sills, 12x16-in. against 8x8-in.; knee posts, 16x18-in. against 8x8-in.; knee stringers, 12x16-in. against 8x8-in.; knee braces, 8x12-in. against 8x8-in. The total lumber for a five stamp battery was 3500-ft. board measure, against 1180.ft. board measure.

Concrete Mortar Blocks.[1]—A full description, with detailed drawings, is given of the new battery foundations of the Kuk-San-Dong mill in Korea. Among the special features are: (1) The diagonally placed anchor bolts for the mortars, so set (instead of vertical) for ease of replacement; (2) cross sills anchored by vertical $1\frac{3}{4}$-in. bolts; (3) posts anchored to the cross sills by four vertical $1\frac{3}{4}$-in. bolts. The results have been satisfactory. The stamp duty per 24 hours has been increased 1.3 tons over that of the older mill which had all other conditions the same, and the breakage of battery parts has been slightly less.

Single-Stamp Batteries.[2]—Among the objections to this form of stamp mill the following are noted: (1) The difficulty of obtaining a good feeder for the individual stamps. (2) The difficulty of keeping the driving pulley keyed tight to the cam shaft when this pulley is at the middle of the shaft. When the pulley is at the end of the cam shaft this trouble does not occur. (3) Individual stamps weighing 1250-lb. dropping 130 times per minute, hight of drop $5\frac{1}{4}$-in., are too heavy for a soft friable ore and the fast drop tears the battery parts and frame to pieces. (4) The mortars tend to fill up and the hight of discharge varies so that there is considerable wear and tear on the screens with new dies. (5) The discharge is uneven when feeding mixed hard and soft ore. A large amount of water must be used. Inside feeding of mercury cannot be practiced successfully and the outside plates require more attention. (6) One individual stamp is said to require as much attention as a five-stamp battery. There appears to be no question that the single-stamp mill has a high crushing capacity. A rearrangement of the mill is suggested to the following order: Stamps, classifier, concentrator, slime tables, amalgamated plates and no amalgamation in the battery. Classification

[1] C. D. Kaeding. *Min. and Sci. Press,* Vol. XCIV (1907), p. 598.
[2] *Min. and Sci. Press,* Vol. XCIV (1907), pp. 115, 147, 303.

before concentrating is theoretically the proper method provided proper concentrators are used. Practically, better results are often obtained by concentrating the pulp direct without classification.

Where the ore contains coarse gold and the concentrates are treated by chlorination or cyanidation it is not proper to place the amalgamation after the concentration. On some ores inside amalgamation is required for best results.

Removal of Broken Stamp Stems.[1]—When a broken stem is left in a boss it may be blasted out with dynamite but the boss is liable to be cracked. Removal by steam hammer is expensive and by hand hammer tedious. The best method is to heat by blast lamp to expand the metal and squeeze out the broken end of the stem in a hydraulic press.

Stamp Mill Plant of the New Kleinfontein Company, Limited, Witwatersrand.[2]—This new plant consists of a crusher house containing the breakers, screens, picking belts, etc., and the mill proper containing 200 stamps, amalgamating plates and concentrating tables. The last is followed by the tailings wheel and cyanide plant. This article is rich in details of all the individual parts of the installation.

Design and Working of Gold Milling Equipment with Special Reference to the Witwatersrand.[3]—This article takes up all the parts of a stamp mill in detail and is fully illustrated. It contains a large number of valuable data for the gold mill man. Owing to the nature of the article it is impossible to reproduce it in a condensed form.

General Milling Practice.

Modern Stamp Mills.[4]—The old stamp mill still remains standard, but has been improved in the line of cast-iron anvil blocks, heavier stamp , shorter lifts, Blanton self-tightening cams and sectional guides. South African stamps are used as heavy as 1300 to 1400 lb., dropping 10 in. 100 times per minute and crushing 5 to 5½ tons per stamp per 24 hours.

The Alaska Treadwell mill has stamps weighing 1050 lb. and crushing 4.95 to 5.79 tons. In the Black Hills 850-lb. stamps crush 4½ tons. The Tenero mill has two stamps per mortar with four screen openings and each stamp crushes six tons per 24 hours through 24-mesh screens. The Tremain steam stamp has two stamps in one mortar, each stamp making 200 or more strokes per minute and crushing from four to 10 tons. The Krause atmospheric stamp makes 190 strokes per minute and crushes 2½ tons per hour of 1½ to 2-in. hard copper-bearing rock through a 16-mesh screen. The blow is struck at a speed of 1200 ft. per minute.

[1] Q. C. McMillan. *Journal*, Chemical, Metallurgical and Mining Society, South Africa, Vol. VII (1906—07), p. 269.
[2] E. J. Way. *Proc.*, Inst. of Civil Engineers, Vol. CLXVIII (1907), p. 252.
[3] G. A. Denny. *Proc.*, Inst. of Civil Engineers, Vol. CLXVI (1906), p. 243. *Le Genie Civil*, Vol. L (1907), p. 232.
[4] C. C. Christensen. *Engineering Magazine*, Vol. XXXIV (1907), p. 29.

Brief outlines are given of (1) stamp mill for free-milling ore, (2) stamp mill with concentrators, (3) wet-crushing silver mill, (4) dry-crushing mill and (5) straight concentrating plant.

Stamp Mill Records.[1]—In running stamp mills, records should be kept of all the details. For each shift the following headings are suggested: Date; shift (day or night): ore hoisted; hours mill run; cause of stops; mercury used; mercury recovered: net mercury used; amalgam recovered; assay of tailings; remarks. For the total run the following headings are suggested: Number of tons; hours mill run; tons per stamp per 24 hours; drops per minute; hight of drop; hight of discharge; screen; weight battery amalgam; weight battery retort; percent. gold in battery amalgam; weight plate amalgam; weight plate retort; percent. gold in plate amalgam; percent. gold in total amalgam; percent. of total gold saved inside; weight smelted gold; loss in smelting; value smelted gold per ounce; total value smelted gold; value per ton of ore; weight of concentrates; assay of concentrates; total value of concentrates; value of concentrates per ton of ore; average value of tailings per ton; total value of ore per ton; percent. of total value saved; total mercury used; total mercury recovered; net mercury used; ounces gold recovered per net ounce mercury used; loss of mercury; loss of mercury per ton of ore crushed.

Cleaning up Sands from Roller Mills.[2]—The residual battery sands from the bottoms of Huntington, Chile and other mills may be quickly cleaned up by screening on a 4-mesh screen and running the undersize over a Wilfley or Overstrom table. Gold and amalgam, together with iron, form the concentrates and the iron may be removed later by a magnet.

Gold Milling.[3]—The efficiency of a stamp mill may be increased by the use of a Blake breaker to crush to 2-in., a trommel with 1-in. holes and a Gates breaker to crush the oversize of the trommel. This would allow a reduction in the weight of the stamps and their hight of drop and an increase in the number of drops. The diameter of the shoes and dies may well be increased from the present 9 or 10-in. to 12 or 14-in. Huntington mills are thought to be close competitors to tube-mills for fine grinding. The area of amalgamated plates should be 250 sq. ft. for five stamps. Plates should be held down by clamps, not by screws, and should overlap. No more water should be used than is necessary to wash the crushed ore over the plates in a thin film. Too many chemicals such as cyanide of potash, nitric acid, etc., should not be used for cleaning plates. The use of hydraulic classification before concentration appears logically to be preferable to its omission.

---

[1] E. Percy Brown. *Canadian Min. Journ.*, Vol. I (1907), p. 426.
[2] H. S. Kenny. *Mining Reporter*, Vol. LV (1907), p. 201; from *South African Mines*.
[3] *New Zealand Mines Record*, Vol. X (1907), p. 330.

Modern Gold Milling.[1]—Modern gold mills are largely constructed of steel instead of wood, are well lighted and ventilated, solidly built and roomy. Concrete enters largely into the foundations, floors and mortar blocks, and when properly mixed and carefully placed it gives satisfactory results. This is especially true of concrete mortar blocks, which should be carefully leveled off and covered with a thin sheet of rubber for the mortar to rest upon. Heavy anvil blocks are unnecessary. There is a tendency toward heavy stamps using a coarse screen and having a high capacity. At the Knights Deep mill in South Africa 1450-lb. stamps have a capacity of 7.8 tons per stamp per 24 hours. Tube-mills are used for regrinding. Feeders of the suspended type are preferred. Where tube-mills are used a moderate length (about 13 ft.) is considered to be more efficient. Silex linings are generally preferred except in Australia where chilled iron is favored. A thick pulp with about 50 per cent. moisture and well classified gives the maximum crushing efficiency. The controversy as to the relative merits of tube-mills and grinding pans is still unsettled.

At the Liberty Bell 80-stamp mill, Telluride, Colorado, the stamps weigh 850 lb. and crush 4.3 tons per 24 hours through a 14-mesh screen. The pulp passes over amalgamated plates and is classified in a Dorr classifier which sends the thickened coarse material to three Abbe[1] tube mills (two at work and one in reserve). These are 22 ft. long, 5 ft. diameter, run at 25 r.p.m. and grind about 95 tons each in 24 hours. The charge of pebbles is about 14 tons which fills the mill two-thirds full and the pebble consumption is 1.3 lb. per ton crushed. The ground product has 0.40 per cent. on 40-mesh, 13 per cent. on 80-mesh, 15 per cent. on 100-mesh, 23 per cent. on 200-mesh and 48 per cent. through 200-mesh. It passes over amalgamated plates and thence to the cyanide plant.

In addition to its use before cyaniding the tube-mill has been employed for recrushing stamp pulp to secure further recovery by amalgamation or concentration or both. The cost of installing and running tube-mills is high. Average figures for running cost are as follows: Power, 3.6c.; wages, 2.82c.; liners, 2c.; maintenance (including tables), 1.36c.; pebbles, 1.16c.; sundry stores, 0.48c.; total, 11.42c. per ton. Figures for the Combination mill at Goldfield, Nevada, are considerably higher owing to the high cost of labor and freight. They total 41.5c. per ton.

Mechanical Treatment of Gold Ore.[2]—Among changes in the last decade are the following: The Utica mortar has increased the stamp duty per 24 hours from 3 to 5.5 tons by the addition of steel liners to the back and sides of the mortar so that the shoes just have clearance. Concrete mortar

[1] G. P. Scholl. *Electrochem. and Met. Ind.*, Vol. V (1907), p. 14. *Journal* Chemical, Metallurgical and Mining Society South Africa, Vol. VII (1906-07), p. 379. *Revue de Metallurgie*, Vol. IV (1907), p. 840. *Min. and Sci. Press*, Vol. XCIV (1907), p. 211.

[2] W. J. Adams. *Min. and Sci. Press*, Vol. XCV (1907), p. 374.

blocks have replaced wooden ones. The Standard mill will now have also mortars of low hight through the covers of which the boss passes; the guides are iron or steel rings; the stamps are hung up from the battery floor; the cams and pulleys are self tightening. Chile mills give results which compare favorably with those obtained with stamps.

Testing Gold Mill Tailing.[1]—Where a high loss occurs in the tailing of a gold mill the source of the loss may be determined by taking a sample of about 2 lb. of tailing and amalgamating it with about 100 grains of clean mercury for an hour. The amalgam, concentrates and tailings are separated from each other and the tailings are further separated by sizing and decantation into several sizes and each size is assayed. A study of the results will show where the losses occur. If in the coarse tailings, then finer crushing is probably the thing; if in the slime, then it would indicate either hard amalgamation, base mineral in the ore that fouls the amalgam, or the ore is being over milled.

Concentration of Slime.[2]—In the California gold mill the vanners save the sulphides coarser than 150-mesh. Further recovery may be made by treating the vanner tailings in classifiers to remove the coarse gangue and running the fine overflow over canvas tables. Two systems of canvas table treatment may be used: (1) Classification of the fine material into two or more grades, each grade to be run over a properly adjusted canvas table; (2) treatment of all the fine material direct on canvas tables followed by a second classification to remove coarse particles of gangue and a final treatment of the very fine particles on differently adjusted canvas tables. The first system saves 15 to 20 per cent. from the mill losses while the second system saves in one instance nearly 50 per cent. The author's plant has 20 stamps and the vanner tailings are run direct in a sluice system over cloth of twisted drilling with a width of distribution of about 1 ft. per stamp and a flow of 12 ft. This saves coarse mineral that has escaped the vanners. The coarse particles of waste are next removed by a hydraulic classifier and the fine material is passed over a sluice system with four times the original width of distribution. A second hydraulic classification takes out all sand coarser than 120-mesh and the pulp is treated on wide tables of the Gates style with six times the original width of distribution. The canvas concentrates in this plant were rich enough to be chlorinated direct without cleaning on special vanners. Good classification is essential for success in canvas plants.

A recent invention has the canvas tables mounted in tiers in a circle on the principle of a merry-go-round. The whole thing revolves slowly around a central axis and is perfectly automatic in its action.

One source of loss in gold milling arises from the flotation of values

---

[1] W. E. Darrow. *Min. and Sci. Press*, Vol. XCV (1907), p. 300.
[2] W. E. Darrow. *Min. and Sci. Press*, Vol. XCV (1907), p. 268.

by oil, either mineral oil which gets in accidentally from the lubrication of machinery, or vegetable oil derived from the crushing of wood chips in the mortar.

The Use and Care of Mercury.[1]—This article is full of valuable information for the gold mill man. It considers the properties of impure mercury, the methods of purification, the formation of amalgams with various metals, the causes of "flouring" and sickening" of mercury and the remedies therefor.

Milling in South Africa.[2]—The following example shows the advantage of ore sorting in South Africa. A mine contains 7,200,000 tons of ore that will yield $10 per ton. The capacity of the mill is 1000 tons per day.

Cost of mining and surface expense	$ 6 per ton
Cost of milling and cyaniding	1 per ton
Net profit	3 per ton
7,200,000 tons give net profit in 20 years of	21,600,000
Profit per year	1,080,000
$1 per year at compound interest 20 years	33.066
$1,080,000 per year at 5 per cent. for 20 years	35,711,280

If 15 per cent. of the ore (1,080,000 tons) worth $1 per ton, is sorted out there remains 6,120,000 tons to be milled, having a gross value of $70,920,000 and costing:

7,200,000 tons hoisted at $6	$43,200,000
Sorting 7,200,000 tons at 5c	360,000
Milling 6,120,000 tons at $1	6,120,000
Total costs	$49,680,000
Product 6,120,000 tons at $11.588	70,920,000
Net profit	$21,240,000
Or $1,249,411 per year for 17 years.	
$1 per year at compound interest 17 years	$ 25.84
$1,249,411 at interest 17 years	32,284,780
$1 at 5 per cent. compound interest 3 years	1.1576
$32,284,780 at interest 3 years	37,372,861

This last figure is the total profit at the end of 20 years in a sorting system. Comparison of this with the previous $35,711,280 profit of non-sorting system shows an increase of $1,661,681 and the mine is worked out in three years less time and the plant sold or utilized on another property.

Ore sorting is practiced in three ways: (1) The old method of sorting on a floor; (2) revolving steel tables about 30 ft. diameter; (3) Robins conveying belts. The last method appears to be the most approved. A mine with 200 stamps requires 50 or more negroes (chiefly young boys) and two white overseers for sorting. Trommels about 4 ft. diameter and 10 to 18 ft. long are used to wash the ore before picking. The waste picked out amounts to from 10 to 30 per cent. of the total ore hoisted.

There are 7000 stamps running on the Rand. The largest mill is the joint 400-stamp mill of the Simmer East and Knights Deep. The crushing capacity is 5 to 8 tons per stamp in 24 hours and it takes 7 to 9 tons of water per ton of ore. The weights of the stamps vary from 1150 to 1600 lb. The wire screens used vary from 20- to 35-mesh. The coarse

[1] *Min. and Sci. Press*, Vol. XCV (1907), p. 216.
[2] J. H. Pitchford. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 467. *Min. and Sci. Press*, Vol. XCIV (1907), pp. 311, 337. *Mines and Minerals*, Vol. XXVIII (1907), p. 49.

screens are used where there are tube-mills. It is customary to install two tube-mills for a 100-stamp mill or three for a 200-stamp mill. The tube-mills are 5 ft. 6-in. diameter and about 20 ft. long inside the silex liners. From 50 to 60 per cent. of the gold recovered is caught as amalgam on the copper plates. The rest is obtained in the cyanide plant from the concentrates, sands, and slimes treatment except a small quantity which is sent to the chlorination works.

Not all of the mines use concentration. Where it is used Frue vanners or Wilfley tables are placed below the battery amalgamating plates. Shaking amalgamating tables covered with copper plates are used below the tube-mills.

Milling Practice at El Oro, Mexico.[1]—The ore undergoes the regular preliminary treatment in rock breakers and stamps which reduce it to 35-mesh size. Tube-mills are used for regrinding the coarse sand preparatory to cyaniding. Details of these mills are shown in the table:

DETAILS OF EL ORO TUBE MILLS.

	Abbé 1	Abbé 2	Krupp 3	Krupp 4	Krupp 5
Internal diameter of drum, in inches	54	54	47	59	59
Internal length of drum, in feet	19.5	19.5	19.5	23	26
Weight of pebbles when mill is half full, in pounds	11,000	11,000	11,700	21,800	24,700
Revolutions per minute	31	31	31	25	27
Horse-power to start	125	125	125	225	225
Horse-power to run	58	43	48	80	87
Weight of set of El Oro iron bar plates, in pounds	14,836	14,836	13,732	20,246	24,566
Amount of sand treated per 24 hr. in tons	100	100	100–120	180–200	230–300
Slime made per 24 hr. in tons	25	25	30	45	60
Pebbles consumed per ton ground, in pounds	....	....	8.44	8.44	8.44
Cast iron liners consumed per ton, in pounds	....	....	1.04	1.04	1.04
Percentage of time mills actually run in July	40.9	54.3	89.6	86.8	88.2

Silex liners, put in flat, last 2.5 months; flat 1-in. cast-iron plates last 2 or 3 months; 1.5-in. cast-iron liners from 3 to 4 months; and El Oro cast-iron plates with bars last from 6 to 8 months. With the last the slots between the bars catch the pebbles which take the wear instead of the iron. Sizing tests of the feed and product are shown in the accompanying table:

SIZING TESTS ON EL ORO TUBE-MILLS.

Size.	Abbé No. 2		Krupp No. 3		Krupp No. 4		Krupp No. 5	
	Feed.	Product.	Feed.	Product.	Feed.	Product.	Feed.	Product.
On 40-mesh	4.5	0.0	1.5	0.0	21.0	0.0	22.5	0.0
Through 40-on 60-mesh	18.5	4.5	12.0	0.5	21.0	2.0	20.0	3.5
Through 60-on 80-mesh	17.0	10.0	17.0	1.5	15.5	6.5	12.5	7.0
Through 80-on 100-mesh	13.5	10.5	17.5	2.0	10.5	8.0	10.0	9.0
Through 100-on 150-mesh	23.0	31.5	35.0	14.5	16.5	29.0	21.5	28.5
Through 150-on 200-mesh	4.0	16.0	5.0	32.0	5.0	12.0	5.5	11.0
Through 200-mesh	20.0	28.0	13.0	50.0	10.5	43.0	8.0	41.5
Tons sand ground	....	95	....	60	....	220.9	....	228
Tons slime made	....	19	....	38.4	....	86.9	....	92.8

[1] E. Burt. *Min. World.* Vol. XXVII (1907), p. 699.

Milling in Western Australia.[1]—In the Kalgoorlie district four mills crush in Krupp ball-mills, one employs Griffin mills, one Krupp and Griffin, one stamps without regrinding, and four use stamps for further classification and fine grinding. Regrinding and sliming are done in Wheeler pans and tube-mills. Two processes are used: (1) Dry-crushing, all-roasting and all-sliming. (2) Wet-crushing, classifying, concentrate-roasting and raw slime bromocyaniding.

WESTERN AUSTRALIAN MILLS.

Name.	Monthly Tonnage.	Mills used.
All-Roasting:		
Associated	8,700	10 No. 5 Krupp
Associated Northern	3,500	3 No. 5 Krupp
Great Boulder	14,000	2 No. 8 Krupp and 12 Griffin
Kalgurlie	11,000	9 No. 5 Krupp
Perseverance	15,000	16 Griffin
South Kalgurlie	8,500	2 No. 8 Krupp and 1 No. 5 Krupp
Boulder Main Reef	2,300	2 No. 5 Krupp
Diehl-Bromo Cyaniding:		
Golden Horseshoe	24,000	150 stamps
Ivanhoe	17,000	100 stamps
Lakeview	11,000	70 stamps
Oroya Brownhill	10,500	50 stamps

Figures are given on men, costs, water used, wear of fine crushing machines, sizing tests of products of different forms of grinders, capacities, extractions. The Diehl process has the following steps: (1) Wet-crushing in stamps with or without amalgamation. (2) Concentration by means of spitzkasten and Wilfley tables. (3) Elimination of the refractory concentrate equivalent to 5 or 6 per cent. of the ore. (4) Roasting of the concentrates and fine-grinding, amalgamation and cyanide treatment of the same. (5) Fine-grinding of the lighter material by tube-mills or grinding pans and bromocyaniding the product. Details of the process as practiced in the different mills are given.

Individual Mills Described.

Montana Tonopah Mill.[2]—The proposed scheme of this mill is as follows:

1. Gates breaker, No. 5. To (2).
2. Two Gates breakers, No. 3, crushing to ¾-in. By elevator to (3).
3. Mill bins. To (4).
4. Forty stamps with 12-mesh screens, crushing 150 to 200 tons in 24 hr. To (5).
5. Classifiers. Spigots to (6); overflow to (7).
6. Wilfley tables. Concentrates; tailings to (7).
7. Two Dorr classifiers. Spigots to (8); overflow to (9).
8. Two tube-mills each 5x22 ft., grinding to 150-mesh. To (9).
9. Two large cone classifiers. Spigots to (10); overflow to cyanide plant.
10. Sixteen Frue vanners. Concentrates; tailings to cyanide plant.

Bullfrog Cyanide Mill.[3]—The ore goes first to (1).

1. Blake breaker, 15x24 in. By elevator to (2).
2. Vezin sampler. Sample elevated to (3); rejected portion to (4).

[1] Ralph Stokes. *Min. World*, Vol. XXVI (1907), pp. 4, 34. *Electrochem. and Met. Ind.*, Vol. V. (1907), p. 62.
[2] F. L. Bosqui. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 805.
[3] E. R. Ayres. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 376.

3. No. 2 Vezin sampler. Sample crushed and sampled down by hand; rejected portion to (4).
4. Mill bin. To (5).
5. No. 1 rolls, 42x14 in. By elevator to (6).
6. Coarse trommel. Oversize to (7); undersize to (8).
7. No. 2 rolls, 42x14 in. By elevator to (9).
8. Single unit dry screen with 8-mesh scalping cloth and 16-mesh primary cloth. Oversize to (7); undersize to (14).
9. Two 8-mesh, dry, centripact screens. Oversize to (12); undersize to (10).
10. Two 16-mesh wet trommels. Oversize to (11); undersize to (14).
11. Unwatering screen. Oversize to (12); undersize to (13).
12. No. 3 rolls, 42x14 in. To (13).
13. Elevator. To (10).
14. Amalgamating plates and traps. Pulp to (15).
15. Dorr classifiers separating material into sand and slimes for the cyanide plant.

Octave Mill.[1]—The ore is crushed in 40-stamps, passes over amalgamation plates and is lifted by two Frenier sand pumps to two 16-ft. Dimmick classifiers which separate the sands into four spigot products and deliver the slimes in the overflow. The spigots go to eight Wilfley tables and the overflow to Wilfley slimers. The middlings from the Wilfley tables, together with the oversize, is elevated to a Huntington mill the product of which is pumped to the classifiers. All tailings go to the cyanide plant.

Mill of the Guanajuato Reduction and Mines Company.[2]—The ore is crushed through a $1\frac{1}{8}$-in. ring by two gyratory breakers in tandem. It is then conveyed and elevated by a traveling belt to the mill bin (2500 tons capacity) of the 80-stamp mill. Stamps weigh 1050 ℔., make 100 drops of 7.5 in. per minute and have 26-mesh, No. 28-wire, steel screens. The mortars weigh 9000 ℔. each and have an extra broad base bolted to heavy concrete mortar blocks. The stamp pulp is classified in spitzkasten into coarse, fine and overflow. The coarse goes to Wilfley tables whose tailings are reground in an Abbé tube-mill and concentrated on Johnston vanners. The fine product is concentrated on Wilfley tables. The overflow passes to a second spitzkasten whose spigot goes to Johnston vanners. All tailings and slime go to the cyanide plant.

Stamp Milling in Nova Scotia.[3]—At the Richardson gold mine near Goldboro, Nova Scotia, the ore is slate and quartz containing free gold and arsenopyrite. It is dumped from the hoisting skip over a grizzly and fed by hand into one of two 9x15-in. Blake breakers. The crushed ore is trammed to the mill bins and fed by suspended Challenge feeders to 60-stamps. The pulp passes over silver-plated apron plates, 12 ft. long, to six Wilfley tables. The concentrates of arsenopyrite are treated by bromo-cyanide; the tailings are waste.

Forty stamps have wooden mortar blocks and 20 have concrete. The latter is more durable when properly constructed. The stamps weigh 950 ℔. and drop 7 to $7\frac{1}{2}$ in., 95 to 98 times per minute. The hight of discharge is about 6 in. The screen used is 18x24-mesh, No. 26 wire, twilled. The 30-mesh plain wire screen, No. 28 wire, formerly used, cost twice as much, lasted half as long and gave less efficiency to the

[1] W. A. Root. *Min. World*, Vol. XXVI (1907), p. 246.
[2] C. W. Van Law. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 649.
[3] E. Percy Brown. *Canadian Min. Journ.*, Vol. I (1907), p. 207. *Min. Reporter*, Vol. LVI (1907), p. 31.

stamps with little difference in the saving of gold. Water is fed at the back of the mortars on a level with the middle of the die. No inside plates are used. Three men are employed on day shift and two on night shift.

Stamp Milling at the Great Fingall Mine.[1]—There are two mills, the old of 60 stamps and the new of 40 stamps. The latter was erected in 1904. Each mill also has six Wheeler pans, 5 ft. diameter, for regrinding. The ore is quartz containing free gold with auriferous pyrites and a little argentiferous galena.

1. Blake breakers, crushing to 2 in. To (2).
2. Stamps usually crushing to 10-mesh size. To (3).
3. Splash plate 8 in. wide or lip plate 12 in. wide. To (4).
4. Apron plates. Pulp to (5).
5. No. 1 Wilfley tables, one for each stamp battery. Concentrates to roaster; tailings to (6).
6. A 40-in. sand wheel elevating to (7).
7. Two spitzkasten in parallel, each 3x3x3 ft. Spigots to (9); overflow to (8).
8. Spitzkasten, one in each mill, 5x5x5 ft. Spigot to (9); overflow clear water for plates, etc.
9. Wheeler pans. To (10).
10. Six-inch splash plate followed by a short amalgamated copper plate. Pulp to (11).
11. No. 2 Wilfley tables, one for each pan. Concentrates to roaster; tailings to waste.

The accompanying table shows the results of testing the pulp coming to the No. 1 Wilfley tables, June 6 to 10, 1906.

SIZINGS, ETC. OF PULP AFTER AMALGAMATION AND BEFORE CONCENTRATION.

Sample No.	Stamp Screen.		Sizing Tests.			Tons ore per stamp per 24 hr. (*a*)	Gallons water per ton of ore.	Assay in gold per ton
	Mesh.	Wire i.s.w.g.	On 30-Mesh %	Through 30-on 40-Mesh %	Through 150-Mesh %			
1	10	22	36.5	10.65	24.65	7.25	1414	$10.25
2	10	22	37.15	10.80	23.25		1185	9.50
3	11	22	32.75	11.25	25.10	8.42 (*b*)	1758	6.10
4	11	22	30.10	11.75	26.75		1508	7.00
5	12	22	28.35	11.60	30.35	6.77	1552	6.00
6	12	22	30.40	12.00	29.15		1640	6.25
7	14	22	18.10	12.5	30.00	6.74	1800	....
8	14	22	18.75	13.2	29.25		1824	....
9	Patent slotted screen		19.25	12.15	32.4	6.18	1714	7.00
10	supposed 14-mesh and 22 I.S.W.G.		20.00	13.15	31.75		1534	6.00

(*a*) Stamp duties figured from samples. (*b*) Some discrepancy here.

Some details of the mill are as follows: Stamps, 1160 lb. each, 106 drops per minute, 8-in. drop, 10-mesh screen (21 i.s.w.g.), hight of discharge 2 to 5 in., no inside amalgamation. Frames of back knee type with cams in front. Mortar block in old mill made of eight 15x15-in. timbers, 13 ft. long, and resting on concrete on felsitic rock. Mortar block of same length as mortar base but $2\frac{1}{2}$ in. wider. New mill had concrete mortar blocks which cracked and were replaced by timber.

Sectional guides of tuart wood steeped in linseed oil several days before use. Guides greased three times in 24 hours with a little dark axle grease. Sectional cast-iron guides are too expensive. Mortar is of the narrow type. Mortar linings in the old mill mortar are of $\frac{7}{8}$-in. cast steel, front and back, 10 in. deep and 50 in. long, held by three $\frac{3}{4}$-in. countersunk

[1] Gilmore E. Brown. *Trans.* I. M. M., Vol. XVI (1906-07), p. 403.

bolts; side liners of $\frac{7}{8}$-in. boiler plate held by two bolts. New mill mortar has back, front, end, and feed-apron liners; end and front liners of $\frac{1}{2}$-in. boiler plate; back liner reinforced by $\frac{7}{8}$-in. cast steel liner; front liner dovetailed in, while other liners held by $\frac{3}{4}$-in. bolts. Back liners wear rapidly and last 290 days in the old mill, 240 days in the new mill.

Screen frames of 3x$\frac{1}{2}$-in. flat iron with opening 12 in. wide. Screens, 10-mesh (21 i.s.w.g.) or 12-mesh (22 i.s.w.g.), last 80 hours, slope 16$\frac{1}{2}$ deg. in the new mill mortar, area 828 sq.in., percentage of opening 46.2. Chuck blocks not used in new mill but in old mill chuck block is 3 in. high with new dies and is reduced to nothing in two stages as dies wear. Inside plates discarded because of scouring with coarse stamping and because of temptation for theft with former fine stamping. Dies of forged steel 7 in. high and 9 in. diameter. Net wear per ton crushed is 0.038 lb. or 0.25 lb. per day. Discarded die is 2 in. thick including foot plate. Dies wear slightly convex in center and concave around the edges. New dies set on layer of concentrates and kept in position by packing the spaces between them and the ends and sides of the mortar with wrought iron liners about 1 in. wide. False bottoms, 3 in. thick, in five pieces are used in the new mill and are kept in all the time. The old mill has a double layer, total thickness 6 in. Shoes of forged steel, 12x9 in, last 78 days, wearing at the rate of 0.082 lb. metal per ton crushed. Discarded shoes are about 1$\frac{1}{2}$ in. thick. Old shoes and dies are melted down to cast shoes and dies for grinding pans.

Stamp stems are turned to 3$\frac{1}{2}$ in. diameter, tapered at both ends. Length 16 ft. Taken out when length gets below 13 ft. Old mill averages 38 broken stems per month, new mill 44. Middle stamp breaks oftener than any other. More breakages in night than in day time. Fewer stems break after clean-up, probably due to removal of loose pieces of iron from mortar. 95 per cent. of breaks are close to the boss. Short stem is lengthened by forging on a piece of 3$\frac{7}{8}$-in. good quality iron. Broken ends of the stem are removed by blasting out with gelignite. Tappets of cast steel, 13 in. long, three untempered steel keys per tappet, last about two years. Tappets set on half the stamps $\frac{1}{4}$ in. daily. Two men reset 10 to 12 tappets per hour.

Cam shaft 6 in. diameter and 8$\frac{1}{2}$ ft. long for five stamps. Last two years. Spare cam shaft with cams on it kept in readiness for a break down so that broken cam shaft is replaced in 4$\frac{1}{2}$ hours. Order of drop 1-3-5-2-4. Cams of cast steel, 2$\frac{1}{2}$ in. wide, double armed, 16-in. radius to give 9-in. drop, held in place by wedges. Clearance between cam shaft and stems $\frac{3}{8}$ in. Feeders of the suspended Challenge type with special friction drive. Water supplied to each mortar by a 2-in. pipe with four nozzles. Water used is 6$\frac{1}{2}$ gal. per stamp per minute equivalent to 1.29 tons of water per ton of ore.

Apron plates are 11x4½ ft. in the old mill, sloping 1 in 11; 10x5 ft. in the new mill. A length of 22 ft. was tried but the last 12 ft. did not catch enough to keep it in condition. Ore contains 0.45 oz. gold per ton. Of the gold caught by amalgamation 10 per cent. comes from the mortar, 22 per cent. from the lip and splash plates and 68 per cent. from the apron plates. Clean pyrite contains 7 to 8 oz. gold per ton but the Wilfley concentrates run about 20 per cent. silica.

The old mill requires ten men and one foreman, and the new mill seven men and one foreman. Shifts are 8 hours long. On the day shift in the old mill one foreman, one assistant amalgamator, one feeder attendant and one laborer. On the afternoon and night shifts are one amalgamator, one feeder attendant and one laborer.

Milling costs in Sept. 1906, were: Labor and salaries, 11.77c.; power, 23.45c.; repairs and maintenance, 14.46c.; assaying and sampling, 0.36c.; fuel, 0.34c.; mercury (5480 oz.), 1.03c.; shoes and dies (35 shoes, 24 dies), 3.11c.; sundry supplies, 2.22c.; total, 56.74c. Fuel is wood and costs $4 per ton.

Treatment of Black Sands.

California Black Sands.[1]—These sands contain minerals ranging from 7 to 20 in specific gravity and most of the values are in the sizes below ⅛ in. The first step in the treatment should be grading to a definite size by means of grizzlies, riddles and screens. The next step is concentration in water by gravity, using many of the well known appliances. Next comes a drying operation which is followed by magnetic separation.

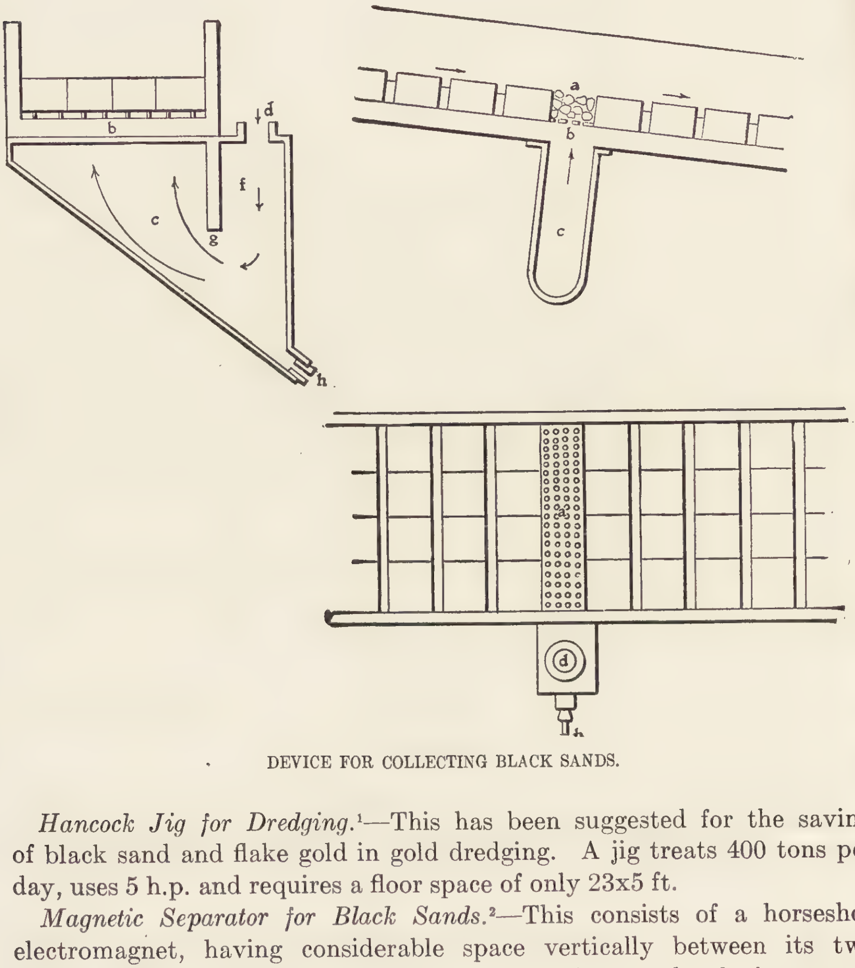
Saving Black Sand.[2]—A very efficient piece of apparatus for this work is the Caribou riffle, which consists of a blanket sluice overlaid with ⅛-in. steel plates perforated with ⅛-in. holes ⅜ in. center to center. These screens are in sections 3 ft. long and the upper end of each screen rests on the blanket while the lower end is elevated about 2 in., thus forming a series of steps. In the bottom of the sluice at the end of each screen section a series of holes of 1 in. diameter and 3 in. apart extends across the sluice. These holes may be partially closed by a slide and thus the amount of material passing through them to an undercurrent may be regulated.

Collecting Black Sand.[3]—Martyn's device is suggested for this purpose. It consists of an endless belt of rakes traveling against the current on the tables of a dredge.

Saving Black Sand.[4]—A device to be placed in the sluices connected with a dredge or hydraulic plant is shown in the figure. It is a hydraulic

[1] J. A. Edman. California State Mining Bureau, *Bulletin* No. 45. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 1047. *Mines and Minerals*, Vol. XXVII (1907), p. 563. *Mining Reporter*, Vol. LV(1907), p. 397.
[2] F. Powell. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 251.
[3] W. Brazenall. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 918.
[4] R. H. Richards. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 1251.

classifier with hydraulic water entering at *d*, spigot discharge at *h*, screen with ¼-in. holes at *b*, and a bed of pebbles corresponding to a jigging bed at *a*.



DEVICE FOR COLLECTING BLACK SANDS.

Hancock Jig for Dredging.[1]—This has been suggested for the saving of black sand and flake gold in gold dredging. A jig treats 400 tons per day, uses 5 h.p. and requires a floor space of only 23x5 ft.

Magnetic Separator for Black Sands.[2]—This consists of a horseshoe electromagnet, having considerable space vertically between its two poles. In this space runs a horizontal endless belt made of wire screen, which becomes magnetized when passing between the poles. If material is fed upon this belt the non-magnetic particles drop through the meshes of the screen into a receptacle below while the magnetic particles cling to the magnetized wires until they are carried out of the magnetic field by the movement of the belt when they drop off into another receptacle.

[1] *Mining Reporter*, Vol. LVI (1907), p. 35.
[2] C. C. Longridge, *Mining Journal.*, Vol. LXXXII (1907), p. 16.

GRAPHITE.

The United States consumes from 30 to 35 per cent. and produces about 20 per cent. of the world's output of graphite. Imports come chiefly from the island of Ceylon where are situated, within an area about 95 miles long and 40 miles wide, the most extensive graphite deposits of the world.

STATISTICS OF GRAPHITE IN THE UNITED STATES.

Year.	Refined Crystalline Graphite.						Amorphous Graphite. Production.		Artificial Graphite. Production.	
	Production.		Imports.		Consumption (*c*)					
	Pounds.	Value (*b*)	Pounds.	Value.	Pounds.	Value.	Tons 2000 lb.	Value.	Pounds.	Value
1897	993,138	$44,691	19,113,920	$ 270,952	20,107,058	$ 315,643	1,200	$11,400	162,382	$10,149
1898	1,647,679	82,385	30,199,680	743,820	31,847,359	826,205	1,200	11,400	185,647	11,603
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953	1,030	8,240	405,870	32,475
1900	4,103,052	164,122	32,298,560	1,389,117	36,401,612	1,553,239	1,045	8,640	860,750	68,860
1901	3,967,612	135,914	32,029,760	895,010	36,997,372	1,067,921	809	31,800	2,500,000	119,000
1902	4,176,824	153,147	40,857,600	1,168,554	45,034,424	1,322,401	4,739	55,964	2,358,828	110,700
1903	4,525,700	164,247	32,012,000	1,207,700	36,537,700	1,371,947	16,591	71,384	2,620,000	178,670
1904	4,357,927	162,332	25,350,000	905,581	29,707,927	1,067,913	19,115	102,925	3,248,000	217,790
1905	4,260,656	170,426	34,914,611	983,034	39,175,267	1,153,460	*d* 21,953	80,639	4,595,500	313,979
1906	4,894,483	170,866	50,974,336	1,554,212	55,868,819	1,725,098	*d* 16,853	(*e*)	4,868,000	312,764
1907	4,586,149	149,548	40,962,000	1,777,389	45,548,149	1,926,937	*d* 26,962	138,381	6,924,000	483,717

(*b*) Nominal. (*c*) Neglecting the small re-exports of foreign product. The exports of graphite from the United States were valued at $834 in 1901, $365 in 1902, $4,220 in 1903, $8,455 in 1904, and $91 in 1905. (*d*) Statistics of U. S. Geological Survey. (*e*) Not reported separately.

Market and Prices.—The graphite industry, especially in the State of New York, is largely controlled by the Joseph Dixon Crucible Company, of Jersey City, N. J. The manufacturing plants of this company consume the product from its mines. Other producers marketed their product at prices varying from 1 to 10c. per ℔., depending on the purity of the material, the average price fetched by the crystalline variety being about 4c. per ℔. Ceylon best pulverized brought from 4 to 8c. per ℔., while German best pulverized was quoted at 1½ to 2c. per pound.

As new uses are found for artificial graphite the demand increases proportionately. The production has shown a steady increase during the last seven years. As is shown by the production table a substantial increase is shown for 1907, the average price fetched by the product being just short of 7c. per pound.

Graphite Mining in the United States.

Alabama.—Extensive deposits of flake graphite occur in this State. Mining operations are being carried on in Clay, Talladega and Alexander

counties, in addition to which several promising prospects in other districts have received attention. In the spring of 1907 the Entiachopeo Graphite Company was organized for the purpose of developing graphite deposits in Talladega and contiguous counties, and a plant for the production of refined graphite was installed. A little later the Iron Mountain Mining Company, of Atlanta, Ga., was also organized to operate in this district. The Redding Graphite Company, of Macon, Ga., the property of which is in Clay county, was incorporated during the summer.

(By Eugene A. Smith.) Graphite is very generally distributed among the metamorphic or crystalline rocks, in which it occurs in two different forms. In the feebly crystalline schists which have been called the Talladega slates, and which in part at least, are Paleozic sediments of as late age as the Coal Measures, the graphite is often found as a sort of black graphitic clay, free from grit. In this condition the graphite is difficult to separate from the other matter with which it is mixed; it is frequently used as a lubricant. Examples of this mode of occurrence are to be seen near Millerville, in Clay county, and about Blue Hill in Tallapoosa county.

In the mica schists and other fully crystalline rocks of this region, the graphite is found in the form of thin crystalline flakes or scales, which may be separated from the inclosing rock without much difficulty. This variety of graphite has been mined at two localities in Chilton county, near Mountain creek, by the Dixie Graphite Company; by the Flaketown Graphite Company, in Coose county; at Fixico, by the Fixico Mining Company; in Clay county, near Ashland, by the Entiachopeo Graphite Company; and at Quenelda, by the Allen Graphite Company. In Coosa county, near Goodwater, are excellent graphite deposits now being developed by Mr. Bunting; also between Mt. Olive and Hollins is another fine property controlled by Dr. B. P. Ivey.

The Alabama ore yields from 3 to 4.75 per cent. graphite and the mills are capable of treating from 1½ to 2½ tons per day, but as there are frequent interruptions it is probable that about four tons per day for the whole State would be a fair estimate of the production at this time. With better appliances this yield could be easily increased.

Michigan.—Early in the summer of 1907 the Detroit Graphite Company resumed operations on its amorphous graphite deposits in Baraga county, at the west extension of the Marquette range. The United States Graphite Company, of Saginaw, mined amorphous graphite which was used for lubricating purposes.

The graphite of Michigan is in a graphitic slate in the Animikie or neo-Huronian. It is nearly all used in preparing graphitic paint and as such slates are widespread it would probably not be difficult to enlarge the output were there sufficient demand.

New Jersey (By Henry B. Kümmel).—Numerous attempts have been made to mine graphite in New Jersey but none of them has been commercially successful, since the deposits are in general lean and scattered. The most recent attempt was at High Bridge, where a graphite-bearing bed of gneiss 30 to 50 ft. wide was opened by a tunnel driven along the strike of the bed for a distance of about 400 ft. The ore-bed is reported to dip 70 deg. westward, with a strike west of north. It is covered by about 6 ft. of soil, mostly disintegrated rock. Pronounced weathering extends to a depth of 30 ft. The ore was reported to contain from 4 to 8 per cent. graphite and the mill test gave about 4 per cent. extraction, but there was a considerable loss in the process. The graphite recovered was divided into four grades. The only ore mined was that taken out in driving the tunnel, and during the seven months that work was continued about 3½ carloads of graphite was obtained and shipped. The plant was closed down in November. The enterprise seems to have been ill-advised in that a costly plant was erected before much development work had been done.

New York (By D. H. Newland).—The year 1907 was uneventful in the crystalline graphite industry of the Adirondacks. As heretofore, the American mine, near Graphite, was the principal source of supply, and it remains the only firmly established enterprise in the region. There were several failures among the newer undertakings, owing to one cause or another. The main obstacle encountered is in refining the product. With the exception of the deposit at Graphite, the quartz schist which carries the graphite frequently contains a small percentage of mica. This mineral is usually developed in scales of about the same size as the graphite flakes, so that it cannot be removed readily by ordinary mechanical separation. Its presence, of course, even in small quantity, limits the uses for the product and brings down the market value as well.

The total production in 1907 amounted to 2,950,000 lb., valued at $106,951. There was thus a small gain in both the quantity and value of the output as compared with 1906. The Crown Point Graphite Company has discontinued operations on the property near Penfield pond, Essex county, and has opened a deposit on Eagle lake. The milling plant has been enlarged. The Glens Falls Graphite Company completed a mill near Conklingville, eight miles west of Hudley, but made no regular production. The Empire Graphite Company began work in its quarries and mill near Greenfield, Saratoga county, in the early part of 1908. In St. Lawrence county some attention was given to a deposit situated on the Indian river, about three miles from Rossie.

North and South Carolina.—Dr. George F. Lee, of Blacksburg, S. C., reported during the summer of 1907, the discovery of a series of graphite deposits extending from Gaffney, S. C., to Grover, N. C. Several grades

of the mineral were discovered ranging in color from deep black to light gray. It is proposed to operate these deposits, power for which may be obtained either from the Southern Power Company, of Charlotte, N. C., or the Electric Manufacturing Company, of Spartansburg, S. C.

Pennsylvania.—The chief producing mines are situated in the vicinity of Chester Springs. The ore is of the crystalline variety and usually occurs in richly impregnated beds of mica schist. In Berks county, near Mertz, the mineral occurs in fine-grained conglomerate or coarse-grained sandstone, both of which have been weathered to a considerable depth.

The Federal Graphite Company, of Chester Springs, did not operate, its properties having been sold to parties who had not resumed operations up to the end of January, 1908. The National Graphite Company, of Anselma, ceased operations on account of financial difficulties. The Parker Graphite Company, of Chester Springs, did not operate its mines. The United States Graphite Company at Byers, ceased operations temporarily at its property. The Philadelphia Graphite Company was merged into the New Philadelphia Graphite Company, then into the Keystone Graphite Company, and later into the Chester Graphite Company which is now operating the properties. The Sterling Graphite Company, of Chester Springs, started its mill during the latter part of October.

Virginia.—The Naylor-Bruce Graphite Company, of Charlottesville, operated only a short time during 1907. However, its plant is now completed and operations will be conducted steadily during 1908. This company's property is situated in Albermarle and Orange counties and was briefly described in The Mineral Industry, Vol. XIV.

Wisconsin (By W. O. Hotchkiss).—An attempt was made in 1907 to develop deposits of carbonaceous pre-Cambrian shales in the central part of this State and a very satisfactory black pigment was made, which is for most practical purposes as good as graphite.

Wyoming (By H. C. Beeler).—Veins of graphite are known at French creek, Plumbago cañon and Halleck cañon, in Albany county, and in the Indian Grove mountains in Carbon county. The veins are large and easily accessible. Analyses of samples from the various localities show the carbon contents to vary from 40 to 60 per cent. So far as known, the ore is of the amorphous variety and would make good fire-proof paint, stove polish or crucibles.

Other States.—During the latter part of 1907 work was resumed on the graphite deposits three miles east of Turret, Chaffee county, Colo. The Crystal Graphite Company, of Dillon, Mont., did not operate steadily throughout the year, although its production exceeded that of 1906. This company's property is situated in Axe cañon, 12 miles east of Dillon. At Black Eagle, Pennington county, S. Dak., a deposit of graphite was opened up late in 1907. It is understood that development work

will be prosecuted. Deposits of graphite were uncovered in the Haystack hills, Laramie county, Wyo. The material is a graphitic schist containing finely disseminated graphite.

Graphite in Foreign Countries.

Austria.—The principal deposits are in Bohemia where the graphite occurs in gneisses, granulites and mica schists; in places crystalline limestone appears. The most important deposit is at Schwarzbach. Here the graphite occurs in three layers, the upper and lower beds consisting of a compact, impure schistose material. The middle bed contains a soft, earthy variety of great purity which is marketed with no other treatment than the removal of a silicious concretion, which occurs sparingly. Graphite is also mined in Styria in the Rottenmanner Tauern, an eastern spur of the Niedern Tauern. Here the mineral is found in beds, as impregnations in chloritoid schists, alternating with true phyllites. This district produces from 25 to 30 per cent. of the Austrian output, the material being employed for covering the interior of molds and in the manufacture of crucibles for melting steel. Of equal importance are the deposits in Moravia in the neighborhood of Alstadt and Goldenstein. Here the mineral occurs in seams in crystalline limestone. Graphite is also found in lower Austria but the deposits have not been exploited to any great extent.

Canada.—Although the production of graphite seems small when compared with the output of other countries, the mineral is widely distributed and the industry is fast becoming of importance. It is found in no less than five provinces, viz., Nova Scotia, New Brunswick, Quebec, Ontario and British Columbia.

It may be stated broadly that the deposits of Quebec are confined to rocks of pre-Cambrian age. The principal mines of the province are those of the Anglo-Canadian Graphite Syndicate, the Buckingham Company, the Buckingham Graphite Company, the Diamond Graphite Company, the Bell mine, in the vicinity of Buckingham, and those of the Calumet Graphite Company, in the Grenville district. There are also some promising prospects. The Diamond Graphite Company is composed of American capitalists and has recently completed a 100-ton concentrating plant, while another mill of the same capacity is in course of construction.

In Ontario the only important mines are those at North Elmsley and the Black Donald mine, owned by Rinaldo McConnel, of Ottawa. Both of these mines are equipped with mills, the former using the dry process and the latter the wet.

The existence of graphite in New Brunswick has been known since 1839, the deposits being near St. Stevens, but no work of any importance has ever been done. An important deposit exists near the city of St. John.

Graphite is widely distributed over Nova Scotia, the principal deposits being on Cape Breton island. Exploitation so far has been insignificant.

Ceylon.—The deposits of Ceylon have been more extensively developed than those of any other country and the quality of the graphite mined is the best. The deposits of commercial importance form beds, veins, or nests in granulitic rocks. As a rule the graphite has a flaky, columnar structure, pure lamellar graphite being rare. Mining is generally conducted through shafts sunk to a depth of 50 or 60 ft., the ore being hoisted in barrels. After rough picking in the dressing shed it is packed in barrels and shipped to Colombo or Galle where it is re-sorted and screened, the larger pieces being broken up. In some cases the leaner material is further concentrated by washing. Ceylon graphite is classified as "large lump," "ordinary lump," "chips," "dust" and "flying dust." A comparatively new feature of the industry in Ceylon is the exportation of a considerable amount of graphite to Japan, that country having taken 1400 tons during the first three months of 1907.

Germany.—The center of the industry in this country is at Pressnitz in Bavaria. Here the graphite occurs in lenticular masses of highly decomposed micacous rocks, near the contact of the granite and gneiss. Other granitic rocks are also found in connection with the deposits. The graphite is flaky, which permits of its easy separation; thus the pyrites is eliminated, leaving only a small quantity of mica as an impurity.

Other Countries.—The industry has not grown to any great proportions in Africa although deposits, some of which are now being worked, are known to exist at Prince Albert and Calvinia, Cape Colony; in the Impetyini forest, Natal; in Rhodesia and British Central Africa, British East Africa, Uganda, and northern Nigeria. Graphite is known in all of the States of the Commonwealth of Australia but nowhere has the industry attained any great importance. An extensive deposit exists within four miles of Netherby, a station on the North Coast Line, Queensland. A mine was opened there three years ago and shipments of graphite, said to have been satisfactory to the trade, were made. However, the mine is not being worked now. It is reported that promising deposits of graphite of steely and metallic appearance have been discovered on the banks of the Gamboa river, in the department of Castro, in the eastern part of Chiloe island, Chile. Graphite is found in many localities in India. The quality of the mineral is not of the best, however, and the output amounts to only three to four thousand tons yearly. Graphite is mined in Italy near Pinerolo in the Cottian Alps, and near Bagnasco in the Bormida valley. The deposits are similar to those of Styria and the production amounts to about 10,000 tons yearly. Unimportant deposits are to be found in Siberia, Mexico and different parts of South America.

WORLD'S PRODUCTION OF GRAPHITE.

(In metric tons.)

Year	Austria	Canada	Ceylon (*d*)	Germany	India	Italy	Japan	Mexico (*f*)	Sweden	United States (*b*)	Totals.
1897	38,504	395	19,275	3,861	61	5,650	204	759	99	450	69,163
1898	33,062	1,107	78,509	4,593	61	346	346	1,857	50	824	125,006
1899	31,819	1,188	29,037	5,196	1,548	9,990	55	2,305	35	1,648	80,962
1900	33,663	1,743	19,168	9,248	1,858	9,720	942	2,561	84	1,799	81,938
1901	29,992	2,004	22,707	4,435	2,530	10,313	88	762	(*e*)56	1,800	74,688
1902	29,527	993	25,593	5,023	4,648	9,210	97	1,434	63	1,895	78,371
1903	29,590	660	24,492	3,720	3,448	7,920	114	1,404	25	2,053	73,436
1904	28,620	410	26,478	3,784	3,800	9,765	216	970	55	2,045	76,143
1905	34,416	491	31,134	4,921	2,324	10,572	209	970	40	1,933	87,010
1906	38,117	405	36,578	4,055	2,600	10,805	177	3,915	(*c*)	2,220	98,872
1907	(*c*)	525	(*c*)	4,033	(*c*)	(*c*)	72	(*c*)	(*c*)	2,080	(*c*)

(*a*) Not reported in the government statistics. (*b*) Crystalline graphite. (*c*) Statistics not yet available. (*d*) The figures for 1897 and 1899 are exports; the enormous production in 1898 as reported in official government publications is not reflected in the exports for that year, which amounted to 24,349 metric tons. (*e*) The production of crude graphite in 1901 was 1727 tons. (*f*) Exports.

Occurrence, Classification and Uses of Graphite.

The largest and most permanent deposits of graphite occur generally in the oldest crystalline rocks, although it is often found in sedimentary strata, in places in connection with coal, forming a kind of graphitic anthracite. In a number of places graphite also occurs in association with beds of altered clay slates and shales of more recent date than the crystalline rocks. All graphite deposits may be divided into three classes, viz., (1) Veins or veinlets; (2) beds or masses; (3) disseminations through country rock. Commercial graphite is known as either *crystalline*, *amorphous*, or *artificial.*

Crystalline and Amorphous.—The distinction between crystalline and amorphous graphite is not well defined. In general, that variety which posseses a lamellar, scaly or flaky structure and which is composed of almost pure carbon, is classed as crystalline; all other forms of whatever occurrence or character are referred to as amorphous.

Crystalline graphite is used chiefly in the manufacture of crucibles and other refractories, pencils, lubricants and dynamo brushes and in electrotyping. Amorphous graphite is more abundant in nature, but owing to the fact that many of the deposits contain gangue material which it is impossible to remove economically, it is not fit for purposes requiring a pure graphite. It is now used chiefly in the manufacture of foundry facings, stove polish and certain varieties of paint.

Apart from the manufacture of refractory materials and pencils, the use of graphite is in its infancy and the future holds forth many possibilities.

Artificial Graphite.—This form of graphite is a product of the electric furnace. The chief ingredient of the charge is anthracite coal or petroleum coke, in connection with which glass sand, foundry coke, and sawdust are used. The furnaces have the form of a long narrow trough, built

with fire brick and lined with a mixture of sand, coke and sawdust. At the end of each trough is a terminal of carbon rods, to which is connected the cables conveying the current. The trough is filled with anthracite coal, in which is imbedded a carbon rod to make electrical connection between the terminals, as the coal is a poor conductor of electricity. The temperature to which the coal is raised before conversion into graphite is said to approximate 7500 deg. F.

Artificial graphite is used in the manufacture of electrodes, dry battery fillers, and pigment; it is also employed in electrotyping and lubrication. In the last mentioned capacity a compound known as deflocculated graphite was recently put on the market. This is produced by adding water, gallotannic acid and ammonia to unctuous graphite as produced in the electrical furnace. Under these conditions the graphite is miscible with the water and assumes what is called a deflocculated condition, a condition of fineness beyond that obtainable by mechanical means—one approaching the molecular state. Deflocculated graphite in water has been used successfully instead of oil in drop-feed oilers and with chain-feed oilers. It possesses the remarkable property of preventing rust or corrosion of iron or steel. The deflocculated graphite has also been successfully employed with kerosene oil as a lubricant.

The International Acheson Graphite Company, of Niagara Falls, N. Y., is the sole manufacturer of artificial graphite, which is known to the trade as Acheson-graphite.

Some Characteristics of Natural Graphite.

By Frederic S. Hyde.

Ceylon graphite excels in luster, refractory quality, and lubricating and polishing properties; other forms of graphite rarely approach the same degree of excellence. A sample of graphite may possess the proper luster and unctuous properties and yet be lacking in other respects; it may be too hard and crystalline for pencils, too micaceous for crucibles, or too granular and earthy for lubrication. Again, the material may have excellent lubricating properties and not be refractory enough for the manufacture of "steel-pots."

The physical peculiarities, such as the refractory quality, the fusibility of ash, the structure and the property of binding with other materials, are equally as important as the percentage of carbon. The Ceylon plumbago will generally mix better with a given clay and will form a stiffer "batch" with better binding properties than some American flake-graphites, which (on account of peculiarities in structure) refuse to bind; hence Ceylon plumbago is generally used for crucibles, retorts, etc.

Properties and Requisites.—As a rule graphites containing 90 to 95 per

cent. graphitic carbon are sufficiently pure to meet the requirements of the general trade. Natural lubricating graphite, sold in dry or flaky form, rarely contains more than 90 per cent. Physical, rather than chemical, properties seem to govern the choice of graphite, and an 85 per cent. Ceylon plumbago may prove superior to a 95 per cent. artificial product so far as general application and selling qualities are concerned. For some purposes, such as for use in foundry facings, the presence of silicious material may act as a positive benefit by causing the graphite to cling or spread better on the surface of the mold.

To distinguish one variety of graphite from another, when both have been subjected to a milling process or reduced to the same degree of fineness, the material should be examined microscopically and comparisons made with known standards. Graphite like certain other minerals possesses an odor more noticeable perhaps when it is moistened or mixed with plastic material. Odor or the tasting method cannot, however, be considered as conclusive tests. The color of the ash obtained after oxidation over a blast-lamp, together with the physical structure, may serve as a further guide in reaching a definite conclusion.

Characteristics of Various Graphites.—Ceylon lump, when ground to flake, leaves on oxidation a granular ash-residue which is brick-red to dark brown in color. About one-half of this residue is silica, the rest consisting of iron, alumina, and alkalies in combination with the silica. Small amounts of magnesia (0.1 to 0.2 per cent.) are generally present. Sulphur in combination with iron, as pyrites, will average about 0.2 per cent. in the original lump form. Alaskan and Norwegian lump may bear a very close resemblance to the poorer grades of Ceylon. The color of the ash may be identical in each case but the Alaskan, as a rule, is likely to be more gritty and poorer in graphite, while the Norwegian may be full of free sulphur in a finely divided condition, although readily ignited in certain cases. In Ticonderoga graphite magnesia is noticeable as a constituent of the ash, in addition to which it carries a large percentage of combined alkalies which tend to increase the fusibility of the material as soon as the surrounding graphite becomes oxidized or "burnt out."

Bad Effects of Sulphur.—In the assay of graphite allowances should always be made for moisture, combustible organic matter and sulphur. Sulphur is an undesirable element in plumbago pots, used for melting silver or high-grade alloys. In fact, sulphur in the form of pyrites should always be eliminated as far as possible in the milling process. It is not unusual to find red spots on kiln-burnt pots, due to the iron of the pyrite. Small plumbago crucibles when subjected to a clean blast in a gas furnace at a yellow heat often exhibit fused pittings due to fluxing of iron oxide with mineral matter; this is generally the case when the plumbago is known to contain pyrites even in very small amounts.

GYPSUM.

By A. Van Zwaluwenburg.

The production of gypsum in the United States has been increasing for a number of years. Part of the increased consumption is due to the normal industrial growth of the country, but also in large measure it is due to the development of new uses for the mineral and its products. The increasing tonnage of calcined gypsum employed in stucco and wall plasters is not accounted for entirely by the growth of the building industry. The cement industry also consumes large quantities of calcined plaster which is used as a retarder. In spite of the financial depression during the latter part of 1907, the production of gypsum in 1907 showed a substantial increase over that of 1906.

In 1906 gypsum was produced in 17 States and Territories besides Alaska, and during 1907 several companies were formed for developing deposits in other localities. Owing to the abundance and the low cost of the mineral, most mining operations are carried on in connection

STATISTICS OF GYPSUM IN THE UNITED STATES.

(In tons of 2240 lb.)

Year.	Production.		Imports.					
			Crude.		Ground or Calcined.		Plaster of Paris.	Total Value of Imports.
	Quantity. (b)	Value. (c)	Quantity.	Value.	Quantity.	Value.	Value.	
1896	195,553	$583,136	180,269	$193,544	3,292	$21,982	$11,722	$ 227,248
1897	268,187	889,177	163,201	178,686	2,664	17,028	16,715	212,429
1898	281,130	864,415	166,066	181,364	2,973	18,501	40,979	240,844
1899	376,840	1,155,581	196,579	220,603	3,265	19,250	58,073	297,926
1900	432,323	1,316,255	209,881	229,878	3,109	19,179	66,473	315,530
1901	588,981	1,577,493	235,204	238,440	3,106	19,627	68,603	326,670
1902	(a) 728,998	2,089,341	305,367	284,942	3,647	23,225	52,533	360,700
1903	(a) 930,093	3,792,943	265,958	301,379	3,526	22,784	54,434	378,597
1904	(a) 840,104	2,784,325	294,238	321,306	3,278	11,276	23,819	356,401
1905	(a) 931 475	(d) 821,967	356,457	402,378	3,471	20,883	22,959	446,220
1906	(a) 1,375,588	3,837,975	390,178	464,724	3,203	22,821	21,297	508,842
1907	(a) 1,564,061	4,942,264	405,278	486,205	1,767	12,825	(e) 38 020	537,050

(a) Statistics of the U. S. Geological Survey. (b) Represents the amount of crude gypsum quarried. (c) Represents the value of the marketed gypsum, including its various finished forms. (d) Value of crude material. (e) Includes $1,392 in gypsum manufactures.

with mills and plants for grinding and burning the product. Michigan is still the chief producer, New York following, with Iowa a close third.

Among the other producing States are: Texas, Ohio, Oklahoma, Kansas, California, Wyoming, Virginia, Nevada, Oregon, Utah, New Mexico, Colorado, South Dakota and Montana.

PRODUCTION OF CRUDE GYPSUM IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1904. (*a*)		1905. (*a*)		1906. (*a*)	
	Tons.	Value. (*b*)	Tons.	Value. (*h*)	Tons.	Value. (*b*)
Cal. Ohio and Va.	101,809	$318,723	147,136	$188,974	(*f*)202,376	$536,940
Colo. and Wyo.	35,778	135,045	26,880	26,930	(*g*)	....
Iowa, Kan. and Tex.	(*c*) 319,080	1,027,792	(*d*) 375,239	296,247	(*d*)639,885	1,546,188
Michigan	238,385	541,197	299,585	143,597	341,716	753,878
New York	158,892	432,358	153,367	151,272	288,631	749,896
Oklahoma	53,523	190,245	(*e*)	....	(*e*)	....
Other States	33,450	138,965	40,995	14,947	67,977	251,073
Total	940,917	$2,784,325	1,043,202	$821,967	1,540,585	$3,837,975

(*a*) Statistics of the U. S. Geological Survey. (*b*) Value includes that of prepared products. (*c*) Of which, Iowa, 145,359 tons—$475,432. (*d*) Includes Oklahoma. (*e*) Included with Iowa, Kansas and Texas. (*f*) Includes Nevada and Oregon. (*g*) Included in other States. (*h*) Value as mined.

PRODUCTION OF GYPSUM IN THE PRINCIPAL COUNTRIES. (*a*)
(In metric tons.)

Year.	Algeria. (*b*)	Canada.	France. (*b*)	Germany. (*c*)		Greece.	India.	United Kingdom.	United States.
				Baden.	Bavaria.				
1896	37,512	187,778	2,051,124	32,801	28,799	120	7,605	196,404	201,305
1897	36,750	217,340	2,004,339	40,702	26,153	51	8,187	184,287	272,493
1898	37,337	198,864	2,115,261	28,037	25,688	83	8,390	199,174	285,644
1899	39,950	221,821	1,807,454	29,419	29,727	81	6,546	215,974	382,891
1900	42,237	228,656	1,774,492	26,381	35,484	129	4,415	211,436	439,265
1901	44,025	266,476	2,385,633	28,183	3,581	671	(*d*)	204,045	598,529
1902	44,975	301,165	2,185,346	33,150	31,701	*Nil*	(*d*)	228,264	740,906
1903	41,550	285,242	1,998,804	29,423	30,894	94	(*d*)	223,426	945,285
1904	48,375	309,133	1,957,802	26,984	22,766	393	3,937	237,749	853,546
1905	34,743	395,341	1,378,145	28,823	46,247	185	4,877	259,596	982,626
1906	27,950	378,904	1,377,429	25,643	50,763	70	(*d*)	228,627	1,397,480
1907	(*d*)	431,286	(*d*)	29,153	(*d*)	(*d*)	(*d*)	247,537	1,564,061

(*a*) From official reports of the respective countries, except the statistics for the United States. (*b*) A part of the product is reported as plaster of paris. In converting this to crude gypsum it has been assumed that the loss by calcination is 20 per cent. (*c*) Prussia is a large producer of gypsum, but there are no complete statistics available. (*d*) Statistics not yet available.

Market Conditions and Prices.—The market for gypsum and its products was fairly active during 1907. Although the quoted prices varied little during the year, the output sold in large and small lots fetched considerably more than that of 1906. The average price of all gypsum products of New York State estimated upon the total production was $3.19 per short ton in 1907, as compared with $2.70 in 1906.

Gypsum in the United States.

Iowa.—At Fort Dodge, 11 mills treated crude gypsum. The United States Gypsum Company operated seven mills; the Cardiff Gypsum and Plaster Company, one; the Iowa Hard Plaster Company, one; the

American Independent Gypsum Company, one; and the Plymouth Gypsum Company, one.

Michigan (By Leroy A. Palmer).—Michigan furnishes about twice as much gypsum as any other State, and most of this comes from one small district in the southwestern part of Kent county, adjoining and included in the city of Grand Rapids. Here an area of approximately six square miles is underlaid with an average thickness of 10 ft. of almost pure gypsum. The gypsum industry in this part of Michigan dates back to 1837, when the rock was ground in corn mills and sold for fertilizer.

The present process of treatment is practically the same at all mills. The rock as brought from the mine is dumped into a jaw crusher, from which it goes to the crackers, a modification of the corn mills used in the early days of the industry, which discharge their product, the size of fine gravel, to buhr stones. The kettles are upright hollow cylinders of boiler steel of about the same depth as hight, having a convex bottom, the most improved types having flues either direct or both direct and return. Just above the bottom of the kettle are the arms of the mixer which is turned by a vertical shaft. Calcining is the most delicate part of the progress and calls for the most skilful labor. The expert calciner judges of the progress of the operation by the appearance of the boiling mass, the amount of steam given off and the creaking of the machinery; the evaporation makes the mixture more dense, thus throwing greater strain on the gears.

Montana (By J. P. Rowe).—The gypsum of this State occurs in two general fields. The deposits of Cascade and Fergus counties belong to the interior field; while those of Carbon county, especially the southern part, belong to the southern field. There are reports from Jefferson county to the effect that large deposits of gypsite have been found in the southwestern part of the county, near Whitehall. Two large deposits are found in the interior field, one near the towns of Armington and Kibby, Cascade county, and the other in the Big Snowy mountains, Fergus county, near Portugese.

The selenite variety of gypsum is found in all the counties of Montana east of the Rockies. It occurs generally in the upper Cretaceous formations, but so far has not been found in commercial quantities. Very commonly it impregnates the waters both of streams and springs, making them unfit for use. At Hunter's Hot Springs, on the north bank of the Yellowstone river, about 20 miles east of Livingston, the hot waters are now depositing gypsum and the old hot-spring fissures are filled by a mass of gypsum and stilbite. Up to the present time these deposits, although of considerable extent, have not been utilized. Crystals of selenite are found in large quantities in the Laramie clays, near Glendive, Dawson county; near Wibaux, Dawson county; near the Bear Paw

mountains, Chouteau county; and near Bowler, Carbon county. The last is not in the Laramie, but probably is Triassic or Jurassic.

The principal gypsum beds now being worked in the interior field are situated in the northwestern part of Cascade county and cover a large area. The beds are between 25 and 20 ft. thick, and extend over an immense area. They are nearly horizontal, having a slight dip to the northwest. The gypsum itself is pure but is interstratified somewhat, in places, with considerable clay. The productive part of the southern field is situated wholly within Carbon county, and this series of beds extends southward into the Big Horn basin and can be traced into the mountains of Wyoming. By far the most promising and largest bed of this field, and the thickest, if not the best bed in the State, is situated about 16 miles south and east of Bridger.

Nevada.—The Arden Plaster Company in 1907 completed a mill for the manufacture of gypsum products 10 miles south of Las Vegas, on the Salt Lake railroad, representing a total investment of $150,000. A narrow-gage railroad, five miles long, carries the rock to the mill. The mill has a capacity of 200 tons per day and is equipped with a 300-h.p. engine, crushers, mills, kettles, mixers, and separators.

New Mexico.—Extensive deposits of gypsum occur at many places in New Mexico, particularly in the southeastern and northwestern parts of the Territory, but they are developed only at Ancho, on the line of the Rock Island railroad, where a plaster mill has been in operation for several years. The gypsum along the western base of the Sierra Nacimiento has been known for more than half a century. The deposits along the western base of the range lie near the top of the "Red Beds" series. At Gallina, on Gallina creek, near the northern limit of the gypsum outcrop, a bed of massive white gypsum appears within the limits of the village, where it has been, to a very limited extent, quarried and burned. The gypsum bed may be traced eastward from Gallina for many miles. Farther south between Gallina and Senorita, the same gypsum bed was observed at many places. East of Lajara the bed, if present, is completely covered by flat-lying Tertiary sediments.

New York (By D. H. Newland).—An aggregate of 323,323 short tons of gypsum were taken from the mines and quarries of the State in 1907, as compared with 262,486 tons in 1906. The output has increased by over 100 per cent. within the last three years, due to the rapid development of the trade in wall plasters, stucco, etc., and to the use of gypsum in portland cement manufacture. The value of the different materials was $1,038,355, as compared with $699,455 in 1906.

Oklahoma (By Charles N. Gould).—The amount of gypsum in Oklahoma is practically inexhaustible. With perhaps two exceptions, every county west of the main line of the Rock Island Railroad contains enough material

to supply the United States indefinitely. All the deposits may be roughly grouped under four general regions, as follows: (1) the Kay county region, occupying the central part of Kay county; (2) the main line of Gypsum hills, extending from Canadian county northwest through Kingfisher, Blaine, Major, Woods, Woodward and Harper counties, to the Kansas line; (3) the second line of Gypsum hills, extending along a line parallel with the main range and from 50 to 75 miles farther southeast—this line of hills begins near Marlow in northern Stephens county and extends northwest through Caddo, Washita, Custer, Dewey, and Roger Mills into Woodward and Ellis counties; (4) the Greer county region, occupying the greater part of western Greer county, as well as southern Beckham and western Jackson counties.

It is estimated that the amount of gypsum available in Oklahoma is 125,000,000,000 tons, considering no deposits at a greater depth than 100 ft. beneath the surface.

There are at present eight gypsum mills in operation in western Oklahoma, at the following points: Okarche, Cement, Marlow, Watonga, Bickford, Ferguson, Southard and Alva; there is also one mill at McAlester, in the eastern part of the State. The capacity of the various mills ranges from 40 to 100 tons per day. In the majority of cases the material used is gypsite, or dirt gypsum. Rock gypsum seems to be in bad repute among practical plaster men in Oklahoma because of the extra cost of crushing the rock. Coal is the fuel used, the greater part coming from the McAlester region, 200 to 250 miles distant. The price of coal at the mills ranges from $4@6.50 per ton. The market is largely to the east, much of the product going direct to Kansas City and Memphis. For several years the demand has exceeded the supply.

There are two problems to be solved in connection with the gypsum plaster industry of Oklahoma, namely, available gypsite deposits and cheaper fuel. The plaster men believe that the greater part of the gypsite has been located and that the supply will soon be exhausted. There is a geological reason for believing, however, that there are vast undiscovered deposits of gypsite in each of a half dozen western counties, and that all that is needed is systematic prospecting. It is probable that, as a conservative estimate, not 10 per cent. of the available gypsum deposits have yet come to light. The fuel problem is more difficult. It is useless to look for coal anywhere in the gypsum region, and the geological structure precludes the probability of petroleum or natural gas being found in quantity. The nearest coal is 200 miles or more from the gypsum, and under existing conditions the railroads get the freight on the coal and the finished products as well. A milling and transit rate to the gas fields in eastern Oklahoma has been suggested as the solution of the difficulty. The last resort is to pipe the natural gas to the gypsum. Gas mains are

already laid as far as Oklahoma City and they will probably soon be extended to El Reno, which is not more than 15 miles from the southern end of the Gypsum hills.

Utah.—Alex. Jennings organized a company to work deposits of gypsum in Loss Creek cañon, Sevier county, south of Selina.

Wyoming (By H. C. Beeler).—The plaster production for Wyoming in 1907 is estimated at 40,000 tons, 30,000 tons being made in the mills of the American Cement Plaster Company (of Lawrence, Kan.) at Laramie, in Albany county, Wyo., the remainder coming from other plants. A number of new plants are stated to be in contemplation for erection during 1908.

Gypsum in Foreign Countries.

Nova Scotia.—The amount of gypsum quarried in 1907 was 332,345 gross tons, a large increase as compared with the output of 1906. Practically the whole output was shipped to the United States. The report of the department of mines names nine producers of gypsum in the Province. The Victoria Gypsum Mining and Manufacturing Company, Ltd., operating at St. Ann's, 10 miles from Baddeck, Cape Breton Island, shipped 40,000 tons of gypsum rock during the season to Philadelphia. The mineral is all extracted by quarrying, and the quarries are connected by three miles of railway with St. Ann's Harbor. There are 25 ore cars of six tons capacity each, and the trains are hauled back and forth by a locomotive. The pier is 35 ft. above tide-water, and the rock is discharged from a tipple through a chute into the hold of a vessel at the rate of 1000 tons a day.

IODINE.

At present the entire production of iodine is obtained from two sources, viz., certain varieties of algae (*Cuminaria*, *Fucus* and *Zastera*), the collecting and preparing of which is an important industry of Japan, Norway, Scotland, Ireland and France; and from the sodium nitrate deposits of Chile. Other possible sources, which have been the subject of considerable investigation, are the mother liquors from brine wells, the water brought up with the petroleum from oil wells, and certain mineral springs. A few Silesian zinc ores and the phosphate rock deposits in several parts of France contain small proportions of iodine, which it has been proposed to utilize. The rare iodide minerals of silver, mercury and lead may also be mentioned. The algae provide the purest iodine, but the output from the Chilean nitrate deposits is the largest; only the product from mineral sources will be referred to in the following paragraphs.

AVERAGE PRICE OF IODINE PER POUND.

Year.	London.	New York.	Year.	London.	New York.
1898....	10s. 0d.	$2.37	1903.....	8s 0d.	$1.90
1899....	10 0	2.37	1904.....	10 2	2.41
1900....	9 6	2.25	1905.....	11 2¾	2.66
1901....	8 1½	1.93	1906.....	8 0	1.90
1902....	8 0	1.90	1907.....	8 0	1.90

EXPORTS OF IODINE.
(In metric tons.)

Year.	Chile.	Germany.	Norway.	Year.	Chile.	Germany.	Japan.	Norway.
1895............	144	....	3	1901............	385	27	....	10
1896............	206	26	2	1902............	244	24	2	....
1897............	243	26	2	1903............	387	29	14	11
1898............	235	26	5	1904............	461	30	31	9
1899............	304	26	16	1905............	564	27	26	12
1900............	318	29	11	1906............	....	46	11	13

Technology.

Occurrence and Treatment in Chile.—Iodine occurs in the sodium nitrate deposits in the form of the calcium iodate, lautarite, $Ca(IO_3)_2$, and as a double salt of calcium iodate and chromate of the composition $Ca(IO_3)_2 + 8CaCrO_4$. During the treatment of the nitrate, the iodine salts are

decomposed and finally appear in the mother liquor in the form of sodium iodate, $NaIO_3$. This mother liquor has about the following composition: Sodium nitrate, 28 per cent.; sodium chloride, 11; sodium sulphate, 3; magnesium sulphate, 3; sodium iodate, 22; water, 33. This liquor is mixed, in lead-lined wooden tanks, with a solution of mono- and disodium sulphites, in the proportion of one of the mono- to two of the double salt, whereupon the following reactions occur:

$$2NaIO_3+3Na_2SO_3+2NaHSO_3=5Na_2SO_4+I_2+H_2O$$
$$NaIO_3+5NaI+3H_2SO_4=3Na_2SO_4+3I_2+3H_2O$$

The washed iodine is then partially freed from water in a filter press and is then pressed into molds of about 20 cm. diameter. The crude iodine contains 65 to 70 per cent. iodine, the remainder consisting of silica, calcium sulphate and water. It is next sublimed in cylindrical iron retorts of about 1000-℔. capacity, which are connected to clay condensers. These operations last four or five days; the iodine is warranted to be 98 per cent. pure.

Extraction from Oil Well Water.—The water which often accompanies a flow of petroleum generally contains a great amount of salts, of which calcium iodide is one. The water from this source in Pennsylvania carries 0.5587 gram of calcium iodide per liter. That from Baku carries only a few centigrams per liter. That from Berekei, in the neighborhood of Derbent, in Russia, carries more. The extraction of iodine from these waters on a commercial scale is thought to be possible.

Electrolytic Production of Iodine.—A great number of processes have been invented for the electrolytic separation of iodine from its alkaline compounds, notably those of Steinberg, Gladstone and Tribe, Parker and Robinson, Rinck, and Engelhardt, but thus far none of them has proved commercially successful. The last mentioned investigator has reduced the cost of producing one kilogram of iodine by the electrolytic method to $3.73, as against $3 by the usual chemical process.

Testing Commercial Iodine.—The crude iodine in the market ranges from 75 to 90 per cent. iodine, the impurities being mainly chlorine and, particularly in the French iodine, considerable amounts of water, up to 20 per cent. or over. Most of these impurities can be removed by sublimation, but not so with the cyanide of iodine which occasionally is present in large amounts. This impurity can be removed by treating the crude iodine with iron, water and potassium carbonate, whereby the cyanide is precipitated with the iron.

To determine the water in an impure iodine, pulverize with mercury and a little alcohol and then dry at a slight heat; the loss is water. To determine the percentage of iodine, a weighed portion is dissolved in a

somewhat concentrated solution of ammonium or sodium sulphate; the solution is filtered and precipitated with silver nitrate. The precipitate is then washed free from silver chloride and bromide with ammonia, and is freed from silver sulphate by treatment with very dilute nitric acid. From the amount of the remaining pure iodide of silver the percentage of iodine in the original sample can be computed. To determine ash, a portion is ignited in a porcelain crucible and the residue is weighed.

IRON AND STEEL.

By Frederick Hobart.

The general prosperity enjoyed by the iron and steel industries during the greater part of 1907 was so marked that the sharp break in October could not offset it, but left the total volume of business above even the record made in 1906, and far above that of any previous year.

The production of iron ore showed a substantial increase, but the end of the year found large stocks on hand, owing to the closing of many of the furnaces. Pig-iron production increased by nearly 2 per cent. while bessemer and open hearth steel decreased by less than 0.2 per cent.

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.
(In tons of 2240 lb.)

District.	1901	1902	1903	1904	1905	1906	1907
Lake Superior	20,589,237	27,571,121	24,099,550	21,822,839	34,353,456	38,522,129	42,245,070
Southern States	4,767,667	4,850,000	5,889,000	5,450,000	7,175,000	7,450,000	7,585,000
Other States	2,530,575	2,215,000	2,483,000	2,190,000	3,050,000	3,265,000	3,125,000
Total	27,887,479	34,636,121	32,471,550	29,462,839	44,578,456	49,237,129	52,955,070
Add decrease in stocks	45,007	....	703,169	....	....	....	....
Add imports	966,950	1,165,470	980,440	487,613	845,651	1,060,390	1,229,168
Total	28,899,436	35,801,591	34,155,159	29,950,452	45,424,107	50,297,519	54,284,238
Increase in stocks	....	1,214,591	....	....	....	....	278,208
Deduct exports	64,703	88,445	80,611	213,865	208,058	265,240	3,750,000
Total consumption	28,834,733	34,499,555	34,074,548	29,736,587	45,216,049	50,032,279	50,256,030

Iron Ore.

The total production of iron ore in the United States in 1907 was 52,955,070 tons as compared with 49,237,129 tons in 1906. The closing down of a great many furnaces during the last two months of 1907, however, left a large quantity of ore in stock, and this will doubtless have a marked effect on the production of 1908. The Lake Superior district ranked first with a production of 42,245,070 tons. The iron mining industry in the several States is reviewed in another part of this article.

Pig Iron.

The pig-iron production of the United States in 1907, as compiled by the American Iron and Steel Association, was 25,781,361 long tons,

showing an increase of 1.8 per cent. over the previous year. Had it not been for the sharp curtailment of production in November and December, the total for 1907 would have run up to about 27,250,000 tons. The average make for the 10 months ending with October was 2,267,000 tons per month. For the whole year, the average was 2,148,447 tons.

PIG IRON PRODUCTION OF THE UNITED STATES.
(In tons 2240 lb.)

Kind of Iron.	1902	1903	1904	1905	1906	1907
Foundry and forge	5,176,568	5,281,200	4,358,295	5,837,174	5,709,350	6,397,777
Bessemer pig	10,393,168	9,989,908	9,098,659	12,407,116	13,840,518	13,231,620
Basic pig	2,038,590	2,040,726	2,483,104	4,105,179	5,018,674	5,375,219
Charcoal	....	504,757	337,529	352,928	433,007	437,397
Spiegel and ferro	212,981	192,661	219,446	289,983	300,500	339,348
Totals	17,821,307	18,009,252	16,497,380	22,992,380	25,302,049	25,781,361

PIG IRON PRODUCTION ACCORDING TO THE FUEL USED.
(In tons of 2240 lb.)

Fuel used.	1901	1902	1903	1904	1905	1906	1907
Coke (a)	13,782,386	16,315,891	15,592,221	14,931,364	20,964,937	23,313,498	23,972,410
Anthracite and coke	1,668,808	1,096,040	} 1,911,347	1,228,140	1,674,515	{ 1,535,614	1,335,286
Anthracite alone	43,719	19,207				{ 25,072	36,268
Charcoal	390,147	378,504	504,757	} 337,529	352,928	433,007	{ (b) 437,397
Charcoal and coke	23,294	11,665	927				{ ....
Totals	15,878,354	17,821,307	18,009,252	16,497,033	22,992,380	25,307,191	25,781,361

(a) Under coke furnaces are included the very few which use raw bituminous coal. It may be assumed that 99 per cent. of this class of iron was made with coke. (b) Includes a small quantity made by the electric furnace in California.

The considerable decrease in bessemer pig was due to the fact that the greater part of that class of iron is made by the furnaces owned by the large steel companies; and it was those furnaces which were the first to go out of blast, the merchant furnaces generally holding on as long as possible. Many of the latter had contracts which they had to fill; though canceling of orders was quite a common feature of the closing of the year.

PRODUCTION OF PIG IRON BY DISTRICTS.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907
New England, N. Y. & New Jersey	782,350	880,074	1,525,094	1,952,288	2,052,060
Pennsylvania	8,211,500	7,644,321	10,579,127	11,247,869	11,348,549
Ohio, Illinois, Michigan, Wisconsin and Minnesota	5,508,034	5,077,549	7,260,712	8,226,778	8,457,045
Maryland	324,570	293,441	332,096	386,709	411,833
Southern States	2,912,509	2,449,872	2,887,577	3,080,507	3,033,388
West of the Mississippi	270,289	151,776	407,774	413,040	468,486
Total	18,009,252	16,497,033	22,992,380	25,307,191	25,781,361

The classification by districts, as made in an accompanying table,

is based chiefly on ores used. It is not strictly correct, as it is impossible to make a strict division; but the errors probably balance very nearly. Thus the New York furnaces which use Lake ores will about offset the Lehigh Valley and other stacks in eastern Pennsylvania which use local ores. Maryland is placed by itself, as its furnaces use Cuban and other foreign ores entirely. The table shows that approximately 76.8 per cent. of the pig iron last year was made in the districts which depend on the mines of the Lake Superior region for their ore supply; while 11.8 per cent. came from the South. That is, over seven-eighths of the raw material of the iron industry was supplied from those two districts.

PRODUCTION OF PIG IRON BY STATES.
(In tons of 2240 lb.)

States.	1903	1904	1905	1906	1907
Massachusetts	3,265	3,149	15,987	20,239	19,119
Connecticut	14,501	8,922			
New York	552,917	605,709	1,198,068	1,552,659	1,659,752
New Jersey	211,667	262,294	311,039	379,390	373,189
Pennsylvania	8,211,500	7,644,321	10,579,127	11,247,869	11,348,549
Maryland	324,570	293,441	322,096	386,709	411,833
Virginia	544,034	310,526	510,210	483,525	478,771
Alabama	1,561,398	1,453,513	1,604,062	1,674,848	1,686,674
N. Car. and Georgia	75,602	70,156			
Texas	11,653	5,530	38,699	92,599	55,825
West Virginia	199,013	270,945	298,179	304,534	291,066
Kentucky	102,441	37,106	63,735	98,127	127,946
Tennessee	418,368	302,096	372,692	426,874	393,106
Ohio	3,287,434	2,977,929	4,586,110	5,327,133	5,250,687
Illinois	1,692,375	1,655,991	2,034,483	2,156,866	2,457,768
Michigan	244,709	233,225	288,704	369,456	(a) 436,507
Wisconsin and Minn	283,536	210,404	351,415	373,323	322,083
West of Miss. Riv	270,289	151,776	407,774	413,040	468,486
Total	18,009,252	16,497,033	22,992,380	25,307,191	25,781,361

(*a*) Includes Indiana.

CONSUMPTION OF PIG IRON IN THE UNITED STATES.
(In tons of 2240 lb.)

	1905	1906	1907
Production	22,992,380	25,307,091	25,781,361
Imports	212,465	379,828	489,440
Total	23,204,845	25,686,919	26,270,801
Exports	49,221	83,717	73,844
Approximate consumption	23,155,624	25,603,202	26,196,957

Steel.

The total production of steel in the United States for a period of years, as reported by the American Iron and Steel Association, is shown in the accompanying table. The total of 1907, though a little below that of the preceding year, was 2.6 times that of 1898, and 61 per cent. in excess of the

production of 1903. It exceeded that of Germany and Great Britain together.

PRODUCTION OF STEEL IN THE UNITED STATES.
(In tons of 2240 lb.)

Kinds.	1901	1902	1903	1904	1905	1906	1907
Bessemer	8,713,302	9,138,363	8,577,228	7,859,140	10,941,375	12,275,830	11,667,549
Open-hearth	4,656,309	5,687,729	5,837,789	5,908,166	8,971,376	10,980,413	11,549,088
Crucible and special	103,984	121,158	112,238	92,581	111,196	141,893	143,000
Total tons	13,473,595	14,947,250	14,527,255	13,859,887	20,023,947	23,398,136	23,359,637
Total metric tons	13,689,945	15,186,406	14,756,691	14,081,645	20,344,330	23,772,506	23,733,391

PRODUCTION OF OPEN-HEARTH STEEL.
(In tons of 2240 lb.)

Year.	Acid.		Basic.		Year.	Acid.		Basic.	
		Per ct.	Tons.	Per ct.		Tons.	Per ct.	Tons.	Per ct.
1901	1,037,316	22.3	3,618,993	77.7	1905	1,155,648	12.9	7,815,728	87.1
1902	1,191,196	20.9	4,496,533	79.1	1906	1,321,653	12.0	9,658,760	88.0
1903	1,094,998	18.8	4,734,913	81.2	1907	1,269,773	11.0	10,279,315	89.0
1904	801,799	13.6	5,106,367	86.4					

The loss of 608,281 tons in bessemer steel in 1907 was not quite offset by the gain of 568,675 tons in open-hearth, so that we find in the total the small decrease of 38,499 tons, or 0.16 per cent. The year 1907 came very near fulfilling the prediction made three years ago that the open-hearth steel output would soon equal that of bessemer or converter steel.

GEOGRAPHICAL DISTRIBUTION OF THE STEEL PRODUCTION IN 1906 AND 1907. (a)

	1906			1907		
	Bessemer.	Open-hearth.	Total.	Bessemer.	Open-hearth.	Total.
Pennsylvania	4,826,725	7,710,940	12,537,674	4,351,841	7,867,705	12,219,546
Ohio	3,769,913	816,483	4,586,396	3,636,679	819,642	4,456,321
Illinois	1,685,056	884,472	2,569,528	1,723,073	1,013,251	2,736,324
Other States	1,993,559	1,559,094	3,552,653	1,955,956	1,848,490	3,804,446
Total	12,275,253	10,970,998	23,246,251	11,667,549	11,549,088	23,216,637

(a) Disregarding the small quantity of crucible steel. In addition to the States named in the table, Massachusetts, Connecticut, New York, New Jersey, Delaware, Maryland, District of Columbia, Virginia, West Virginia, Kentucky, Michigan, Wisconsin, Minnesota, Missouri, Colorado, and Oregon made steel ingots or castings in 1907 by the standard bessemer process or by modified bessemer processes.

Acid and Basic Steel.—In this connection the accompanying tables show the steel output of the three chief producing countries, classed by the processes used. The tables show wide differences in the use of processes. The choice is controlled largely by the nature of the ore supplies in the different countries.

STEEL PRODUCTION OF PRINCIPAL COUNTRIES.
(In metric tons.)

Kind of Steel.	1906			1907		
	United States	Germany.	Great Britain.	United States	Germany.	Great Britain.
Acid converter	12,081,942	407,688	1,286,564	11,854,230	387,120	1,300,800
Basic converter	....	6,772,804	590,737	....	7,212,454	588,207
Total converter	12,081,942	7,180,492	1,877,301	11,854,230	7,599,574	1,889,007
Acid open-hearth	1,300,800	230,668	3,325,483	1,290,089	212,620	3,438,936
Basic open-hearth	9,497,426	3,534,612	1,157,721	10,443,784	4,039,940	1,299,168
Total open-hearth	10,798,226	3,765,280	4,483,204	11,733,873	4,252,560	4,738,104
Crucible and special	116,633	189,313	....	145,288	211,498	....
Total	22,996,801	11,135,085	6,360,505	23,733,391	12,063,632	6,627,111
Proportions steel to pig iron	92.3	89.2	63.7	90.6	92.5	65.7

MAKE OF ACID AND BASIC STEEL.

	1906		1907	
	Acid. Long Tons.	Basic. Long Tons.	Acid. Long Tons.	Basic. Long Tons.
United States	13,596,866	9,649,385	12,937,322	10,279,315
Germany	704,677	10,425,054	674,371	11,199,282
Great Britain	4,685,840	1,776,434	4,665,095	1,857,653
Total	18,987,383	21,850,873	18,276,788	23,336,250

Changes and Consolidations.

The most important change during 1907 was the sale of the Tennessee Coal, Iron and Railroad Company to the United States Steel Corporation. The control of the Tennessee company, the most important in the Southern field, had for several years been owned by a speculative group in New York, and the company had been operated in connection with the Southern branch of the Republic Steel and Iron Company. The panic in New York placed the owners in a precarious position, and to save themselves they sold their interests to the Steel Corporation, which also agreed to take the minority stock. The price was $84 per share payable in Steel Corporation second-mortgage, 5 per cent. bonds at 84. The sale was, therefore, made on the basis of $11,094.76, face value, in the bonds for $10,000 par value of the stock. Practically all the stock was deposited under this agreement; the total issued bv the Tennessee company having been $33,067,900 common and $248,300 preferred. In 1906 the company turned out 641,887 tons of pig iron and 401,882 tons of steel; quantities equal to 5.7 and 3 per cent., respectively, of the Steel Corporation's output. The improvements and additions in progress were expected to increase production to 1,000,000 tons of pig iron and 600,000 tons of steel.

The main value of the acquisition was in the supplies of raw material controlled. The policy of the Tennessee company had been to secure iron-ore reserves wherever possible in its territory. It owned by far the largest quantity of ore existing in the United States anywhere outside of the Lake Superior region. An expert's careful estimate of the iron ore unmined in the lands which it controls is 700,000,000 tons. This ore is of lower grade than the Lake Superior ore; but the company also had coal lands estimated to contain 2,000,000,000 tons, and the proximity of the coal and iron-ore deposits, reducing transportation to its lowest possible limit, offset the lower tenor of the ores. The Southern ores are, almost universally, above the bessemer limit in phosphorus, and steel making there is necessarily confined to the basic open-hearth process. The use of the open-hearth furnace, however, is quite in line with recent developments. With the reserves acquired the Steel Corporation added largely to the hold on the raw material of the iron industry, the control of which has, apparently, always been its object.

The purchase added about $2,500,000 to the yearly fixed charges which the Steel Corporation has to meet. There is not doubt, however, that with proper management the property purchased can provide for these charges without difficulty. Moreover, the iron ore control is an asset the value of which can hardly be estimated in money.

New Work.

The most important work in progress during 1907 was on the great new steel plant of the United States Steel Corporation at Gary, Ind. Substantial advance has been made in the construction of this plant, and some of its units were ready to operate in 1908. The Carnegie branch of the Steel Corporation made many changes and improvements, and at the close of the year had begun the remodeling of its Edgar Thomson plant, the construction of several new mills and the substitution of open-hearth furnaces for a number of its bessemer converters. The Steel Corporation also began work on an extensive steel plant at Duluth, which is intended to supply the trade in the Northwest.

The Jones & Laughlin Steel Company at Pittsburg, the Cambria Steel Company at Johnstown and several other independent companies were engaged in making additions to their productive capacity. The Tennessee Coal, Iron and Railroad Company began work on the additions to its steel plant at Ensley, Ala., which will double its capacity.

Imports and Exports.

The figures in the accompanying tables are taken from reports of the Bureau of Statistics of the Department of Commerce and Labor; the tables include only the more important items of exports and imports.

TOTAL VALUE. (a)

	1904	1905	1906	1907
Exports	$128,455,613	$142,928,513	$172,555,588	$197,036,781
Imports	21,621,970	26,392,728	34,827,132	38,789,992
Excess, exports	$106,833,643	$116,535,785	$137,728,456	$158,246,789

(a) Including machinery.

UNITED STATES EXPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900	1901	1902	1903	1904	1905	1906	1907
Pig iron	286,687	81,211	27,487	20,379	49,025	49,221	83,317	73,844
Billets, blooms, etc	107,385	28,616	2,409	5,445	314,324	237,638	192,616	79,991
Bars	94,665	45,105	31,549	37,182	55,472	51,870	88,102	98,654
Rails	361,619	318,055	67,455	30,656	414,845	295,023	328,036	338,906
Sheets and plates	54,865	30,832	18,300	18,093	55,204	75,034	110,654	122,696
Structural steel	67,714	54,005	53,859	30,641	55,514	83,193	112,555	138,442
Wire	78,014	88,238	97,843	108,521	118,581	142,601	174,014	161,228
Wire-rods	10,652	8,165	24,613	22,360	20,073	6,514	5,896	10,653
Nails and spikes	43,379	29,881	35,994	42,664	45,112	47,756	59,491	56,826

UNITED STATES IMPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1900	1901	1902	1903	1904	1905	1906	1907
Pig iron	52,565	62,930	625,383	599,574	79,500	212,465	379,828	489,440
Billets, blooms, etc	12,709	28,164	289,318	261,570	10,807	14,637	21,337	19,334
Scrap iron and steel	34,431	20,130	109,510	82,921	13,461	23,731	19,091	27,687
Bars	19,685	20,792	28,844	43,393	20,905	37,298	35,793	39,746
Rails	1,448	1,905	63,522	95,555	37,776	17,278	4,943	3,752
Wire-rods	21,092	16,804	21,382	20,836	16,206	17,616	17,999	17,076
Tin plates	60,386	77,395	60,115	47,360	71,304	65,740	56,983	57,773

The United States Steel Corporation.

Inasmuch as the United States Steel Corporation controls two-thirds of the iron and steel products of the country, its report for 1907, given herewith in condensed form, furnishes an accurate indication of the course of the iron trade generally.

The capital stock remains unchanged at $360,281,100 preferred and $508,302,500 common stock. The bonded and debenture debt is $600,947,081, made up as follows: Fifty-year, 5-per cent. bonds—of which $23,758,000 are held by the sinking fund and deducted from the total debt —$393,957,000; 10-60 year 5-per cent. bonds, $200,000,000; obligations of subsidiary companies, $120,758,081. In addition there are mortgages and purchase money obligations of subsidiary companies amounting to $5,393,941. The sinking and reserve funds amounted on Dec. 31 to $94,479,324, and undivided profits to $122,645,244. Current liabilities amounted to $45,063,825; but they appear small in comparison with $274,411,303 current assets, which include cash, accounts receivable, stocks and materials on hand.

SUMMARY OF EARNINGS AND EXPENSES. (a)

	1905	1906	1907
Gross receipts....	$585,331,736	$696,756,926	$757,014,768
Operating expenses.......	$440,013,432	$517,083,955	$564,166,777
General expenses.	18,570,374	22,907,679	25,395,379
Total.........	$458,583,806	$539,991,634	$589,562,156

	1905	1906	1907
Net earnings...	$126,747,930	$156,765,292	$167,452,612
Interest on investments, etc.	6,057,134	9,159,864	9,748,950
Total.....	$132,805,064	$165,925,156	$177,201,562
Account, subsidiary Cos....	13,017,406	9,300,883	16,236,889
Final net earnings..........	$119,787,658	$156,624,273	$160,964,674

(a) Includes operations of Tennessee Coal, Iron and Railroad Company for November and December, 1907.

The operating expenses shown in the accompanying table include all current repairs and maintenance of plant, as well as all working expenses. These expenses were 74.5 per cent. of the gross earnings. The expenditures made by all companies during 1907 for maintenance and renewals, including the relining of blast furnaces, and for extraordinary replacements, amounted to $55,828,253, an increase in comparison with the

PRODUCTION OF THE U. S. STEEL CORPORATION.

	1904	1905	1906	1907
Iron Ore Mined—	*Tons.*	*Tons.*	*Tons.*	*Tons.*
From Marquette Range..............................	934,512	1,359,722	1,442,290	1,170,496
From Menominee Range..............................	1,186,104	1,871,979	1,874,680	1,625,358
From Gogebic Range..............................	1,271,831	1,671,747	1,465,375	1,425,457
From Vermilion Range..............................	1,056,430	1,578,626	1,794,186	1,724,217
From Mesabi Range..............................	6,054,210	12,004,482	14,068,617	16,458,273
Total..	10,503,087	18,486,556	20,645,148	22,403,801
Coke Manufactured..............................	8,652,293	12,242,909	13,295,075	12,373,938
Coal Mined, not including that used in making coke.........	1,998,000	2,204,950	1,912,144	1,841,259
Limestone Quarried..............................	1,393,149	1,967,355	2,227,436	2,957,163
Blast Furnace Products—				
Pig iron..	7,210,248	9,940,799	11,058,526	10,631,620
Spiegel..	100,025	158,071	150,044	130,554
Ferro-Manganese and Silicon........................	59,148	73,278	58,807	57,794
Total..	7,369,421	10,172,148	11,267,377	10,819,968
Steel Ingot Production—				
Bessemer Ingots..............................	5,427,979	7,379,188	8,072,655	7,556,460
Open-Hearth Ingots..............................	2,978,399	4,616,051	5,438,494	5,543,088
Total..	8,406,378	11,995,239	13,511,149	13,099,548
Rolled and Other Finished Products for Sale—				
Steel Rails..	1,242,646	1,727,055	1,982,042	1,733,814
Blooms, Billets, Slabs, Sheet and Tinplate Bars	932,029	1,253,682	1,096,727	758,699
Plates..	404,422	780,717	836,399	877,682
Heavy Structural Shapes..............................	313,779	484,048	620,823	587,954
Merchant Steel, Skelp, Hoops, Bands and Cotton Ties....	577,384	982,782	1,240,548	1,316,387
Tubing and Pipe..............................	710,765	911,346	1,025,913	1,174,629
Rods..	84,934	84,049	111,488	126,095
Wire and Products of Wire..............................	1,226,610	1,283,943	1,399,717	1,481,226
Sheets—Black, Galvanized and Tin Plate..............	757,482	924,439	1,112,542	1,070,752
Finished Structural Work..............................	357,488	404,732	643,622	719,887
Angle and Splice Bars and Joints....................	72,470	150,265	176,730	195,157
Spikes, Bolts, Nuts and Rivets....................	46,003	61,496	70,233	67,991
Axles..	62,981	149,596	181,913	189,006
Sundry Iron and Steel Products....................	25,787	28,236	79,736	77,463
Total..	6,792,780	9,226,386	10,578,433	10,376,742
Spelter..	29,963	29,781	28,884	31,454
Copperas (Sulphate of Iron)..............................	15,805	20,040	21,933	24,540
	Bbl.	*Bbl.*	*Bbl.*	*Bbl.*
Universal Portland Cement..............................	539,951	1,735,343	2,076,000	2,129,700

expenditures for the same purposes during the preceding year of $7,495,163, or 15.5 per cent. The expenditures in 1907 were the largest of any year in the organization's history. Of these expenditures $35,503,668 were for maintenance and repairs, and $20,324,585 from the special funds provided for extraordinary replacements and repairs. On Dec. 31 there was a total of $41,360,655 remaining to the credit of the various depreciation and replacement funds.

The production of the several properties for a period of years, excluding the operations of the Tennessee Coal, Iron and Railroad Company for 1907, is shown in the accompanying table, the figures being in long tons, except cement, which is given in barrels.

Employees.—The average number of employees in the service of all companies during 1907, in comparison with 1906, was as follows, the figures for 1906 being given in parenthesis: Manufacturing properties, 151,670 (147,048); coal and coke properties, 21,447 (21,929); iron ore mining properties, 16,462 (14,393); transportation properties, 18,133 (16,638); miscellaneous properties, 2468 (2449); total, 210,180 (202,457); total annual salaries and wages, $160,825,822 ($147,765,540).

On Jan. 1, 1907 (on March 1, 1907, in case of the coke companies), an advance was made in the wages and salaries of approximately 65 per cent. of the total employees of all companies. This advance increased the wages and salaries of the employees affected about 6.6 per cent. The average rate of wages and salaries paid all employees during 1907 was above 5 per cent. higher than the similar average rate in 1906.

In January, 1908, there was again offered to the employees of the United States Steel Corporation and of the subsidiary companies the privilege of subscribing for 25,000 shares of preferred stock on substantially the same conditions as offered previously, except the price was fixed at $87.50 per share. The offer was over-subscribed by about 100 per cent., applications having been received from 24,884 employees for an aggregate of 50,075 shares. Allotments were made as follows: Each subscriber for one share was allotted the same, and all others were allotted 50 per cent. of their subscriptions. The total number of shares allotted on this basis was 30,621.

Reference was made a year ago to the proposal to establish from the employees' bonus fund for 1907 a separate fund to be used for pension purposes. Accordingly the sum of $1,000,000 was reserved for this purpose. The plan under which the benefits for the pension fund will be extended to employees is under consideration.

Tennessee Purchase.—In November, 1907, the corporation acquired $30,374,825, par value, of the common stock, and of common stock subscription receipts of the Tennessee Coal, Iron and Railroad Company, being all but $220,160 of the total. Of this foregoing aggregate of $30,374,-

825, par value, $29,742,170 were purchased at par, the corporation delivering in payment therefor its 10-60 year 5-per cent. sinking fund gold bonds as the equivalent of cash, at the rate of $11,904.76, par value, of said bonds for $10,000, par value, of Tennessee common stock; the balance of $632,655 of said stock acquired was paid for in cash, being the installment of 20 per cent. payable Dec. 15 on the stock subscription receipts. The total cost to the corporation of the stock acquired as above was $35,317,635. Of the bonds delivered as above, $30,000,000 were bonds of this issue held in the corporation's treasury, which had been executed in 1903. The balance of the bonds delivered for the purpose aforesaid was acquired through purchase in the open market.

During 1907 the production of the mineral and manufacturing properties of the Tennessee Company was as follows: Iron ore, 1,576,757 long tons; limestone and dolomite, 244,059 long tons; coal, not including that used in making coke, 1,709,251 short tons; coke, 1,170,826 short tons; pig iron, 602,827 long tons, of which 287,354 tons were for conversion into steel, and 315,573 for market; open-hearth steel ingots, 243,444 long tons; steel rails, 146,171 tons; billets, plates and bars, 88,009 tons. The net profits for the Tennessee Coal, Iron and Railroad Company for 1907, after charging off $437,667 for depreciation and extraordinary replacements, and $885,552 for net interest charge on bonded and floating debt, were $1,426,684. Extensive outlays were made during the year for construction and improvement work. The total amount expended aggregated $6,589,117. Large further outlays are now being made on improvements of the property, especially the steel plant.

General Conditions.—During the first 10 months of 1907 the several departments of the subsidiary companies were operated at substantially their maximum capacity. The production by the manufacturing properties of finished products for sale during the period from Jan. 1 to Nov. 1, 1907, showed an increase of 5 per cent. over the corresponding period in 1906. Owing, however, to the severe check to general trade conditions in the fall of 1907, the volume of business of the subsidiary companies was materially curtailed, resulting during the last two months of the year in a decrease in production of finished products which exceeded the gain made during the preceding 10 months. The production of finished steel products for sale in the entire year of 1907 shows a decrease compared with 1906 of 201,691 tons, or about 2 per cent. The decrease in actual shipments of products made to customers showed a somewhat larger falling off, 10,451,488 tons of all kinds of manufactured materials (including furnace products and scrap) having been shipped in 1907, against 10,862,425 tons in the preceding year. Prices of steel commodities for domestic sale were not generally advanced during 1907, notwithstanding there were marked increases in the cost of raw materials and supplies used in

manufacturing, in railroad freight charges, in wages and in taxation charges.

The satisfactory results obtained from the export business through the building up of a permanent and continuous export trade, as noted in previous reports, have continued. During 1907 there were shipped for export 1,014,082 tons of steel commodities of various kinds, a decrease of 6 per cent. as compared with the shipments in the previous year. The gross receipts for the 1907 shipments, however, exceeded those for 1906 by 16 per cent. The average mill price per ton received for all exported materials was only 7½ per cent. less than the average price received for all domestic shipments. The advantages to the employees, the domestic consumer, and the manufacturer, of a fair volume of foreign trade during periods of business depression in the United States have been emphasized in previous reports.

Substantial progress was made during 1907 in the construction of the new plant at Gary, Ind., the building of the city of Gary, and in the terminal railroad work and facilities adjacent to the steel plant. The expenditures in connection with the foregoing to Dec. 31, 1907, including the cost of about 9000 acres of land, amounted to $24,063,388. This has been provided entirely from surplus net profits of the corporation.

There has been purchased a site containing about 1580 acres (of which 300 acres are now submerged) in St. Louis county, Minn., 10 miles from the center of Duluth, on which it is proposed to construct a moderate-sized iron and steel plant. The tract fronts on the St. Louis river, which is navigable for large lake steamers, and is situated on the line of the Northern Pacific Railway. It is expected, however, that there will be constructed in connection with the steel plant a belt railway extending northwardly to a connection with the Duluth, Missabe & Northern Railway, to enable the prompt and economical delivery of ore to the plant from the Minnesota ore ranges; also eastwardly from the steel plant across the St. Louis river into Superior, Wis., where connections will be made with five trunk lines of railways reaching the head of Lake Superior from the south and west. It is proposed to have the plant constructed by the Minnesota Steel Company, a subsidiary corporation.

The Iron and Steel Markets.

The year 1907 opened with pronounced activity everywhere. Not only were mills and furnaces busy on all sides, but new orders were coming in freely, and a large demand was manifest. Prices of pig iron were on a high level, but this did not seem to check buying. Owing to the policy strictly adhered to by the Steel Corporation, prices of finished steel were not advanced, but maintained at about the same level as in the second half of 1906. Before the end of March, the capacity of mills and furnaces

had been engaged well up to the end of the third quarter of the year, and even into the fourth quarter.

Toward the middle of the year some uncertainty seemed to develop. Consumers hesitated to make commitments too far ahead, and began to consider whether concessions in price might not be obtained. The strike of the Mesabi miners late in July, while it was ended too soon to affect the iron trade seriously, seemed to be a signal for a further development of distrust. It was aided by the growing tension of the money markets and the difficulty of obtaining new capital for construction enterprises. It began to be evident that credit had been strained, and that business required a period of rest and recovery. The failure of one or two construction companies, which had been operating largely upon borrowed money, gave an impetus to the feeling of uncertainty. New business declined rapidly, and fourth-quarter orders were few in number. In pig iron and steel billets, which are practically the only open markets, there were declining prices, though the fall was gradual. The bank panic, which struck the financial centers in late October, put an end to new orders for the time, and practically for all the rest of the year.

The year closed with this doubtful situation. At the end, however, a little more hopeful feeling was manifest, as to the prospects of the new year, though no one expects more than a gradual revival of trade.

Details of the markets will be found in the following paragraphs:

Pittsburg (By S. F. Luty).—Clever management by the leading interest prevented a disastrous termination of 1907, which at the start promised to be the banner year in the history of the iron and steel industry. A tightening of the money market late in October caused alarm, and buyers of material began to send in cancellations of orders which threatened to cause a serious slump and demoralize the markets. The United States Steel Corporation refused to accept cancellations, but entered the orders as postponed and large tonnages remained on the books, independent interests following the plan of the big producer.

About the middle of November, when the situation looked decidedly unfavorable, it was decided to restrict production in all the weak lines. Accordingly on Dec. 1 the Steel Corporation had closed more than one-half of its 95 blast furnaces and a large number of its steel plants were on the idle list. Merchant furnaces were closed during the last month in the year when all contracts had been completed.

The year opened with the pig-iron market decidedly strong; weakness developed in February, but recovery came in the following month, and prices were at the high point in May. Active buying of pig iron for second-half delivery began early in April, when some large orders were booked. Sales of second-half iron in the first two weeks of April exceeded 260,000 tons. Late in May many inquiries were received by furnaces for all grades

of pig iron for 1908 delivery, but no important contracts were closed for standard bessemer, and in a few weeks interest in iron for extended delivery disappeared. With the exception of some sales at slightly reduced prices in June, there were no sales of importance during the closing half of the year.

The average monthly prices of bessemer pig iron based on sales of 1000 tons and more are as follows, the first figure being the price at Valley furnaces, and the second at Pittsburg: January, \$22.06—\$22.91; February, \$21.93—\$22.78; March, \$22.05—\$22.90; April, \$21.36—\$22.21; May, \$23.28—\$24.13; June, \$23.25—\$24.15; July, \$22.41—\$23.31; August, \$22—\$22.90; September, \$22—\$22.90; October, \$21.88½—\$22.78½; November, \$19.75—\$20.65; December, \$18.87½—\$19.77½.

Some important changes in pig-iron freight rates to this district were made during the year. Effective Jan. 1 the rate from Shenandoah furnaces to Pittsburg was reduced from \$2.55 to \$2.40, but from other Virginia furnaces was advanced 25c., from \$2.55 to \$2.80. On Aug. 1 the rates were advanced 10c., making the rate from Shenandoah \$2.50 and other furnaces \$2.90, except Bristol, which remained at \$4.35. The rate from Birmingham, Ala., to Pittsburg was advanced on April 1 from \$4.60 to \$4.85. On June 1 the pig-iron rate from Valley furnaces to Pittsburg was increased from 85c. to 90c. On July 1 the rate for coke from the Connellsville region to Valley furnaces was advanced from \$1.30 to \$1.35, but the rate to Pittsburg remained at 80c.

There were no labor troubles of any consequence during the year and the big strike in the ore regions did not effect operations of the furnaces in the Pittsburg or surrounding territory.

Six new blast furnaces were blown in during the year in the Pittsburg

AVERAGE PRICES AT PITTSBURG, 1907.

Month.	Pig Iron.			Ferro Man-ganese.	Steel.					Nails.	
	Bessemer	No. 2 Foundry.	Gray Forge.		Bessemer Billets.	Rails.	Sheets No. 28.	Tank Plate.	Steel Bars.	Wire Per Keg.	Cut Per Keg.
	\$	\$	\$	\$	\$	\$	Cts.	Cts.	Cts.	\$	\$
January......	23.35	25.35	22.35	81.00	29.50	28.00	2.60	1.70	1.60	2.00	2.05
February.....	22.85	24.35	21.85	74.00	29.00	28.00	2.60	1.70	1.60	2.00	2.05
March........	23.85	24.85	21.85	75.00	29.00	28.00	2.60	1.70	1.60	2.00	2.05
April.........	23.85	25.85	22.10	71.00	30.00	28.00	2.60	1.70	1.60	2.00	2.05
May.........	24.85	25.85	22.85	69.00	30.50	28.00	2.60	1.70	1.60	2.00	2.05
June.........	24.40	25.40	22.90	64.00	29.50	28.00	2.60	1.70	1.60	2.00	2.05
July.........	23.90	23.90	22.90	63.00	30.00	28.00	2.60	1.70	1.60	2.00	2.05
August.......	22.90	22.90	21.90	62.00	29.50	28.00	2.60	1.70	1.60	2.00	2.10
September....	22.90	21.90	20.90	58.00	29.50	28.00	2.60	1.70	1.60	2.05	2.10
October......	22.90	20.90	19.90	57.00	28.00	28.00	2.60	1.70	1.60	2.05	2.05
November....	20.40	20.40	19.40	53.00	28.00	28.00	2.60	1.70	1.60	2.05	2.00
December....	19.90	19.40	18.90	52.00	28.00	28.00	2.60	1.70	1.60	2.05	2.00

district and surrounding territory. On Jan. 31 the Carnegie Steel Co. broke ground at Youngstown, O., for an open-hearth steel plant of twelve

50-ton furnaces. The work was completed in February, 1908. The Pittsburg Steel Company, the largest independent wire producer, in June began the erection of a steel plant adjoining its works at Monessen, Penn.

The pig-iron prices in the accompanying table are the average prices for the different months based on all sales.

Chicago (By E. Morrison).—Opening with high prices and heavy consumption of pig iron, the year 1907 was for the first three months highly satisfactory to sellers of iron. The second quarter developed a plenty of raw material and a production up to the consuming limit on the part of melters. In the third quarter sales of pig iron became generally confined to the needs of the next two or three months. By the beginning of the fourth quarter hardly anybody was buying for more than 30 days ahead, and this condition continued down to the end of the year. Iron and steel products showed a course similar to that of pig iron. From a beginning of unsurpassed activity the December market went to the other extreme The causes of the year's decline are general and well understood.

IRON AND STEEL PRICES AT CHICAGO.

Material.	1906		1907	
	Highest.	Lowest.	Highest.	Lowest.
Lake Superior Charcoal	$26.50	$19.00	$28.00	$25.00
Northern No. 2 Foundry	27.00	18.00	27.00	19.00
Southern No. 2 Foundry	26.90	16.90	27.35	19.35
Connellsville Coke	6.90	5.40	5.15	5.50
Bar Iron	1.85c	1.665c	1.865c	1.75c
Structural Material (*a*)	1.865c	1.865c	1.88c	1.865c

(*a*) Beams and channels, 3 in. to 15 in., and angles 3 in. to 6 in. x ¼ in. or heavier.

Alabama (By L. W. Friedman).—Despite a complete reversal in conditions of the pig-iron market during the last two months of the year, Alabama's iron output for 1907 did not fall so much under what it was in 1906, when the State produced 1,674,848 tons. The figures of the statisticians give Alabama credit for 1,629,856 tons of iron during 1907.

The high mark in production was reached in May. This was followed by a reduction, owing to the necessity of repairing several furnaces; but an increase was recorded in July. In October the total was 151,815 tons; a sharp curtailment brought it down to 126,156 tons in November, while in December the production was cut fully 40 per cent. from the highest point, the month being given credit for 97,952 tons.

During the summer there was some trouble in securing raw materials at the furnace, partly because labor was scarce, and partly on account of transportation delays. This difficulty was overcome before the general drop came.

Owing to the conditions prevailing during the year, prices were exceptionally hard to report. The prices of No. 2 foundry, the standard upon which all prices are based, were as follows, the first figure being for spot and those in parenthesis for second half delivery: January, $24.00 (. . . .); February, $22.50 (. . . .); March, $23.50 ($19.00); April, $24.00 ($20.00); May, $24.00 ($20.00); June, $22.50 ($19.00); July, ... ($19.00); August, ... ($19.00); September, ... ($18.50); October, ... ($17.50); November, $16.00 ($14.00); December, $14.00 ($12.75).

The steel works at Ensley and Gadsden were full of work throughout the year. Cast-iron pipe works were busy until November, when there came a sudden cessation of orders, and some of them shut down. Foundries, stove works and machine shops did well. The rolling mills were also busy for 10 months of the year. The most important new work done was on the enlargement of the steel works of the Tennessee company at Ensley, making it practically a new plant, double the capacity of the old one. Two new blast furnaces were completed, Gadsden, of the Alabama Consolidated Coal and Iron Company, and Vanderbilt, of the Birmingham Iron and Coal Company. Three old blast furnaces were rebuilt. A new railroad entered Birmingham, the Atlanta, Birmingham & Atlantic, which gives the district a new outlet to the sea at Brunswick. The most important change of the year was the sale of the Tennessee Coal, Iron and Railroad Company to the United States Steel Corporation. This involved a reorganization of the Southern branch of the Republic Steel and Iron Company, which had been practically consolidated with the Tennessee company.

Iron and Steel Production of the World.

The world's production of pig iron during 1907 showed an increase of 1,505,000 tons over that of 1906. The United States furnished 43 per cent. of the total, Germany and Great Britain coming next in order as producers.

PIG IRON PRODUCTION OF THE WORLD.

(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,311,949	1,161,180	87,612	2,714,298	7,549,665	23,990	2,296,191
1901	1,300,000	765,420	248,896	2,388,823	7,785,887	25,000	2,869,306
1902	1,335,000	1,102,910	325,076	2,427,427	8,402,660	24,500	2,597,435
1903	1,355,000	1,299,211	269,665	2,827,668	10,085,634	28,250	2,486,610
1904	1,369,500	1,307,399	274,777	2,999,787	10,103,941	27,600	2,978,325
1905	1,372,300	1,310,290	475,491	3,077,000	10,987,623	31,300	2,125,000
1906	1,403,500	1,431,160	550,618	3,319,032	12,478,067	30,450	2,350 000
1907	1,405,000	1,427,940	590,444	3,588,949	13,045,760	32,000	2,768,220

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	289,788	526.868	9,003,046	14,009,870	625,000	39,599,457
1901	294,118	528,375	7,977,459	16,132,408	635,000	40,950,692
1902	330,747	524,400	8,653,976	18,003,448	615,000	44,342,579
1903	380,284	506,825	8,952,183	18,297,400	625,000	47,113,730
1904	386,000	528,525	8,699,661	16,760,986	633,000	46,069,501
1905	383,100	531,200	9,746,221	23,340,258	655,000	54,054,783
1906	387,500	552 250	10,311,778	25,706,882	650,000	59,074,861
1907	385,000	535,000	10,082,638	26,193,863	625,000	60,680,014

STEEL PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,145,654	655,199	23,954	1,565,164	6,645,869	115,887	2,217,752
1901	1,142,500	526,670	26,501	1,425,351	6,394,222	121,300	2,230,000
1902	1,143,900	776,875	184,950	1,635,300	7,780,682	119,500	2,183,400
1903	1,146,000	981,740	181,514	1,854,620	8,801,515	116,000	2,410,938
1904	1,195,000	1,069,880	151,165	2,080,554	8,930,291	113,800	2,811,948
1905	1,188,000	1,023,500	403,449	2,210,284	10,066,553	117,300	1,650,000
1906	1,195,000	1.185,660	515,200	2,371,377	11,135,035	109,000	1,763,000
1907	1,195,500	1,183,500	516,300	2,677,805	12,063,632	115,000	2,076,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	144,355	300,536	5,130,800	10,382,069	400,000	28,727,239
1901	122,954	269,897	5,096,301	13,689,173	405,000	31,449,869
1902	163,564	283,500	5,102,420	15,186,406	412,000	34,972,497
1903	199,642	317,107	5,114,647	14,756,691	418,000	36,298,414
1904	193,759	333,522	5,107,309	13,746,051	415,000	36,148,079
1905	237,864	340,000	5,983,691	20,354,291	426,000	43,900,648
1906	251,600	351,900	6,565,670	23,772,506	420,000	49,635,998
1907	247,100	353,000	6,627,112	23,733,391	405,000	51,183,340

The increase in the steel production of the world in 1907 was 1,281,261 tons, or 2.6 per cent., as compared with 1906. The ratio of steel to pig iron produced was 90.6 for the United States, 92.5 for Germany and 65.7 for Great Britain.

PIG IRON PRODUCTION IN BELGIUM.
(In metric tons.)

	1903	1904	1905	1906	1907
Foundry iron	....	99,350	98,170	101,430	100,020
Forge iron	....	224,410	206,309	226,900	226,430
Steel pig	....	963,840	1,006,641	1,103,130	1,101,490
Total	1,216,500	1,287,400	1311,120	1,431,460	1,427,940

FOREIGN TRADE OF BELGIUM.

	Imports.				Exports.			
	1904	1905	1906	1907	1904	1905	1906	1907
Pig iron	347,455	507,970	694,530	....	39,708	41,295	31,445	....
Wrought iron	88,937	97,864	83,643	....	529,284	628,473	530,119	....
Steel	231,801	230,169	259,077	....	387,063	395,229	245,101	....
Total	668,193	936,003	1,037,250	....	956,055	1,044,997	807,665	....

Canada.—The accompanying table shows the production of pig iron in Canada, as reported by the American Iron and Steel Association. The

PIG IRON PRODUCTION IN CANADA.

	1902	1903	1904	1905	1906	1907
Foundry and forge	....	....	....	146,698	130,120	84,979
Bessemer	....	....	....	149,203	165,609	154,910
Basic	....	....	....	172,102	246.228	341,257
Total	319,557	265,418	270,942	468,003	541,957	581,146

total increase, which was 7.2 per cent., was entirely in basic pig. On Dec. 31, 1907, Canada had 16 completed furnaces, of which 14 were in blast and 2 were idle. Of the total 13 usually use coke for fuel and 3 use charcoal. In addition 3 coke furnaces upon which work was suspended some time ago were partly erected on Dec. 31. During the first half of 1907 the total number of furnaces in Canada actually in blast for the whole or a part of the period was 12, of which 10 used coke and 2 used charcoal. During the last half of the year the total number of active furnaces was 15, of which 13 used coke and 2 used charcoal. The most important addition during the year was the furnace of the Atikokan Iron Company at Port Arthur, Ont., which went into blast in July.

The most important iron ore producer in Canada is the Helen mine on the Michipicoten range. The following are the shipments from this mine for a period of years: 1900, 53,470 long tons; 1901, 232,505; 1902, 298,430; 1903, 203,413; 1904, 118,355; 1905, 169,527; 1906, 121,555; 1907, 142,832.

PIG IRON PRODUCTION IN FRANCE.
(In metric tons.)

	1902	1903	1904	1905	1906	1907
Foundry	....	....	....	635,672	591,275	651,700
Forge	....	....	....	705,691	741,571	673,885
Bessemer	....	....	....	160,411	149,971	122,046
Basic	....	....	....	1,530,671	1,784,726	1,988,343
Special irons	....	....	....	44,267	51,489	152,975
Total	2,427,427	2,827,668	2,999,787	3,076,712	3,319,032	3,588,949

STEEL PRODUCTION IN FRANCE.
(In metric tons.)

	1902	1903	1904	1905	1906	1907
Wrought iron	625,826	595,831	554,632	669,841	747,900	687,249
Steel ingots	1,635,300	1,854,620	2,080,554	2,210,284	2,436,322	2,677,805
Finished steel	1,231,652	1,317,400	1,482,708	1,442,071	1,454,456	2,261,217

France.—The materials used in the manufacture of steel in 1907 comprised 35,000 tons of iron ore, 91,152 tons of bessemer pig iron, 1,852,506 tons of basic pig iron, 61,179 tons of manganiferous iron, 203,927 tons of forge iron, 143,621 tons of special pig iron, 35,750 tons of muck bar and 581,055 tons of old material. The finished steel products reported in 1907 were 297,762 tons rails; 43,845 tons tires; 1,049,824 tons bars, beams, shapes, pipes, etc.; 352,042 tons sheets and plates; 33,570 tons forgings; 31,505 tons castings. In addition makers sold 452,669 tons in the form of blooms and billets.

Germany.—The production of pig iron since 1903, as given by the German Iron and Steel Union, is shown in the accompanying table. This

PRODUCTION OF PIG IRON IN GERMANY.
(In metric tons.)

	1903	1904	1905	1906	1907
Foundry iron	1,798,773	1,865,599	1,905,668	2,108,684	2,259,416
Forge iron...	859,253	819,239	827,498	854,536	786,113
Steel pig.....	703,130	636,350	714,335	943,573	1,034,650
Bessemer pig	446,701	392,706	425,237	482,740	471,355
Thomas pig .	6,277,777	6,390,047	7,114,885	8,088,534	8,494,226
Total ...	10,085,634	10,103,941	10,987,623	12,478,067	13,045,760

shows increases of 155,732 tons in foundry iron; 91,077 tons in steel pig, which includes spiegeleisen, ferromanganese, ferrosilicon and all similar alloys; 405,692 tons in Thomas, or basic pig. There were decreases of 68,423 tons in forge iron, and 11,385 in bessemer pig. The total gain last year over 1906 was 572,693 tons, or 4.6 per cent. The statement of the steel production, from the same source, is shown in the accompanying table.

PRODUCTION OF STEEL IN GERMANY.
(In metric tons.)

	1905		1906		1907	
	Acid.	Basic.	Acid.	Basic.	Acid.	Basic.
Converter ingots..........................	424,196	6,203,706	407,688	6,945,526	387,120	7,212,454
Open-hearth ingots........................	165,930	3,086,590	230,668	3,534,612	212,620	4,039,940
Special steels............................	65,369	120,762	77,596	111,717	85,421	126,077
Total............................	655,495	9,411,058	715,952	10,591,855	685,161	11,378,471

ACID AND BASIC STEEL IN GERMANY.
(In metric tons.)

Kind.	1903	1904	1905	1906	1907
Acid steel....................................	613,399	610,697	655,495	715,952	685,161
Basic steel....................................	8,188,116	8,319,594	9,411,058	10,591,855	11,378,471
Total....................................	8,801,515	8,930,291	10,066,553	11,307,807	12,063,632

GERMAN IMPORTS AND EXPORTS OF IRON ORE
(In metric tons.)

	1904	1905	1906	1907
Imports	6,061,127	6,085,196	6,730,636	8,476,076
Exports	3,440,846	3,698,563	3,212,977	3,904,400

GERMAN EXPORTS AND IMPORTS OF IRON AND STEEL.
(In metric tons.)

	1904	1905	1906	1907
Exports	2,770,276	3,349,968	3,619,796	3,432,707
Imports	344,967	322,907	690,081	813,104

Russia.—Some recovery in iron and steel is reported though the industry is still affected by the unsettled condition of the country. It is difficult to secure authentic figures, but estimates give the following production:

PRODUCTION OF IRON AND STEEL IN RUSSIA.
(In metric tons.)

	1902	1903	1904	1905	1906	1907
Iron ore	3,987,303	4,218,600	5,272,300	4,050,000	4,580,000	4,400,000
Pig iron	2,597,435	2,486,610	2,978,325	2,125,000	2,350,000	2,768,220
Steel ingots	2,183,400	2,410,938	2,811,948	1,650,000	1,763,000	2,076,000
Steel rails	382,152	332,367	401,541	275,000	315,000	....

Spain.—Exports of iron ore in 1907 were 8,635,868 metric tons, a decrease of 636,614 tons over the previous year. There was a noticeable depression in the iron ore industry during 1907, owing to the low prices offered by the English iron-masters, and the increased cost of mining at greater depths.

PRODUCTION OF IRON AND STEEL IN SPAIN.
(In metric tons.)

	1902	1903	1904	1905	1906 (a)	1907 (a)
Pig iron	330,747	380,284	386,000	383,100	387,500	385,000
Wrought iron	53,252	53,288	53,177	52,250	57,100	53,200
Bessemer steel	103,389	105,263	93,100	113,664	116,200	115,500
Open-hearth steel	60,175	94,379	100,659	124,200	135,400	131,600
Total steel	163,564	199,642	193,759	237,864	251,600	247,100

(a) Estimated.

Sweden.—According to a summary published in *Stahl und Eisen*, the finished production of Sweden in 1907, including castings, blooms and billets, bars, rolled shapes, sheets, pipes and tubes, wire and nails, was 399,500 metric tons, as compared with 401,300 tons in 1906. The production of iron ore in 1907 was approximately 3,500,000 tons.

United Kingdom.—The British Iron Trade Association has reported the production of pig iron as given in the accompanying table. Of 507

PRODUCTION OF PIG IRON IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907
Forge and foundry	3,875,826	3,841,975	4,276,943	4,587,606	4,412,985
Bessemer	3,700,422	3,362,883	4,070,222	3,990,820	3,776,797
Basic	991,610	1,192,120	1,057,999	1,263,317	1,406,038
Spiegel and ferro	183,346	165,680	187,573	307,645	228,036
Total	8,811,204	8,562,658	9,592,737	10,149,388	9,923,856

furnaces standing the average number in blast was 368 in 1906, and 366 in 1907. The average yearly output per furnace was 27,598 tons in 1906, and 27,096 in 1907; a decrease of 502 tons. By districts, the larger yearly makes in 1907 were 46,726 tons in South Wales and 40,224 in the Cleveland district; the smaller were 16,132 in Scotland, and 14,213 in Derbyshire. A feature in 1907 was the gain in basic pig, when there were decreases in all other kinds.

There was a remarkable decrease in the stocks in public stores, which were reported at 730,752 tons on Dec. 31, 1906, and 221,885 at the end of 1907. The decrease of 508,867 tons was 283,335 tons more than the loss in production. It does not follow, however, that consumption was greater last year than in 1906. The decrease in iron in stores was partly due to the fact that the use of public stores and warrant trading are on the decline, and more iron is stored in furnace yards.

PRODUCTION OF STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907
Open-hearth	3,124,083	3,245,346	3,879,748	4,554,936	4,663,489
Bessemer	1,910,018	1,781,533	2,009,712	1,907,338	1,859,259
Total	5,034,101	5,026,879	5,889,460	6,462,274	6,522,748

The gain of 108,533 tons in open-hearth steel was partly offset by a loss of 48,079 tons in converter steel; the total gain in 1907 being 60,474 tons, or 0.9 per cent. The year slightly increased the lead of open-hearth over bessemer steel.

ACID AND BASIC STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1905			1906			1907		
	Acid.	Basic.	Total.	Acid.	Basic.	Total.	Acid.	Basic.	Total.
Open-hearth	3,084,510	795,238	3,879.748	3,378,691	1,176,245	4,554,936	3,384,780	1,278,709	4,663,489
Converter	1,117,731	891,981	2,009,712	1,307,149	600,189	1,907,338	1,280,315	578,944	1,859,259
Total	4,202,241	1,687,219	5,889,460	4,685,840	1,776,434	6,462,274	4,665,095	1,857,653	6,522,748

The proportion of the total steel to pig iron production was 63.7 in 1906, and 65.7 in 1907. This proportion is much lower than in the United States or in Germany. The puddling furnace is still more in use in Great Britain than in the other great iron-making countries, though it shows now some decrease. The output of wrought iron was 1,010,346 tons in 1906, and 975,083 tons in 1907; a decrease of 35,263 tons. Partial reports of the output of finished steel show that bessemer was preferred for rails, bars and similar forms; while open-hearth was used for plates, sheets, structural shapes and ship building material.

PRODUCTION OF FINISHED STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1905		1906		1907	
	Bessemer.	Open-hearth.	Bessemer.	Open-hearth.	Bessemer.	Open-hearth.
Blooms and billets	286,082	402,535	277,845	498,656	245,644	580,961
Rails	951,552	108,953	854,740	94,626	832,576	79,532
Bars, including tin-plate bars	288,980	839,415	242,706	939,087	321,138	950,938
General merchant steel and shapes	187,973	....	82,246	198,389	72,728	266,821
Plates and sheets	....	1,765,111	....	1,734,446	....	1,769,855
Total	1,714,587	3,116,014	1,457,537	3,465,204	1,472,086	3,648,107

Imports of iron ore into Great Britain were, in 1904, 6,100,556 long tons; 1905, 7,344,786; 1906, 7,823,084; 1907, 7,638,934. Of the imports in 1907, Spain furnished 239,845 tons of manganiferous ores and 5,472,-645 tons iron ores.

The British foreign trade, including machinery, is estimated by the Board of Trade as follows:

	1903	1904	1905	1906	1907
Exports	£54,741,296	£53,587,013	£60,524,755	£75,256,655	£88,448,689
Imports	8,662,481	12,529,212	13,128,270	13,486,724	12,527,157

The chief items of the iron and steel exports were as follows, in long tons:

	1903	1904	1905	1906	1907
Pig iron	1,065,380	810,934	981,891	1,662,820	1,947,925
Wrought iron	203,619	170,505	183,406	200,182	211,771
Sheets	....	385,408	407,021	442,414	469,329
Plates	161,722	152,337	204,503	275,045	300,590
Rails	604,076	525,371	546,644	460,328	433,638
Steel shapes, etc	156,821	122,930	151,809	226,230	338,716
Tin plates	292,800	359,634	354,951	374,802	405,329
All other kinds	....	735,723	891,290	1,040,379	1,059,068

The larger exports to the United States in 1907 were 439,437 tons pig iron, an increase of 128,738 tons; 58,920 tons of tin-plates, a decrease of 2598 tons from the preceding year.

Lake Superior Iron Ore.

By Dwight E. Woodbridge.

Lake Superior iron ore production for 1907 was nearly 9 per cent. greater than in 1906 and more than double that of 1900. Of the total iron ore produced during the past 55 years, half has been mined within the last six years. The production of iron ore for a series of years since the industry became commercial is shown in the accompanying table.

PRODUCTION OF IRON ORE.
(In tons of 2240 lb.)

Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.
1855	1,449	1875	881,166	1895	10,429,037	1904	21,822,839
1860	114,401	1880	1,948,334	1900	19,059,393	1905	34,353,456
1865	193,758	1885	2,466,642	1902	27,562,566	1906	38,522,239
1870	859,507	1890	9,003,725	1903	24,289,674	1907	(a)42,406,405

(a) The Atikokan Iron Company shipped 18,500 tons in 1907; this tonnage has not been considered in totals heretofore published.

The division of shipments by ranges during 1905, 1906 and 1907 is shown in the accompanying table.

SHIPMENTS OF IRON ORE FROM LAKE SUPERIOR.
(In tons of 2240 lb.)

Range.	1904	1905	1906	1907
	Tons.	Tons.	Tons.	Tons.
Marquette	2,843,703	4,210,522	4,057,187	4,388,073
Menominee	3,074,848	4,495,451	5,109,088	4,964,728
Gogebic	2,398,287	3,705,207	3,643,514	3,637,907
Vermilion	1,283,513	1,677,185	1,792,355	1,685,267
Mesabi	12,152,008	20,153,699	23,792,553	27,492,949
Baraboo	67,480	111,391	128,742	76,146

LAKE SUPERIOR ORE SHIPMENTS TO END OF 1907.
(In tons of 2240 lb.)

Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.
Marquette	85,245,874	22.4	Gogebic	54,107,342	14.2	Mesabi	150,235,558	39.5
Menominee	63,641,213	16.7	Vermilion	26,785,426	7.1	Baraboo	401,678	0.1

IRON ORE AT LAKE ERIE PORTS.
(In tons of 2240 lb.)

Ports.	Receipts.			Stocks, Dec. 1.		
	1905	1906	1907	1905	1906	1907
Toledo	1,006,855	1,423,741	1,314,140	368,024	281,000	518,645
Sandusky	51,202	35,847	83,043	52,977	17,467	44,546
Huron	825,278	778,453	971,430	208,023	245,499	415,730
Lorain	1,605,823	2,191,965	2,621,025	271,695	336,321	366,271
Cleveland	5,854,745	6,604,661	6,495,998	1,330,619	1,224,606	1,281,335
Fairport	2,008,621	1,861,498	2,437,649	759,961	590,783	523,981
Ashtabula	6,373,779	6,833,352	7,521,859	1,589,951	1,631,312	2,056,820
Conneaut	5,327,552	5,432,370	5,875,937	976,976	1,057,424	1,090,774
Erie	2,112,476	1,986,539	2,294,239	564,961	552,631	652,219
Buffalo	3,774,928	4,928,331	5,580,438	315,780	315,412	435,407
Total	28,941,259	32,076,757	35,195,758	6,438,967	6,252,455	7,385,728

LAKE ORE PRICES.

Bessemer Ores:	Old Range.		Mesabi.		Non-bessemers:	Old Range.		Mesabi.	
	1906	1907	1906	1907		1906	1907	1906	1907
Iron %, natural	56.70	55.00	56.70	55.00	Iron %, natural	52.80	51.50	52.80	51.50
Moisture	10.00	10.00	10.00	10.00	Moisture	12.00	12.00	12.00	12.00
Iron %, 212 deg	63.00	61.12	63.00	61.12	Iron %, 212 deg	60.00	58.52	60.00	58.52
Phosphorus %	0.045	0.045	0.045	0.045	Base price	$3.70	$4.20	$3.50	$4.00
Base price	$4.25	$5.00	$4.00	$4.75					

During 1907 six concerns shipped 80 per cent. of all iron ore mined in the Lake Superior district. Their names, the tonnage shipped and the percentage of the total are as follows: United States Steel Corporation, 22,719,898 tons, (53.6 per cent.); Corrigan, McKinney & Co., 3,020,933 (7.1); Pickands, Mather & Co., 3,014,980 (7.1); Cleveland Cliffs Iron Company, 2,399,363 (5.7); Tod-Stambaugh Co., 1,745,106 (4.1); Cambria Steel Company, 1,333,848 (3.1). These companies all mined at least part of their ore for their own consumption; part was mined for sale.

Aside from eight properties on the Mesabi, no single mines of the Lake region reached the million-ton class. Two groups, the Norrie, of the Cogebic, and the Cleveland Cliffs, of the Marquette, are credited with respective productions of 1,109,085 and 1,030,928 tons. Generally speaking, the larger mines of the old ranges show a diminution of production from maximums. Of the eight big properties on the Mesabi, six are operated by the Steel Corporation, one by the Mahoning Ore and Steel Company, and one by Corrigan, McKinney & Co.

Labor.—The labor situation of the year was serious. Attempts of the Western Federation of Miners to organize the region, more particularly the Mesabi, culminated in June in a great strike. This complicated and followed a strike of ore-dock men at Duluth and adjacent ports. The conditions that presented themselves, however, were met by the mine

owners so successfully that the strike was broken and the Federation left with no strength and no standing.

Mines and Railroads.—In various past years the shipments of various individual mines aside from the Mesabi range ran into the seven-figure class. But a change has come. Of the eight mines of the million-ton class this year, all are on the Mesabi. All but two are Oliver Iron Mining Company properties. One of the Oliver mines produced the enormous total of 2,900,000 tons, and several were about 2,000,000 tons. Such figures have been unknown heretofore.

Railroads that have handled such tonnages deserve their share of the credit for the year's operations. It is no small thing to move from mines to ships, in one month 2,250,000 tons, all one-way freight. The Duluth, Missabe & Northern road did this.

The tonnages of iron ore carried by the various railroads serving the mines and shipped over their docks to lower lake ports are shown in the accompanying table.

SHIPMENTS OF IRON ORE.
(In tons of 2240 lb.)

Railroad.	1905 Tons.	1906 Tons.	1907 Tons.	Railroad.	1905 Tons.	1906 Tons.	1907 Tons.
Duluth, Missabe & N..	8,804,443	11,230,218	13,445,970	Duluth, S. S. & Atlantic................	1,243,388	1,074,945	1,323,346
Duluth & Iron Range.	7,778,768	8,205,126	8,188,905	Wisconsin Central....	799,394	693,892	878,730
Chicago & Northw'n..	6,729,975	6,706,986	6,971,111	Wis. & Mi. (car ferry).	....	62,757	....
Great Northern.......	5,118,385	6,133,057	7,505,125	All rail, not included..	198,000	....	....
Chic., Mil. & St. Paul.	1,310,021	1,931,244	1,585,961	Algoma Central.......	169,527	121,555	....
L. Sup. & Ishpeming..	1,844,823	1,889,631	1,916,168				

The roads have few plans for improvements during 1908. The Duluth & Iron Range is adding a steel ore-shipping pier at a cost of $1,100,000; the Duluth, Missabe & Northern road is constructing a large coal-receiving dock; and the Chicago & Northwestern is rebuilding its Ashland pier at a cost of $7,000,000.

The Western Mesabi.—The new west Mesabi field is of vast extent. While thousands of drill holes have been put through ore it has not yet been completely explored. But enough has been proved to give assurance that, with operations on the scale planned, there is ore to last many decades. At its Canisteo property the Oliver company is digging a mining pit that will be more than 200 acres in extent. Three miles northeast is the Holman deposit, itself a vast iron ore body. Still further east is the Arcturus, of many million tons, and other deposits of importance. Open-cut mining will be the method generally followed in this part of the region. The ore of these deposits will not average more than 40 per cent. metallic iron, and will carry a heavy excess of silica. As it comes from the concentrator it is brought up to an ore at least 20 per cent. better in iron, frequently

inside the bessemer limit as to phosphorus, and with 80 per cent. of the silica removed.

New Explorations on the Mesabi.—For the first year since the discovery of the Mesabi range it is a question if the new ore developed there has kept pace with that taken out. Few new deposits have been found. Most of the work of exploration was confined to the edges and bays appertaining to old ore finds, and in many cases this closer exploration has drawn together the confining lines of known orebodies as much in one place as it has expanded them in another. In the Embarras Lake district, east Mesabi, some ore has been found and some holes have shown a remarkable depth of high-grade iron. Some of these holes have gone more than 200 ft. in good ore, and the tonnage found there amounts to several million tons. On the west end of the range several good deposits of merchantable ore have been found, notably about 5,000,000 tons in section 9-56-23 close to the quartzite. West of the Mississippi river ore has been found, but it is low grade and in somewhat inaccessible locations. It seems evident that in the future additional Mesabi tonnages will be slight and will come from the more precise delimitation of known deposits, rather than from discoveries of new lenses.

Vermillion Range.—On the Vermillion range little new ore was found and the work of exploration in progress a year ago has ceased. The most favorable of these—that on section 30-63-11—has not been successful as hoped for, and has stopped.

Cuyuna District.—The new Cuyuna district, west of Duluth, is still a puzzle. There is no question that one mine has been found the past year, and so far it is the net result of drillings extended over a period of five years. The firm of Rogers, Brown & Co. was so well assured of a mine that it has paid large sums in advance royalties and is now sinking an operating shaft. Lands near have shown lean ores more than 800 ft. deep, with a vast tonnage, but useless on account of their low grade.

Menominee Range.—There has been much activity on the Menominee range. This is a wide and strong and persistent formation, and a number of companies are finding good deposits, most of which were mentioned last year. A number of new explorations in the vicinity of Stambaugh, Iron River, and Crystal Falls may prove mines before another winter. In the Iron River section the new shippers of the year were the Baker, of Corrigan, McKinney & Co.; the Mineral Mining Company's James mine; and Chatham of Oglebay, Norton & Co., the Hiawatha of the Buffalo & Susquehanna; and the Fogarty of Pickands, Mather & Co. Several new shafts are being sunk, notably at the Berkshire and Groveland. Many pieces of land on the iron bearing formation in the district have been optioned and ready takers are found for all developments.

Marquette Range.—The chief development of the year on the Marquette

range was the opening of the Maas mine, on which the Cleveland Cliffs Iron Company has been working for the past five years. Hoisting through a vertical shaft of large dimensions, which is now 1140 ft. deep, was commenced in the summer of 1907.

The Stephenson, Smith and Austin mines are new developments of the Cleveland Cliffs Company in the Cascade district. These mines will become an important center of ore production, and this may be referred to as one of the very important developments of the year.

Gogebic Range.—A number of footwall steel shafts and fireproof shaft-houses were erected along the Gogebic. Deep work under cross-dikes at the Newport, some time ago, gave the utmost confidence in this range, and this was increased by later developments. It is regarded as a safe proposition to go 2500 to 3000 ft. in search of ore. Deep drilling is of slight value, and the only way to determine the facts is by sinking and crosscutting. This is under way at many places. In the earlier days most Gogebic range work was from the hanging wall, and could be regarded as merely temporary. Of the five deep shafts recently sunk there, which have either steel headframes or are steel lined throughout, all but one are on the footwall side of the formation. All of the 10 all-steel shafts now sinking are in the same side. These are all planned to go to at least 2000 ft., and some will doubtless continue steadily to 3000 ft.

Baraboo Range.—In the Baraboo district, south of the lakes, the year saw little change. There is a narrow deposit extending a considerable distance and to some depth, and the average annual product is 50,000 tons. This is coming from the mine of the International Harvester Company, other people finding the district unfavorable.

Canadian Range.—On the Canadian side of the lake the Oliver Iron Mining Company has been working near Steep Rock lake, west of Lake Superior, and has developed one fair mine and some prospects. This is near the Atikokan deposit which is now being mined for the 100-ton furnace that was blown in at Port Arthur last June. The great deposits reported to exist east of and near Port Arthur, where 200,000,000 tons were said to be proven, have not yet been opened. The explorations that were maintained in the far north, east of Nepigon lake, have been discontinued and are succeeded by others still more distant, on the shores of Hudson bay, where distances are so vast and climatic conditions so severe that ore must command an abnormal price to permit exploitation. There remains only the Moose Mountain region, where preparations for the shipment of considerable tonnages of high-grade ore are well along.

Mine Equipment.—The process of standardizing and remodeling Lake iron-mine power plants and equipments on the latest and most efficient designs proceeded during the year. It was aided by the strength of mining companies and their willingness to make expenditures that a few years ago

would have been impossible. Hoisting equipment, shafthouses, shafts themselves, underground and overhead haulage, stripping devices, everything, has been standardized. Mesabi mines now use standard-gage locomotives and all-steel self-dumping stripping cars for their removal of overburden. There is no more of the dinky engine and the little one-ton stripping car. Tracks in mines are heavily and permanently built. Shafts are steel-lined, and are placed in solid ground, well away from the workings.

Costs.—There was little change in the expense of mining ore. Stripping costs vary now from 23c. up to 60c., the higher figures on account of winter and shallow work, long hauls for the waste, jobs of little yardage, bad weather or other untoward conditions. A steam shovel will handle in 10 hours, week by week, 2000 to 2500 yards. The average cost of mining ore underground and placing it on stockpile is not far from $1 per ton.

For years the ideas of mine operators as to the depth permissible in stripping overburden have been undergoing change and revision. Today the Oliver company is of opinion that it will pay to strip if the final cost of winning ore is as great as if mined underground. This because of the fact that the open pit gives a chance for big tonnages to be got immediately when needed, does away with a large amount of unskilled labor, cuts out timbering and the dangers incident to underground work, and has other advantages. This idea, carried out, will permit stripping to far greater depths than at present, and today mine operators are taking off a thickness of 100 ft. It will also permit a greater development of the stripping plan in small and thin deposits, and the rule of 2 ft. of surface to 1 ft. of ore, radical as that appeared two or three years ago, must give way to the 3 or 4 ft. to one.

Outlook for 1908.—Indications for 1908 are not especially brilliant. The enormous shipments of the past summer and fall have not been taken to furnaces, and there is little probability they will be consumed this winter. There will be accumulations of ore at lower lake ports and in furnace yards in the spring, greater than in the past. Labor troubles are likely on the lakes and the fleet will start out late. Careful and well posted mining men estimate that there will be a decline in shipments for 1908 of from 25 to 33 per cent. from those of 1907. This will bring the estimates for the new year to below or about 30,000,000 tons.

Concentration Experiments.—An interesting feature of the year was found in the concentrating experiments. The most important of these was that conducted by the Oliver Iron Mining Company on the West Mesabi. While this is by no means new, changes were introduced that seem to have solved the question of the concentration of the sandy ores of the western end of that range. The company first installed a washery, where the coarse ore, carrying a large proportion of free sand closely intermingled and attached, was broken and jarred and washed till the

bulk of this sand was carried away in the tailings. This method lost a considerable proportion of the finer ore, which enriched the tailings. The problem was to find some method that, while economical, might reduce the losses to the point where further refinements of the savings would become uneconomical. That has been the work of the past year. "Turbos," or supplementary log washers, Hancock jigs, settling tanks, classifiers, etc., have been added from time to time, and the problem seems now solved by the addition of the turbos and the jigs. The losses of fine ore have been reduced, while the costs of operation have not been increased to the point where the saved ore does not more than pay for itself.

Canisteo District.—While a certain share of the ores found in the Canisteo district are suitable for shipment without treatment, the tonnage that must be washed is enormous, and the preparations made for mining and shipment have been on a vast scale. Probably not less than $6,000,000 have so far been expended on the Canisteo district, in the construction of railway lines, the stripping of mines and building of towns, and in the experiments with concentration. One unit of what will in time be a plant capable of treating upward of 10,000 tons of ore per day has been erected for these experiments. By the successful outcome of these experiments a tonnage of hundreds of millions of tons of what, when washed, is desirable iron ore, has been added to the reserves of the Mesabi district.

Concentration on the Menominee.—Another interesting experiment in concentration is now in progress on the Menominee range. Great bodies of lean magnetic ores exist in that district, and these are to be crushed and concentrated by magnetic processes, including some new devices, eliminating the jasper and, to some extent, the phosphorus. By subsequent semi-roasting the concentrates are partially oxidized. These experiments have not yet progressed beyond preliminary stages, but semi-hematite nodular ores of fine character have been produced.

Electric Plants.—Aside from the Oliver company's various developments to which reference has been made, the installation of a complete underground electrical equipment for the Penn Iron Mining Company, at its Norway and Vulcan properties, is most interesting. This company has a group of mines producing about 500,000 tons a year, whose fuel costs are said to be reduced $110,000 per annum by this installation. A water power on the Sturgeon river, four or five miles from the mines, was developed for a head of 40 ft., and wheels driving 3700-kw. revolving field generators were placed.

Review of the Iron Ore Industry in the United States.

Alabama (By Eugene A. Smith).—The iron ores of Alabama in the order of their economic importance are: Red ore, or hematite; brown ore, or limonite; gray ore; and black band and clay iron stone. Until

very recently only the first two have been mined to any great extent. These ores are used in the manufacture of pig iron for foundries, mills and pipe works, and for making basic iron for the open-hearth steel plants. As a rule they carry too much phosphorus for use in the manufacture of bessemer steel. Practically all the ore mined in Alabama is smelted in the State, the quantity shipped out being about equal to that received from other States.

Alabama stands third in the production of iron ores and fourth in the production of pig iron. The production of pig iron during 1907 was 1,686,674 long tons, which indicates that the production of iron ore was about 4,198,324 tons, figuring that 2.49 tons of ore made one ton of pig iron, as in 1906. Approximately three-fourths of the production of iron ore was hematite and one-fourth limonite. There are 44 coke furnaces in the State, 31 of which were in blast on July 1, 1907; 5 charcoal furnaces were also in operation.

Red hematite, the most important ore of iron of the State, occurs in beds of varying thickness and degree of purity in the Red Mountain ridges to the northwest of the Coosa Valley. The gray ore occurs in the upper part of the Weisner formation, in the southern part of Talladega county. Brown hematite, or limonite, is found in largest quantities in Coosa, Murphree's and Jones' valleys. Important deposits also occur in the Tennessee valley.

Indiana (By W. H. Blatchley).—During 1907 interest has revived in the iron ore deposits of the State and the construction at Gary, Ind., of what will eventually be the largest system of blast furnaces and iron mills in the world, makes it an assured fact that Indiana is certainly to become one of the chief iron-producing States. The iron-ore deposits of Indiana are well worthy of consideration.

The larger deposits, which can be more easily worked, would yield at least 15,000,000 tons of ore, and the smaller accessible deposits, which would be easily worked out by the owners if furnaces were within a reasonable distance, would bring the total up to at least 25,000,000 tons of ore. In many cases the smaller deposits contain the better grades of iron; this would compensate for the extra cost in mining.

Iowa (By James H. Lee).—There was no production of iron ore in Iowa in 1907. The Missouri Iron Company, with headquarters at St. Louis, has for more than a year been working on a deposit of low-grade limonite mixed with some hematite, situated on Iron Hill, at Waukon, Allamakee county. This company is endeavoring to find a process by which this ore can be profitably treated and made of economic value. So far its efforts have not met with success, but this is hoped for in the near future.

The deposit covers an area of over half a square mile and forms a cap

for the highest hill in the country. The ore contains about 56 per cent. metallic iron, and very small amounts of phosphorus, sulphur and manganese. It has been mined to some extent for shipment to Milwaukee for use in furnace mixtures, its manganese content making it especially desirable for such purposes.

There are also other smaller deposits in the northeastern part of the State, but no extended attempts have been made to develop them.

Maryland (By Wm. Bullock Clark).—The production of metalliferous ores in Maryland is so small as to affect but little the State's total mineral output. Iron ore is much the most important metallic product, but with the closing down of the Muirkirk furnace and the consequent cessation of the mining of carbonate ore there is no production to report from the eastern counties. There was some revival in the mining of the brown hematite ore of the Piedmont district in Carroll and Frederick counties, which are known to contain, at several points distant from the railroads, extensive deposits of this material. The ore was shipped to furnaces in Pennsylvania. The total production of these products during 1907 is valued at $34,767, as compared with $15,624 in 1906.

Missouri (E. R. Buckley).—During 1907 there was increased activity in the iron mining industry, chiefly along the line of exploiting deposits which were previously but little developed. In Wayne and Butler counties there was considerable development work done and one washing plant was installed; development work also progressed at Billings. In fact everywhere throughout the Ozark region deposits which give any evidence of being of workable size were investigated.

The old and widely known mines at Iron Mountain passed into the hands of a new company; preparations are being made to open up the abandoned mines with the hope of striking new orebodies. South of Doe Run a company is developing the deposits of micaceous iron ore of that region with the expectation of using the output in the manufacture of a lubricant. An examination of a 2500-ft. drill core obtained by the Bureau of Geology and Mines from a hole drilled at Forest City, Holt county, in 1901, shows a 5-ft. bed of oolitic and fossil iron ore of Clinton age at a depth of 1952 ft. The occurrence of Clinton iron ore at this depth in this locality is not of any immediate economic importance although it is extremely interesting from a scientific standpoint.

New Jersey (By Henry B. Kümmel).—The total production of iron ore in this State during 1907 was 558,137 tons, as compared with 542,488 tons in 1906. The increase would have been more substantial had it not been for the closing down of several mines during the latter part of the year.

The center of the iron mining industry is Morris county, in which are situated the greatest number of mines as well as the most productive ones.

During 1907 there were 16 iron mines in operation within the State, all of which produced more or less continuously. The chief producing mines are in the hands of a few interests, Joseph Wharton, the Empire Steel and Iron Company, the Thomas Iron Company, the Musconetcong Iron Company, the Pequest company, and the Hudson Iron Company being the principal operators.

The ore of all these mines, with few exceptions, is magnetite. Much of the magnetite is of high grade. Between 1894 and 1904 there were shipped from the Richard mine over 1,000,000 tons showing an average content of 60.19 per cent. metallic iron. The phosphorus averaged 0.75 per cent.; silica, 6; lime, 5; alumina, 3. Some of the magnetite carried considerable sulphur in the form of iron pyrites.

In nearly all the magnetite mines the orebodies are most frequently long, flattened, pod-shaped masses, the longest axes of which pitch northeasterly at angles of from 15 to 25 deg. The thickness varies greatly, ranging from that of mere stringers up to 15 ft. in some cases.

New Mexico (By Chas. R. Keyes).—There is only one locality in New Mexico where iron is at present extensively mined. This is at Fierro, in Grant county. All of this ore is shipped to Pueblo, Colo., to the plant of the Colorado Fuel and Iron Company. The annual production is about 200,000 tons.

New York (By D. H. Newland).—Iron mining has undergone uninterrupted expansion during the last few years. The output for 1907 amounted to 1,018,013 long tons and exceeded that of any previous year since 1890. There were 13 mines under exploitation, or two more than in 1906, when the production was 905,367 tons. Several additional mines are under development preliminary to active work. The Clinton ore belt was the center of special interest, and large tracts of land in Wayne and Cayuga counties were acquired by companies with a view to mining operations. The Fair Haven Iron Company began shipments from this region for the first time in 1907. The Adirondack region also shared in the activity. The Benson mines, in St. Lawrence county, and the Cheever mine, near Port Henry, were reopened, while the deposits of titaniferous ores at Lake Sanford received attention; their operation is postponed only for the want of railroad facilities, which are planned for the near future. With a return of the iron market to normal conditions, it may be expected that the iron-ore production of New York will soon develop beyond all proportions of the past.

Wyoming (By H. C. Beeler).—The iron-ore production of Wyoming for 1907 is given at 558,849 long tons. This came from the mines of the Colorado Fuel and Iron Company at Sunrise, in northern Laramie county; this represents the total production of the State at the present time.

Deposits are known at Rawlins and in the Seminole district, northern Carbon county, but they are at present too far from transportation.

Progress in the Metallurgy of Iron and Steel in 1907.

By Bradley Stoughton and Cornelius Offerhaus.

Iron Ore Developments.

Canada.—Great interest is felt in Canada in the development of native ores, for, although many deposits are known, the greater amount smelted is imported from the United States. Several valuable discoveries were made in northern Ontario, and one large deposit of what is said to be high-grade bessemer hematite, containing over 200,000,000 tons, was discovered on Thunder Bay on the northern shore of Lake Superior. The Moose Mountain district was visited by the American Institute of Mining Engineers in July where some very promising deposits are located, which have just been made available by the opening of a railroad to the township of Hutton. The transportation facilities are now excellent. The ore is fairly high-grade magnetite, low in phosphorus and sulphur.

Cuba.—What appears to be the most important iron ore development in recent years is that of a deposit said to contain about 500,000,000 tons of limonite, bessemer ore, averaging when dry about 46 per cent. of iron and only 0.015 per cent. of phosphorus, with 10 per cent. alumina, 1.75 per cent. chromium, and 5 per cent. silica. It is located in the eastern end of the island and is owned by interests controlled by the Pennsylvania Steel Company. The ore lies on the surface so as to require no stripping, and is a blanket formation of red earth about 10 miles long, 4 miles wide and 15 ft. deep. It is situated on a plateau elevated about 2000 ft. above the sea, which drops off abruptly so as to make a railroad approach impracticable. An incline will therefore deliver the ore in 50-ton cars, loaded by steam shovels, at the foot of the plateau, where a railroad will take it to the town of Felton on the sea-coast, 10 miles distant. There the ore will be dried and loaded on steamers. (*Iron Age*, 1907, Vol. 80, pp. 421-6.)

Sweden.—The famous Grangesberg ore district was described by N. Hedberg (*Jernkontorets Annaler*, Vol. 62, pp. 67-125, abstracted in *Journal*, Iron and Steel Institute, No. 111, 1907, p. 322), where 64,000,000 tons of ore are said to be still available, the deposit having been known since 1584 and statistics preserved since 1783. The present annual output is about 605,000 tons. The ore is very free from sulphur and titanium, but high in phosphorus (0.07 to 8 per cent.). O. Stutzer gives an account of the Gellivare deposits, which he says are the most important in Sweden (*Berg- und Hüttenmännische Rundschau*, Vol. 3, pp. 273-6, abstracted in *Journal*, Iron and Steel Institute, No. 111, 1907, p. 323). The output is

over 1,000,000 tons per year, chiefly magnetite, running in three grades according to the phosphorus content: A-ore, less than 0.035 per cent.; C-ore, 0.035 to 0.8 per cent., and D-ore, over 0.8 per cent.

The Blast Furnace.

Distributor.—A stock distributor differing in principle from those described in previous reviews in The Mineral Industry was devised by Guy R. Johnson (*Iron Age*, 1907, Vol. 79, p. 1654). It consists of a curved revolving spout, with a valve at top and bottom which correspond respectively to the upper and lower bells of the ordinary mechanical charging devices. When the upper valve is raised the ore is dumped into the spout from the usual skip, and thence it is delivered as desired by lowering the lower valve. The top of the spout is in the middle of the furnace, but the lower end curves toward the side, and can be revolved to any point in the circumference before delivering the charge. The claims for the device are greater simplicity, less repairs, less weight and less breakage of coke.

Avoiding Explosions.—The same inventor described a method used by himself to avoid the explosions so often produced when blowing in a blast furnace, due to premature ignition (*Iron Age*, 1906, Vol. 78, 488). It is also said to give good iron from the very start. The method consists in building a box, extending from the center of the bottom of the furnace half-way through the tapping hole and filled with a mixture of one-half sand, one-half coke breeze. On top of this is placed hand-picked coke, free from dirt, up to the level of the tuyeres and then 3 ft. of tangled, dry, coarse kindling wood. Above this the furnace is filled as usual; the bells, the bleeder, the stove farthest from the furnace and the boiler farthest from the furnace are all opened, and about 10,000 to 12,000 cu.ft. of air started through the stock. As soon as the blast is on the kindling is lighted simultaneously from each of the tuyeres by means of kerosene-soaked shavings and a red-hot iron rod. In about a minute gas shows at the top and when this lights properly from a fire-basket swung over the bell, both bells are closed. The gas now passes through the mains to the stoves and boilers, driving the air before it without danger of an explosion. When it is burning well at the stove and boiler farthest from the furnace, the other burners are opened gradually, the blast being increased advisedly. When there is sufficient iron in the hearth the hole should be opened, and the zone of mixed sand and coke breeze extending from half-way into the tapping hole to the center of the hearth permits this to be done very effectively.

J. W. Dougherty has patented a method of avoiding explosions during blowing-in and after a shut-down by filling up all the spaces in the furnace, the down-comer, dustcatcher, stoves and underneath the boilers with

steam or some other incombustible gas. This acts as a piston between the gas and air, which drives the air before it and prevents any mixing which would produce premature ignition. (*Iron Age,* 1907, Vol. 79, p. 1587.)

Blast-Furnace Gas.—Léon Greiner, engineer of the Cockerill company (*Stahl und Eisen,* 1907, 1109), in a paper on the industrial production of power in metallurgical works using coke-oven and blast-furnace gas, states the available gas of coke ovens to be 35 per cent. of the total gas produced or 84 cubic meters of 4000 calories per cubic meter for each ton of coal. He states the available gas of the blast furnace for every ton of pig iron to be 40 per cent. of the total 4500 cubic meters produced, averaging 950 calories per cubic meter. The Cockerill works have two gas driven power houses; the larger one runs by blast-furnace gas, the other by coke-oven gas. Both generate direct current. During March, 1907, the works generated 5700 kw. The cost of 1 kw. during 1901 was 530 marks and this was reduced to 320 marks for the year 1906.

Dry vs. Moist Blast.—Josef Vajk ("Zur Frage der Windtrocknung"; *Stahl und Eisen,* 1907, p. 344) observed, that a barometric depression and the accompanying saturation of the air with water vapor caused a better running of the blast-furnace. He was able in such a case to decrease the temperature of the blast 50 to 100 deg. C. and to increase its pressure. The output increased and the regularity of the furnace was not affected. He explains these facts by the formation of ozone. Humid air is relatively richer in ozone than dry air and reduction of the air pressure can cause a maximum ozone-content of the air. The increase in temperature in the smelting zone surpasses the decrease caused by the greater humidity. It is known that the nitrogen of NH_3 unites more readily with iron than nitrogen of cyanides. Humidity causes decomposition of cyanides, forming ammonia compounds. Drying the blast therefore probably prevents the entrance of nitrogen into the iron, which would have a bad effect on its quality. This is perhaps an advantage of drying the blast which has not been given much consideration until recently.

Turbo-Blower.—Details were published of the Parsons turbo-blower for blast-furnaces (*Iron Age,* 1907, Vol. 80, p. 485, from *Zeitschrift des Verein Deutscher Ingenieure*). The advantages of the turbo-blower, of which there are now 23 in operation are: Cost is only one-half that of a reciprocating steam-engine blower, floor space occupied is only one-fifth, and there is greater economy in use of steam, of oil and in cost of repairs. A special feature of this type of machine is the governor which must control the turbine within a range of 2350 to 3400 r.p.m. A comparison of the results obtained with a turbo-blower at the Trzynietz works in Austria and with a reciprocating steam blower of equal capacity at the same works and at Hernadthal is given:

COMPARISON OF RECIPROCATING AND TURBO-BLOWERS.

	Turbine Trzynietz.	Reciprocating Engines.	
		Trzynietz.	Hernadthal.
Blast per minute, cubic meters...	570	540	558
Steam consumption per hour, kg..	5,349	5,880	5,500
Pressure, atmospheres..........	1.454	1.466	1.45
Consumption of steam per minute per 100 cubic meters of air, kg.	15.45	18.1	16.45

Gas Washing.—Notable advantages are secured by the washing of gas even when not used in gas engines, on account of the increased life and efficiency of the stoves. A combined gas washer and recuperator has been put upon the market in the United States (*Iron Age*, Jan. 17, 1907, p. 198), and a very valuable discussion of the subject took place before the Cleveland Institution of Engineers, England (*Proceedings*, 1906-7, pp. 93-142 and 145-197; also *Iron Age*, Feb. 28, 1907, p. 663), and in several articles (*Iron Age*, 1906, Vol. 78, pp. 542-5, 602-8, 1004-5; Vol. 79, pp. 1124-5, 1414-15; Vol. 80, pp. 155-6. *Journal*, Iron and Steel Institute, No. 111, 1907, pp. 210-15. *Iron Trade Review*, 1907, Vol. 40, pp. 739-740. *Stahl und Eisen*, 1907, Vol. 27, pp. 222-8, 1190-7, 618-623. *Engineering*, 1907, Vol. 82, p. 738. *Electrochemical and Metallurgical Industry*, 1906, Vol. 4, pp. 291-2).

Electric Smelting.

Types of Furnaces.—V. Engelhardt ("Electrische Inductionsöfen und ihre Anwendung in der Eisen-und Stahlindustrie." *Electrotechnische Zeitschrift*, 1907, pp. 1051; 1084; 1104; 1124) gives a very full account of the different types of induction furnaces used in the iron and steel industry. He mentions the patents of Ferrantis (1887) and Colby (1890) and a proposition by Dewey, which contain the fundamental ideas from which Kjellin built his furnace in Gysinge in 1899. He divides the induction furnaces into pure induction furnaces, in which the material in the total secondary circuit is heated only by induction, and combined furnaces, in which direct resistance heating is applied or in which only a part of the secondary circuit consists of the charge. To the first type belong the furnaces of Colby, Kjellin, Hjorth, Ferrantis, Frick, Schneider, Gin, and Wallin; to the second type the Röchling-Rodenhauser furnaces and also a furnace of Hjorth. The description of the different types of furnaces is illustrated by drawings. The author discusses the conditions for a practical economical induction furnace and how to approach them. The article contains a review of the evolution of the Kjellin furnace and a table of the Kjellin and Röchling-Rodenhauser furnaces built, containing location, kilowatts and capacity.

The following is a summary of the number of the different types of electro-steel works in operation, compiled by Pitival (*Comptes rendus* de la Société de l'Industrie minérale, March, 1907, p. 88), and V. Engelhardt (*Stahl und Eisen*, June, 1907, p. 807): Kjellin, 14; Gin, 2; Schneider, 1; Frick, 2; Wallin, 1; Colby, 1; Héroult, 10; Keller, 2; Girod, 1; Stassano, 3. A similar list is given in *Iron Age*, 1907, Vol. 79, p. 973, to which must be added a list on p. 1136.

Electric Smelting in Canada.—The final report by Dr. Eugene Haanel, Superintendent of Mines, Ottowa, Canada, on the experiments at Sault Ste. Marie on the smelting of Canadian ores by the electrothermic process was published in 1907. According to this report, hematite, magnetites, roasted pyrrhotite and titaniferous ores were successfully and economically treated, producing iron of different grades of silicon, satisfactorily eliminating sulphur, avoiding the usual objections to titaniferous ores and employing charcoal cheaply produced from otherwise useless refuse. On the basis of a cost of $50 per electric horse-power, Dr. Heroult estimates that a plant to produce 120 tons of pig iron per 24 hr. could be erected at a cost of $700,000, in which the expense of producing pig iron would be as follows: Ore (55 per cent. metallic iron) @ $1.50 per ton, $2.70; charcoal, ½ ton @ $6 per ton, $3; electric energy, amortization, etc., $2.43; labor, $1; limestone, $0.20; 18 lb. electrode at 2c. per lb., $0.36; general expenses, $1; total, $10.69 per ton.

Black Sands.—A report on the electric smelting of the black sands in Oregon in 1905 was issued by the United States Geological Survey; the results have not been successful up to the present.

Norwegian Ores.—Albert Hiorth (*Teknisk Ugeblad*, 1907, p. 92) first succeeded in making pig iron from Norwegian ore, containing 13 per cent. titanic acid, and Norwegian graphite, containing 20 per cent. SiO_2. He uses the electric furnace and the iron produced contains no titanium and only 0.01 per cent. silicon. His results are of special importance to Norway.

Additional References.—Several descriptions and discussions of electric smelting were published. (*Electrochemical and Metallurgical Industry*, 1907, Vol. 5, pp. 24-6, 92-3, 272-5. *Iron Age*, 1907, Vol. 79, pp. 332-6, 1740. *Journal*, Iron and Steel Institute, No. 1, 1907, pp. 476-86; No. 111, pp. 434-46). Bergmanns (*Teknisk Tidskrift*, 1907, pp. 90-92) describes the furnace and process of the Electrometall Company in Ludvika to produce iron electrically. The furnace has the form of a blast furnace; no blast, however, is applied. The gases are sucked out of the upper part of the furnace and again introduced in the lower part. B. Neumann ("Die Erzeugung von Roheisen im electrischen Ofen;" *Stahl und Eisen*, 1907, p. 1256) discusses the results of making pig iron in the electric furnace.

Manufacture of Steel.

Duplex Process.—In the manufacture of steel probably the most important development of the year 1907 in America was the increased amount of attention given the duplex process. This is now in use at Ensley, Alabama; Monterey, Mexico; Sydney, Nova Scotia; Pueblo, Colorado; it was tried in connection with the Talbot furnaces of the Jones & Laughlins' Company with success, but was discontinued because the arrangement of the plant was not advantageous for transferring the metal from the bessemer to the open-hearth furnaces, and the bessemer steel was needed for other purposes. The commoner type of practice is to blow the bessemer heats until the carbon is about 1 to 1.50 per cent., the silicon and manganese of course having been previously entirely eliminated, and then to work off the remainder of the carbon, the phosphorus and as much sulphur as possible in the open hearth. Ordinarily one-half the bessemer heats are blown down soft and the other half are blown to about 2 per cent. or so of carbon, which is the same as blowing all to 1 per cent. carbon. At Ensley the equipment consists of two 20-ton acid converters, four 100-ton rolling, basic, open-hearth furnaces and one 250-ton mixer. The blown metal is poured into the open-hearth on top of a heated mixture of lime and iron oxide, which produces a violent reaction that carries the phosphorus into the slag. At Sydney the slag at the end of each open-hearth heat is returned to the furnace, enriched with lime, and it is said that at the end of about four heats the slag is rich enough in phosphorus to be sold as fertilizer. The usual time of duplex heats is four to seven hours.

T. S. Blair, Jr., of the Lackawanna Steel Company, patented a method of operating the duplex process which has been tried experimentally and is said to have advantages over the usual procedure. The blown metal is poured into ladles situated on the same level as the open-hearth charging platform; each open-hearth furnace is independent of every other, both as to its receipt of metal and supplies and its disposition of finished steel; each open-hearth furnace is made 60 ft. between ports and has a chilled bridge wall extending across the middle of the hearth, dividing it into two hearths, each capable of holding 60 tons of metal. The object of this division is to get the fuel economy of the long hearth together with the two 60-ton furnaces at less cost. (*Iron Age*, 1907, Vol. 80, pp. 1452-4.)

H. H. Weaver and G. E. Thackray of the Cambria Steel Company patented a method of carrying out the duplex process which consists in charging some pig iron into a basic open-hearth furnace where it is dephosphorized, partly desulphurized and desiliconized. This metal is then introduced into a mixer together with the unrefined pig iron that is to be used for the bessemer process, and lowers its content of phosphorus and sulphur.

Open Hearth.—Th. Naske ("Beitrag zur Metallurgie des Martinprocess"; *Stahl und Eisen,* 1907, p. 127) made experiments on a large scale with the intention of contributing to the knowledge of the metallurgy of the open-hearth furnace. His conclusions are as follows:

Manganese as a catalytic agent. Mn is oxidized to MnO and Mn_3O and enters the slag as a base. Mn acts as an oxidizer or as a reducer until an equilibrium between the Mn in the metal-bath and the Mn in the slag is established. This equilibrium is influenced in the first place by the concentration of the Mn in the slag base. For a complete equilibrium a fixed proportion of iron and manganese is a necessity ($\frac{Fe}{Mn} = Q$). This ratio tends to approach 1 as a limit. When Q is greater than 1, oxidation of Mn of the bath, and when Q is less than 1, reduction of Mn of the slag, may be expected. The bulk of the Mn of the iron goes into the slag during the beginning of the process, the oxidation of Mn and the formation of slag being very rapid in this early stage. For this reason it is immaterial whether Mn is present in the pig iron or added in a suitable form afterwards. The presence of Mn is a necessity. Lack of Mn causes iron oxides to enter the metal bath which affects the quality of the product.

Phosphorus oxidizes and enters the slag; both reactions are exothermic. For the dephosphorization of iron, the presence of iron oxides or lime is necessary. The dephosphorization is favored by a high temperature. A definite iron-lime-silicate slag can be considered as most effective in dissolving the oxides of phosphorus. In this slag iron and lime cannot replace each other, but Mn can probably be partly substituted for the iron. Mn has no influence upon the oxidation of the P. The slag being saturated with P (the degree of saturation is fixed by the composition of the slag and its temperature) carbon, CO and iron (the last as a catalytic agent) may reduce the phosphates to P (an endothermic reaction).

Naske finally points out, that in the acid open-hearth process not alone oxidizing reactions, but also just as many reducing reactions take place.

Basic Bessemer.—The Flohr modification of the basic bessemer process has been adopted at several works. It consists in adding to the bath briquettes made of about 75 per cent. oxides of iron and 10 per cent. of lime, instead of the usual addition of lime at the drop of the carbon flame. The addition is made when the blow is hot and the slag still liquid enough to allow the briquettes to pass through it. The iron oxide cools the bath more quickly than lime, does not make the slag so infusible and gives an intense action which shortens the dephosphorization. In consequence of this shortening the waste is less, the lining is saved, steam is saved, and the productive capacity is increased. The iron oxide reduced also decreases the loss. The less amount of lime used saves expense and produces a slag higher in phosphorus.

Water-Cooled Fittings.—T. S. Blair, Jr., patented a water-cooled,

open-hearth port and bulkhead which has given excellent service at the Lackawanna Steel Company's plant. The furnace has already run more than twice as long as the average life of this type of furnace and gives indication of being practically indestructible. The inventor believes that with this type of port the only repairs to an open-hearth furnace that can not be made during the week-end shut-down is the replacement of regenerators, which will cause a loss of time of about 36 hours per year.

Lash Process.—Horace W. Lash patented a process for producing steel from a mixture of shotted pig iron, or pig iron turnings and borings, with double their weight of iron ore, by briquetting the mixture and melting in an open-hearth or electric furnace. Several tests are said to have been made with a good product and high yield. Magnetic ore has been used and titanium is no disadvantage (*Iron Age*, 1907, Vol. 80, p. 360; *Electrochemical and Metallurgical Industry*, 1907, Vol. 5, pp. 344-5, 455-6).

Mechanical Treatment.

Electrical Equipment.—There has been a very great extension of the use of electricity in rolling mills for all power purposes and especially for driving the rolls. (*Oesterreichische Zeitschrift für Berg-und Hüttenwesen*, Vol. 54, 417-24, 429-36, 442-7, 453-7, 467-70, 482-6, *Electrical Engineer*, Vol. 39, pp. 2, 165-8, 201-2, *Engineering Magazine*, Vol. 33, pp. 537-52.) Rolling mill engines continue to increase in capacity and a 20,000-h.p. engine no longer seems excessive. (*Iron Age*, 1906, Vol. 78, pp. 338-40, 1015, 1139-40; *Iron and Coal Trades Review*, 1907, Vol. 75, p. 1470; *Iron Trade Review*, 1907, Vol. 41, pp. 116-7; *Engineer*, Vol, 103, p. 602.)

Rolling and Edging Mill.—A new device which makes it possible both to roll and edge steel in the same pair of rolls was devised by Edwin E. Slick of the Carnegie Steel Company and adopted by one of the works of that company (*Iron Age*, 1907, Vol. 79, p. 1188). It consists of a pair of rolls, called "pinch rolls," between which the steel passes as it emerges from the flat pass. The pinch rolls do not touch the steel until the metal has completed this pass ; then the pinch rolls grasp the last end, raise it up in line with the edger pass and turn it through an angle of 90 deg. The pinch rolls are mounted on a rocker arm so that they can be moved to accomplish this transfer. They are also capable of being actuated by power so that, when they have brought the steel piece into line with the edger pass, they can feed into it.

Heating Furnace.—A new form of continuous ingot-heating furnace (*Iron Age*, 1907, Vol. 79, p. 287) contains a track submerged in water along which the buggies carrying the ingot travel. The lower part of the buggies are also submerged in water, while the upper part and the ingot are above the water and in the furnace chamber proper, the water serving as a seal for this chamber. The buggies travel continuously

toward the fire at one end, and there are hydraulic arrangements for introducing the buggies at one end and taking them out at the other.

Armor Plate.

Cost of Production.—The report of a special naval board of the United States, appointed to investigate the manufacture of armor-plate, was made public in 1906 (*Iron Age*, Vol. 78, pp. 1604-8), from which it appears that the price paid per ton for armor plate by the leading countries is as follows:

PRICE OF ARMOR PLATE.

Country.	Average of all Armor.	Krupp Armor.
Austria	$449	$ 557
England	626	681
France	569	572
Germany	450	450
Italy	521	550
Japan	400	400
United States	365.92	346

An estimate of the cost of making armor plate assuming that the manufacture was at the full capacity of the plant (a condition not realized in present practice), was $273.38 per ton for plates less than 5 in. thickness, and $296.89 for larger plates. A further estimate of the cost of a plant with an annual capacity of 10,000 tons of armor plate, not including land or an allowance on constructing capital, was $3,750,000. The following costs were furnished: Midvale Steel Company's armor-plate plant, over $3,500,000; Bethlehem Steel Company's armor-plate plant, $4,625,000 (including the value of land occupied); Carnegie Steel Company's armor-plate plant, $4,730,425.

European Armor Plate.—An account of the manufacture of armor plate in Europe (*Revue Générale des Sciences*, Vol. 17, pp. 682-95, 729-33) states that the composition of the metal is: Carbon, 0.30 per cent.; manganese, 0.35 per cent.; nickel, 2.50 per cent.; and chromium, 0.60 per cent. It is also stated that the amount of nickel is sometimes altered but the maximum amount is 4 per cent., and that the chromium increases in proportion to the nickel, so that steel with 4 per cent. nickel would contain about 1.8 per cent. chromium. It is generally stated that the composition in the United States includes 3.5 per cent. nickel and 1.50 per cent. chromium, with 0.25 per cent. carbon, except, of course, on the outer surface where the carbon is increased by cementation to about 2.50 per cent. It is stated that the Charpy process uses metal considerably different in composition. This process is secret and is said to be applicable to thicker plates than the Krupp process.

Properties of Cast Iron.

Effect of Composition.—Max Orthey ("Chemische Zusammensetzung und Festigkeit des Gusseisens;" *Metallurgie,* 1907, p. 196) investigated the relation between chemical composition and physical properties (tensile strength and transverse strength) of cast iron. The physical properties depend mostly on the content of combined carbon and for this reason he determined first how much combined carbon cast iron contains in cooling it slowly, varying the thickness of the test piece and the content of silicon and sulphur (Mn was not considered). The problem was then brought back to that of finding the relation between combined carbon and physical properties. The results of his experiments show that in order to get a high tensile strength and a moderate transverse strength 20 to 25 per cent. of the total carbon should be present as carbide and this is procured with a silicon content of 1 to 1.5 per cent. and a sulphur content of 0.06 to 0.15 per cent., both depending upon the thickness of the casting. The content of manganese must be about 0.5 per cent. and the content of phosphorus, 0.2 to 0.5 per cent. In order to get a high transverse strength and a moderate tensile strength 1.4 to 2 per cent. silicon should be present, depending upon the thickness of the casting and as little manganese, phosphorus and sulphur as possible. For a high tensile strength and a high transverse strength, the mean of the above proportion should be taken.

Malleable Castings.—F. Wüst ("Untersuchunger über die Festigkeitseigenschaften und Zusammensetzung des Tempergusses;" *Metallurgie,* 1907, p. 45) investigated the strength and properties of malleable castings as a function of their composition. After reviewing the theory of the process, the investigator considers the influence of impurities upon the mechanical properties of malleable castings, and the change in chemical composition and mechanical properties caused by tempering twice in different melting apparatus (cupola and crucible). The main conclusions are the following:

1. The tensile strength is independent of the percentage of Si, P, and S, as long as 1.2 per cent. Si, 0.1 per cent. P and 0.2 per cent. S are not reached. 2. The iron carbide being once broken up by the heat, it is immaterial as far as the tensile strength is concerned, how much of the temper-carbon is burned out. 3. The elongation and reduction of area are greatly reduced if more than 0.15 per cent. S is present. 4. Tempering twice does not influence the tensile strength and elongation. The reduction of area, the flexibility or toughness may increase.

F. Wüst ("Ueber die Theorie des Glühfrischens;" *Metallurgie,* 1908, p. 7) showed by experiment that the oxidizing reaction in the operation of malleableizing is introduced by oxygen, which forms CO_2 with the temper carbon. The CO_2 penetrates the flowing iron and reacts with the

temper-carbon, forming CO. CO reduces the iron ore to FeO or metallic iron, regenerating CO_2. If there is not enough oxygen, the re-formation of CO_2 does not take place and if the concentration of the CO is high enough, cementation may take place, by breaking up CO in CO_2+C.

Variation in Quality.—Max Orthey ("Die Chemie der Eisengiesserei;" *Metallurgie*, 1907; p. 78), showed in an article entitled "Die Bedeutung der Chemie für die Eisengiesserei," in *Metallurgie* 13; 1906, containing a great many figures, how the chemical analysis of different pig iron supplies, even if they come from the same plant, differ and that they seldom answer the specifications. The pigs are seldom selected before shipment. The present article contains further figures to show that the foundry can not be operated properly without chemical control of its pig iron. A number of analyses go to show the lack of uniformity in the composition of different pig iron supplies. Furthermore analyses of different parts of castings made of unselected and of analyzed selected pig iron are given. The author is of the opinion that the foundries could, by steady chemical control of their pig iron, induce the iron works to keep up to the specification and to supply them with a uniform product. The cost of the chemical control could not be compared with the great benefit derived herefrom.

Soundness of Castings.—Henning (*Eisenzeitung*, 1907, p. 183) proposes to mix pig iron with steel in order to get sound castings. He first produces an intermediate product consisting of 40 parts of steel and 60 parts of pig iron, melted separately and thoroughly mixed. A steel of 0.6 per cent. C. and a pig iron of 3.5 per cent. C. gave a mixture containing 2.43 per cent. C. Thirty parts of this mixture is then melted with 70 parts of pig iron and gives a final product containing 3.22 per cent. carbon.

Properties of Steel.

Relation of Carbon to Strength.—Wawrzinisk ("Die Elastischen Eigenschaften von Stahl und die Abhängigkeit derselben von der chemischen Behandlung des Materiales;" *Metallurgie*, 1907; p. 810) studied the influence of the carbon-content on the tensile strength and elastic limit (not yield point) of steels treated in the same way, and the influence of the temperature of annealing on these factors, with a small number of steel bars (14 all together). He found (with 7 bars) that in annealing at 1050 deg. C. the tensile strength and the elongation decrease with the carbon-content in about the same way as found by Campbell (*Metallurgie*, 1906, p. 741; *Journal* of the Franklin Institute, Vol. 163, p. 407). The elastic limit decreases but the modulus of elasticity (the reciprocal of the coefficient of expansion at the elastic limit) does not change noticeably. The elastic limit is considerable lower than the yield point and this is still

more pronounced if the bars are not annealed. The elongation decreases with increasing carbon content, annealed or not. No relation exists between elastic limit and carbon-content in the unannealed steel. The modulus of elasticity has a tendency to decrease with the carbon content, which confirms Benedicks' experiments. Similar experiments with steels containing 0.6 to 1 per cent. chromium (3 bars) show also that the modulus of elasticity varies slightly, in spite of their higher tensile strength, from the average of the carbon steels. The chromium content has an influence on the elastic limit, which does not decrease as much as with carbon steels. Four bars containing 0.76 to 1.13 per cent. carbon were heated to temperatures, varying from 450 to 1154 deg. C. It was found that the elastic limit is not influenced up to 700 deg. C. From 700 to 1000 deg. C. it falls and then rises again somewhat if heated above 1000 deg. C. Concerning the modulus of elasticity no definite conclusions can be reached.

Treatment for Rails.—François Limbourg (*Revue de Metallurgie*, 1907; p. 989) investigated the influence upon mechanical properties of steel due to the replacing of part of the pearlite by sorbite by heat-treatment. His results led him to recommend this treatment for rails. Stead and Richards have already proved that such a thermal treatment of rails lengthens their life from 25 to 50 per cent. and increases their cost but little.

Relation of Manufacture to Properties.—F. W. Harbord (*Journal*, Iron and Steel Institute, No. 1, 1907, pp. 181-199) showed that steels having almost the same chemical analysis differ in several physical properties according to the process by which they were manufactured, which includes the acid and basic bessemer and the acid and basic open-hearth. The carbon in the different samples varied from about 0.10 to 0.60 per cent. and curves were drawn showing the relation of strength and hardness to amount of carbon. The hardness was determined by the Brinell ball test, and the order of the curves in the diagrams was the same in each case, the acid bessemer being highest, basic bessemer next, acid open-hearth next and basic open-hearth lowest. It is stated that the curves for elastic limit, elongation and reduction of area compare with the tensile strengths. The results show that, in order to obtain the same strength or the same hardness in basic open-hearth steels as in acid bessemer steels. we must put more carbon in them, everything else being the same.

Cooling of Ingots.—H. M. Howe and B. Stoughton ("Ueber den Einfluss des Giessverfahrens auf die Lunker- und Saigerungserscheinungen nach Beobachtungen an Wachsblöcke;" *Metallurgie*, 1907, 793) have a contribution concerning the phenomenon of segregation. Wax with an addition of 0.75 to 2.5 per cent. (mostly 1.5 per cent.) oleate of copper and a trace of cerasin for color contrast are poured in ingots, varying the forms of the ingots, materials of the mold, rapidity of cooling and rapidity

of pouring. The ingots are cut longitudinally and the appearance of the cut is discussed.

Effect of Nitrogen.—J. Petrén and Alf Grabe (*Jernkontorets Annaler,* 1907, pp. 1-30) discussed in a lengthy article the occurrence of nitrogen in iron and steel. They think Hjalmar Braune's views concerning the harmful influence of nitrogen on iron and steel to be somewhat exaggerated. In their opinion a minimum content of nitrogen can not influence pig iron. They intend to make further researches concerning its influence on steel.

Thermit for Steel.—A. Oberholzer ("Zur Frage der Vermeidung von Lunkerbildung;" *Stahl und Eisen,* 1907, p. 1117) studied the effect of Goldschmidt's "Lunkerthermit" in casting steel ingots (2000 and 4000 kg.). His results favor the application of lunkerthermit, taking the added cost in consideration.

Heat Treatment of Steel.—W. Campbell ("Ueber die Wärmebehandlung von Stahlen mittleren Kohlenstoffgehaltes. Der Einfluss der Abkühlungsgeschwindegkeit auf physikalishe Eigenschaften und Structur;" *Metallurgie,* 1907, p. 772) gives data concerning heat treatment of steels with medium carbon-content and the influence of the content of carbon on their physical properties and structure. He comes to the following conclusions: In order to make the grain of a steel with 0.4 to 0.5 per cent. carbon finer, heating to Ac_2-$_3$, viz., about 750 deg. C. is required. A fine material needs a somewhat lower temperature, a coarse material (cast steel or overheated steel) a higher temperature. The lower the carbon and manganese-content the higher the refining temperature. Seven hundred and thirty-five and 785 deg. C. are safe limits. After having been heated above Ac_1 (710 deg. C.) followed by cooling in the air, the tensile strength increases and the elongation decreases. Cooling in the furnace reverses these conditions. Slow cooling gives a complete separation of the excess of ferrite and a more pronounced eutectoid (pearlite). Cooling quickly does not give a complete separation of the ferrite and transforms the eutectoid into the sorbitic form.

Quenched Steels.—P. Breuil (*Bulletin de la Société de l'Industrie Minérale,* Serie IV, Bd VI, 1907) published his experiments on the structure of quenched steels. The experiments were carried on with steels with from 0.38 to 1.8 per cent. C., a sample of chilled cast iron with 3 to 3.5 per cent. C. and cementated iron with a varying carbon-content. Upon quenching, the dimensions of the sample, the temperature of quenching, the duration of the heating, the quenching bath and the temperature of annealing were taken into consideration. The article contains a great number of microphotographs, valuable results and suggestions for further investigation. An abstract, however, can not well be given and therefore those interested in the subject are referred to the original article.

Blow Holes.—E. von Maltitz (*Transactions,* American Institute of Mining Engineers, 1907) considers the subject of blow holes in steel ingots in the light of evidence produced by H. M. Howe (ibid.), and others, and of his own researches. He finds that the gases contained in blow holes consist chiefly of hydrogen, with much smaller amounts of nitrogen, carbon monoxide being absent or else very small in amount. On the other hand, he finds that the gases given off by solidifying steel consist almost as much of carbon monoxide as of hydrogen, and sometimes more; whence he concludes that carbon monoxide is a factor in the liberation of gases occluded in molten steel. He finds confirmation for this belief in the well-known fact that superoxidized steel is wilder in the molds than that relatively free from oxides. He also concludes that ferrous oxide is more soluble in pure than in impure iron, and that its solubility increases with the temperature, so that, at very high temperatures, the affinity of iron for oxygen is even greater than the affinity of carbon for oxygen. If therefore a steel bath has absorbed ferrous oxide at high temperatures, this oxide will be decomposed by carbon when the temperature is lowered and carbon monoxide will be formed and liberated, which liberation acts mechanically to assist in the liberation of occluded hydrogen and nitrogen. The hotter the steel is when poured into the molds the more dissolved oxide will it contain, therefore the quicker will the liberation of carbon monoxide begin and the nearer to the surface will the blow holes occur, except that, if the overoxidation of the steel be corrected in the furnace, it will pour quietly and form no blow holes. Upon theoretical and experimental grounds von Maltitz disagrees with the opinion that even deep-seated blow holes will weld up in rolling or forging. He admits that they may be reduced in size, but denies that they are eliminated.

He ascribes the chief cause of blow holes to the overoxidation of the steel, which, he says, can be partially avoided in the bessemer process by not blowing too hot or too long and, in the open-hearth process, by not allowing the temperature to become too high and by maintaining the slag at the latter part of the process in a good, thin condition and with a minimum of oxidizing power. Unavoidable overoxidation can be remedied to some extent by stirring the bath with a cold iron rod, just before tapping, and by adding the deoxidizing elements (carbon and manganese) in the furnace instead of in the ladle, although this latter is expensive on account of the loss of manganese into the slag. He summarizes the means available for the prevention of blow holes as follows: Medium temperature of steel during the last period of the process; avoidance of overblowing and of the over-addition of ore; boiling out the last ore added; a fluid-finishing slag not too rich in oxygen; stirring the heat before tapping to destroy ferrous oxide; adding sufficient deoxi-

dizers, and allowing sufficient time for the separation of manganese oxide and silicates of manganese and alumina.

Effect of Phosphorus.—F. Wüst ("Beitrag zum Einfluss des Phosphors auf das System Eisen-Kohlenstoff;" *Metallurgie*, p. 73, 1908) contributes to our knowledge of the influence of phosphorus on the iron-carbon system. As a result of a thermal and metallographical investigation, he finds that the temperature at which a saturated iron-phosphorus-carbon alloy begins to freeze is lowered by the addition of phosphorus until 6.7 per cent. P is present. Every 1 per cent. P reduces this temperature on an average 27 deg. C. More than 6.7 per cent. P causes a rise of the temperature at which the alloy begins to freeze. The addition of phosphorus produces a new critical point in the cooling curve of iron-carbon alloys. This retardation occurs at 950 deg. C. and is independent upon the content of phosphorus. The duration of this critical point increases with the phosphorus-content up to 6.7 per cent. P. With further addition of phosphorus, its duration decreases again and it becomes zero when 15 per cent. P is present, which corresponds to the compound Fe_3P. The pearlite-point (710 deg. C.) is not influenced by a phosphorus-content. Its duration, however, decreases with an increasing phosphorus-content and is perhaps zero at 15 per cent. P. Addition of phosphorus diminishes the solubility of carbon in iron. The curve, which shows this relation has a nick when about 6 per cent. P is present. In cooling down saturated iron-phosphorus-carbon alloys, a ternary eutectic of approximately the composition P, 6.7 per cent.; C, 2 per cent.; Fe, 91.3 per cent. solidifies at 950 deg. C. Looked at under the microscope saturated iron-phosphorus carbon alloys show the ternary eutectic. Alloys with more than 6.7 per cent. P show an increasing quantity of free Fe_3P up to about 15 per cent. phosphorus.

Iron and Carbon.

Iron-Carbon Alloys.—Metallurgie, 4 (1907), p. 137 ("Ueber den augenblichlichen Stand unserer Kenntnisse der Erstarrungs und Erkaltungsvorgange bei Eisenkohlenstofflegierungen"), contains the second part of an article upon the present condition of our knowledge of the iron-carbon alloys. In the first part of this article (*Metallurgie*, 3, p. 175) the statement is made that we have to consider the system iron-carbon as a system iron-iron carbide and that those alloys which contain carbon as an element (it may be as graphite or temper-carbon) originated from a system containing iron carbide. The present article (second part) sets forth further considerations and experiments to support this statement and treats on the solidification of iron-carbon alloys. It is shown by experiment that for about eutectic white iron (4.08 per cent. C), the melting point and solidification point are the same. A heating curve from gray cast

iron shows the fact already known, that the melting point in this case is higher than the solidification point (1130 deg.). Two heating curves of white cast iron with 3.5 per cent. C, and with different size of grains, show that the melting point is about 1130 deg. C., independent of the size of the grains. A rough experiment shows that in heating gray cast iron (3.29 per cent. graphite and 1.43 per cent. combined carbon) up to temperatures between 1130 and 1140 deg. C., diffusion from graphite into the unmolten mass does not take place and that formation of carbide always takes place at a temperature whereat gray cast iron starts to melt. The article contains a splendid reproduction of a section of an iron-carbon alloy containing 7.52 per cent. C made in the electric furnace by Girod.

The third part of this article (*Metallurgie*, 4, p. 173) treats on transformations after solidification. A carbide-containing and a graphite-containing system respectively with 4 per cent. C, and 3.29 per cent. graphite and 1.43 per cent. combined carbon are subjected to different heat treatments and a micrographical examination is carried on. The conclusion is that the transformation of the solid solution also agrees with the assumption that systems with elementary carbon are formed from systems containing carbide.

Cementation.—Hjalmar Braun (*Bihang till Jernkontorets Annaler*, 1907, pp. 191-204) cementates iron with different kinds of coal and finds that the amount of nitrogen taken up depends greatly upon the nature of the coal. In using bone-coal powder or coal containing alkali, which favors the formation of cyanides, more nitrogen is taken up than in using graphite. Glowing of the cementated material brings the content of nitrogen down again.

Alloy Steels.

Influence of Chromium.—P. Goerens and A. Stadler ("Ueber den Einfluss des Chroms auf die Lösungsfähigheit des Eisens für Kohlenstoff und die Graphitbildung;" *Metallurgie*, 1907, p. 18) studied the influence of chromium upon the solubility of carbon in iron and also its effect upon the formation of graphite. They found that the amount of carbon necessary to saturate iron increases with its chromium-content. An alloy with 62 per cent. Cr will contain 9.2 per cent. C. Cooling-curves of the iron-chromium-carbon alloys show two critical points; one at about 710 deg. C. (pearlite-separation) which is not influenced by the amount of chromium present and the other at about 1130 deg. C. (eutectic solidification of pure cast iron) which is independent of their chromium-content up to 10.5 per cent. Cr, and increases thereafter with the amount of chromium present, being at 1535 deg. C. for an alloy with 62 per cent. Cr and 9.2 per cent. C.

By adding silicon to the iron-chromium-carbon alloys it was found that a chromium-content hampers the formation of graphite. An alloy with 2 per cent. Si and 1.5 per cent. Cr and 4.2 per cent. C does not separate any graphite. Micrographical examination showed that addition of chromium shoves the eutectic to one side and that an alloy with 5 per cent. Cr and 4.7 per cent. C (instead of 4.2 per cent. C) must be considered as a eutectic. Alloys with more chromium show, in addition to the eutectic, white crystals which are probably mixed crystals of iron- and chromium-carbide.

Boron Steel.—L. Guillet ("Les Aciers au Bore;" *Revue de Métallurgie*, 1907, p. 784) studied the influence of boron (0.215 to 1.514 per cent.) on the mechanical properties of steel with 0.18 to 0.56 per cent. C. It was found that addition of boron increases the tensile strength, especially if quenched (at 850 deg. C.) and that 0.8 per cent. boron was most effective in regard to tensile strength and elastic limit. Metallographical examination showed that (in etching with picric acid) the content of pearlite is less than would be expected, considering the carbon-content. The ferrite seems to have undergone a transformation. This fact brought the investigator to the conclusion that a solid solution of iron-boron-carbon was present. Round grains were visible; they proved to be hard, and sodium-picrate colored them brown like cementite. Their quantity increased with increasing content of boron without decreasing the pearlite. Quenching (at 850 deg. C.) made them disappear, provided the content of boron was less than 0.8 per cent. If more than 0.8 per cent. boron was present, the grains did not disappear even if the steel was quenched at 1200 deg. C. Annealing at high temperature developed the grains; cementation did not affect them and heating with iron oxides destroyed them. These grains were probably iron-boron-carbide. The results of the micrographical examination were: 1. If the boron-content is low, boron steels consist of a solid solution of iron-boron, pearlite and an iron-boron-carbide. 2. Iron-boron-carbide causes the brittleness of boron steels. 3. The brittleness disappears by quenching, the iron-boron-carbide being in solid solution.

Steel Hardening Materials.—Wilhelm Venator ("Ueber Eisenlegierungen und Metalle für die Stahl Industrie;" *Stahl und Eisen*, 1908, pp. 22, 41, 149, 225) gives a very complete summary of the metals and alloys used in the steel industry. The following alloys and metals are discussed: Spiegeleisen and ferro-manganese, ferro-silicon, ferro-manganese-silicon, silico-spiegeleisen, aluminium and ferro-aluminium, ferro-chromium, nickel, ferro-nickel and ferro-nickel-chromium, tungsten and ferro-tungsten, ferro-molybdenum, ferro-vanadium, ferro-titanium, ferro-phosphorus and carborundum. The commercial side of the subject is also taken into consideration.

Analytical Methods.

Determination of Chromium.—M. Philips ("Ueber die Bestimmung von Chrom in Chromstahl," *Stahl und Eisen*, 1907, p. 1164) reports on the determination of chromium in chrome steel. He oxidizes the solution of chrome steel in H_2SO_4 with a cold saturated solution of ammonium persulphate using a few drops of $\frac{N}{10}AgNO_3$ solution as a catalytic agent. The persulphate solution oxidizes chromium and manganese to chromic acid and permanganic acid and aids in dissolving insoluble carbide. He destroys the excess of persulphate by boiling with HCl and titrates the chromic acid with ferrosulphate and $KMnO_4$, adding a $MnSO_4$ solution containing H_3PO_4.

G. v. Knorre ("Ueber die Chrombestimmung im Stahl ins besondere bei Anwesenheit von Wolfram;" *Stahl und Eisen*, 1907, p. 1251) determines chromium in chromium-tungsten steel by the persulphate method and keeps tungsten in solution by addition of Na-phosphate to the alkaline (NaOH) solution, forming phosphortungstic acid.

Determination of Manganese.—v. Knorre ("Ueber die Mn-bestimmung bei Anwesenheit von Wolfram;" *Stahl und Eisen*, 1907, p. 380) gives his experience in determining manganese in the presence of tungsten. Lüdert showed that v. Knorre's method for the determination of manganese in iron (precipitation of manganese with ammonium persulphate, destruction of manganese persulphate by boiling, dissolving of the MnO_2 in a measured quantity of H_2O_2 and titration of the excess of H_2O_2 with $KMnO_4$) gives too high results, if tungsten is present (tungsten steel, ferro-tungsten). Von Knorre confirms this and shows that the use of a ferrous sulphate solution instead of H_2O_2 gives quantitative results. He shows, further, that the presence of molybdenum does not interfere.

Determination of Nickel.—O. Brunck ("Die Bestimmung des Nickels im Nickelstahl;" *Stahl und Eisen*, 1908, p. 331) recommends his method for the determination of nickel (precipitating it with dimethylglyoxim and weighing the voluminous precipitate dried at 110 to 120 deg. C.) also for the determination of nickel in nickel steel and chromium-nickel steel. He keeps the iron in solution, forming a complex iron salt by the addition of tartaric acid. A number of analyses show that the results are satisfactory. The only disadvantage of the method is the high price of dimethylglyoxim. This reagent can be regenerated, however.

Determination of Nitrogen.—Pretrén and Grabe (*Jernkont. Annaler*, 1907, p. 27) recommend a colorimetric and an iodometric method of determining nitrogen in iron and steel. Both methods dissolve the iron in HCl, add NaOH and distil all the NH_3 over. The nitrogen is then determined colorimetrically with Nessler's solution or iodometrically by

titration of the excess of a N/20 H_2SO_4, solution using the following reaction $KIO_3+5I+3H_2SO_4=6I+3K_2SO_4+3H_2O$.

Determination of Phosphorus.—M. Frank and F. Willy Hinrichsen ("Ueber die phosphorbestimmung in Stahl;" *Stahl und Eisen,* 1908; p. 205) noticed that the results acquired by different operators in determining the phosphorus-content of a great number of steels, using the molybdate method, differed considerably. They found that this was due to the presence of arsenic which causes high results. Pure arsenic acid solutions are not precipitated with ammonmolybdate solution. Arsenic is therefore carried down with phosphorus. In steels containing as high as 0.05 per cent., as the error caused by the co-precipitation of the arsenic with the phosphorus does not surpass 0.015 per cent. Specifications which do not include arsenic should consider this. The co-precipitation of arsenic is favored by NH_4Cl and reduced by the presence of free HCl and to a less extent by an excess of ammonium-molybdate.

Determination of Sulphur.—H. Kinder ("Schwefelbestimmung in Eisen und Stahl;" *Stahl und Eisen,* 1908; 249) report on the results acquired by the chemist commission of the "Verein deutscher Eisenhüttenleute" in comparing different methods of determining sulphur in iron and steel. In case of umpire analysis the barium-sulphate method is proposed as the normal method. A full description for its running is given. As a rapid method the cadmium sulphide method, titrating the cadmium sulphide with iodine, is recommended. This method is also minutely described. In both methods pure conc. HCl (sp.g., 1.19) must be used. The combustion-tube of Campredon and Schulte (*Stahl und Eisen,* 1897, p. 436 and 1896, p. 865) can be omitted.

Determination of Tungsten.—F. Willy Hinrichsen ("Ueber die Bestimmung von Wolfram im Stahl bei Gegenwart von Chrom," *Stahl und Eisen,* 1907, p. 1418) finds that the determination of tungsten with benzidinchlorhydrate does not give good results if chromium is present. He confirms, however, that Knorre's determination of chromium is not influenced by presence of tungsten. He recommends precipitation of chromium and tungsten as mercurous salt, determination of Cr_2O_3 and WO_3 together and thereafter determination of Cr_2O_3 in the mixture. He shows that this method gives reliable results.

LEAD.

By W. R. Ingalls.

The production of refined lead in the United States in 1907, together with the corresponding figures for 1906, is given in an accompanying table, based on reports received directly from the smelters and refiners, in which the old classification has been adhered to, although it is now incorrect and meaningless, but it is not easy to suggest an accurate one. The term "soft lead" formerly applied to the lead produced in the Mississippi valley was never of any special significance. The antithesis of soft lead is hard lead, and antimonial lead is the only really hard lead, although some of the lead produced in Missouri and called "soft" figures also in the market as "chemical hard." Desilverized lead is soft lead, and the doubly refined desilverized, known as "corroding lead," is the highest grade of the metal, but electrolytically refined lead is fully its peer.

STATISTICS OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Produced from Domestic Ores.				Imported in Ores and Bullion. (c)	Total Production and Imports.	Exported in all Forms.
	Desilverized.	Soft. (a)	Antimonial. (b)	Totals.			
1897	144,649	45,710	7,359	197,718	92,117	302,859	60,353
1898	169,364	50,468	8,643	228,475	89,209	348,845	78,168
1899	171,495	40,508	7,377	217,085	76,423	317,196	74,944
1900	221,278	47,923	9,906	279,107	114,397	425,824	100,288
1901	211,368	57,898	10,656	279,922	112,471	458,033	100,026
1902	199,615	70,424	10,485	280,524	107,715	458,456	82,228
1903	188,943	78,298	9,453	276,694	106,407	418,601	81,971
1904	200,858	90,470	10,876	302,204	112,852	415,056	84,142
1905	205,665	105,623	11,186	322,474	98,378	420,852	59,741
1906	220,095	118,000	10,120	348,215	84,134	432,349	47,323
1907	213,383	127,133	9,614	350,130	79,815	429,945	51,502

(a) Since 1904 a large part of the so-called soft lead was desilverized, but this (being of Missouri origin) has been included in the old classification. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin. In 1905, the first year in which the separation has been possible, the antimonial lead from foreign sources amounted to 2730 tons. In 1906 the antimonial lead from foreign sources was 2686 tons. (c) Includes "pigs, bars and old."

METALLURGICAL PRODUCTION OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Domestic Origin.						Foreign Origin.		Grand Total.
	Desilverized	Antimonial	S. E. Mo.	S. W. Mo.	Miscel.	Total.	Desilverized	Antimonial	
1905	205,665	8,456	81,299	21,324	3,000	319,744	83,504	2,730	405,978
1906	220,095	7,434	100,492	16,528	980	345,529	67,441	2,686	415,656
1907	213,383	9,614	108,510	17,833	790	350,130	76,016	(a)	426,146

(a) The production of antimonial lead, which in 1906 was divided according to domestic and foreign, in 1907 has been wholly entered under domestic.

The refiners who produced the lead entered as desilverized in the table did so mainly from Western work-lead, but they bought considerable quantities of non-argentiferous lead ore from Missouri and elsewhere. On the other hand, one of the refiners whose product is entered under "Southeast Missouri," turned it all out as desilverized of exceptionally high grade, and moreover, smelted not only ore from Missouri and adjacent States, but also purchased considerable supplies in the far West. There were in 1907 six companies, operating nine refineries, which produced desilverized lead. At eight of these works the Parkes process is employed; at one the Betts electrolytic process.

In computing the statistics of lead production in the United States there is more or less the same difference as in the case of copper with respect to refined lead and crude lead, the latter being most nearly representative of the production of the mines. I am unable to give statistics compiled strictly on the basis of crude lead, but another table showing the distribution according to the States of origin, is prepared partly on that basis and partly on the basis of refined lead, and its total may be considered most nearly representative of the production of the mines of the United States in 1907. It will be observed that the total in the table of the production by States does not agree with the total in the first table of refined lead production, nor should it, inasmuch as it represents a different thing.

CONSUMPTION OF LEAD IN THE UNITED STATES.

(In tons of 2000 lb.)

	1905	1906	1907
Supply:			
Production desilverized	289,169	287,536	289,399
Production soft lead	105,623	118,000	127,133
Production antimonial lead	11,186	10,120	9,614
Imports foreign refined lead	5,720	11,763	9,277
Stock, domestic lead, Jan. 1	10,000	4,000	4,000
Foreign in bond, Jan. 1	11,481	8,148	5,691
Total supply	433,179	439,567	445,114

	1905	1906	1907
Deductions:			
Re-exports of foreign	58,631	47,223	51,424
Exports of domestic lead	63	74	55
Stock, domestic, Dec. 31	4,000	4,000	50,000
Foreign in bond, Dec. 31	8,148	5,691	12,897
Total deductions	70,842	56,988	114,376
Apparent consumption	362,331	380,122	330,738

IMPORTS OF LEAD IN ORE, BASE BULLION, PIGS, BARS AND OLD. (*a*)

(In tons of 2000 lb.)

Source.	1901	1902	1903	1904	1905	1906	1907
United Kingdom	201.3	396.3	776.4	247.3	795.0	4,926.4	217.0
Germany	335.6	476.4	704.9	365.6	125.1	1,003.2	228.3
Other Europe	1.2	671.1	225.7	82.8	58.8	1,960.5	3,461.1
Canada	26,065.0	9,732.5	9,600.4	8,951.9	8,181.5	9,257.1	6,636.5
Mexico	81,726.8	93,742.3	93,068.3	102,903.0	87,583.8	66,756.3	68,767.5
South America	4,108.7	2,690.1	1,947.8	290.0	1,577.2	157.6	441.7
Other Countries	32.5	6.1	83.2	11.0	56.3	73.7	62.9
Total	112,471.1	107,714.8	106,406.7	112,851.7	98,377.7	84,134.8	79,815.0

(*a*) Refined lead, i e., in pigs, bars and old, is a small part of the total. It was in 1901, 604 tons; 1902, 2,529 tons; 1903, 3,023 tons; 1904, 8,724 tons; 1905, 5,720 tons; 1906, 11,763 tons; 1907, 9,277 tons.

PRODUCTION OF LEAD BY STATES.

(In tons of 2000 lb.)

State	(*a*) 901	(*a*) 1902	(*b*) 1903	(*b*) 1904	(*d*) 1905	1906	1907
Colorado	73,265	51,833	43,276	49,290	(*f*) 57,856	(*f*) 52,992	47,332
Idaho	79,654	84,742	94,611	103,411	(*h*)107,000	(*i*) 121,584	111,697
Missouri-Kansas	(*c*)67,172	(*c*)79,445	(*c*)86,439	(*c*)92,119	(*g*)102,500	(*k*)115,103	125,813
Montana	5,791	4,438	3,138	3,454	(*e*) 3,500	(*k*) 2,485	2,005
Utah	49,870	53,914	48,573	53,647	44,500	(*k*) 56,268	54,738
Others	8,452	6,425	6,365	5,283	(*e*) 4,900	(*l*) 6,877	(*m*) 10,652
Totals	284,204	280,797	282,204	307,204	320,256	355,309	352,237

(*a*) Statistics of the U. S. Geological Survey, representing "lead content of ore smelted." (*b*) Statistics of the U. S. Geological Survey, representing production of merchant lead. (*c*) Includes also the production of Wisconsin, Illinois, Iowa, Virginia and Kentucky. (*d*) Production of crude lead, distributed according to States of origin. (*e*) Estimated. (*f*) Report of State Commissioner of Mines. (*g*) Includes 1500 tons from Iowa, Illinois and Wisconsin; this may not represent the total production of these States, anything in excess appearing under the classification of "Others." (*h*) Partly estimated. (*i*) Report of State Inspector of Mines, less allowance of 5 per cent. for loss in smelting. (*k*) Smelters' reports. (*l*) Chiefly the production of Wisconsin, Arizona and Nevada. (*m*) Includes Arizona, Arkansas, California, Kentucky, Illinois, Iowa, New Mexico, Nevada, Oklahoma, Oregon, Tennessee, Texas, Washington and Wisconsin.

The totals in the table of distribution by States are larger both for 1906 and 1907 than the totals for the first table. This is to be explained by the loss that is experienced in refining the work-lead and also by changes in the amount of work-lead in transit and in stock. Thus certain smelters reported a greater output of work-lead than the refiners reported as received from them. Such differences explain why there was a small increase in the domestic production of refined lead in 1907, while in the same year there was a small decrease in the production based on the figures of smelters which are nearer to the mines. It may be remarked that the total for Idaho agrees very closely with the output of lead in ore reported by the mines of that State, allowing 5 per cent. for loss in smelting.

The lead industry of the United States suffered keenly from the effects of the recession in business, which became manifest during the second quarter of the year. By June the trade had become very sluggish, and repeated reductions of price failed to bring about improvement, in spite of strenuous efforts to enforce a curtailment of production. The largest

independent producers in Idaho and Missouri refused to make any restriction; the American Smelting and Refining Company and the United States Smelting, Refining and Mining Company succeeded in effecting some reduction in output by discouraging the delivery of lead ore; the Federal Lead Company, of Missouri, reduced its production. But in spite of these measures the stocks of refined metal, about 4000 tons at the beginning of 1907, increased, especially since the mid-year, until at the middle of November they amounted to 25,000 to 30,000 tons, and at the end of the year were estimated at 50,000 tons. I adopt the latter figure in computing the consumption of lead in the United States in 1907.

The slackening in the demand for lead created new conditions in the market. For several years upward of 90 per cent. of the lead produced from domestic ores had been refined and sold by three interests, chief among which is the American Smelting and Refining Company. The market price had been practically controlled by the latter, inasmuch as the price fixed by it was adopted by the other large producers. A comparatively small amount of lead was sold during 1906 by independent dealers at prices higher than those of the trust, but the total volume of this business was insignificant in comparison with the great production marketed on the established terms. The trust showed great moderation at that time in not marking up prices when it might easily have done so.

It has been fully recognized, however, from the beginning that the apparently successful regulation of the price of lead by the American Smelting and Refining Company was based on a generally rising market; and that the test of its control would develop on a falling market. This was proved by the experience of the second half of 1907. When the demand became slack, independent producers, who previously had been selling at the trust price, began to undersell, and manufacturers whose contracts expired decided to take their chances in the open market, so that the latter acquired a magnitude greater than for many years. By September the open market had become the one in which competition was active and wherein was transacted the bulk of the business not previously arranged by contract.

Consequently there were two markets for lead, viz., the trust market, in which deliveries were made under contract at the selling price fixed by the trust; and the open market in which wholesale trading was done in the ordinary way. The difference in prices between the two markets steadily increased under the pressure of the independents to sell their product until it stood about $\frac{1}{4}$c. per lb. below the trust price. Finally, in November, the American Smelting and Refining Company abandoned the establishment of card prices and entered the general market in competition with the other producers.

REVIEW OF LEAD MINING IN THE UNITED STATES.

Arizona.—The lead production of this Territory comes from a variety of sources. There is no silver-lead smelting works in the Territory, the ore being shipped chiefly to the Arizona-Mexican Mining and Smelting Company, of Needles, Cal., and to the American Smelting and Refining Company, at El Paso, Tex.

Colorado.—The lead production of this State showed a considerable decrease in 1907, chiefly in the output of Leadville, which produced only 17,032 tons against 23,918 tons in 1906. Aspen and Creede also showed decreases. The lead production of Colorado is reported by the State commissioner of mines at a slightly lower figure than mine, his figure being 46,493 tons. My figure includes the lead obtained from zinc-smelting residues received from Pueblo, Colo., and Iola, Kan., the origin of which is partly Colorado and partly other States. The chief producer of lead in Colorado is the American Smelting and Refining Company, operating one works at Denver, two at Pueblo, and one each at Leadville and Durango. According to information furnished to the Denver newspapers, these works in 1907 treated 938,083 tons of ore, containing 496,065 oz. of gold, 14,636,634 oz. of silver, 94,145 tons of lead, and 10,333,130 lb. of copper. The gold was derived chiefly from Colorado, but large quantities of gold-bearing ore were received from Nevada. The origin of the silver was chiefly Colorado, as was also the case with copper. The lead came chiefly from Idaho. Besides the American Smelting and Refining Company the only other producer of base bullion in Colorado is the Ohio & Colorado Smelting Company, of Salida.

LEAD PRODUCTION OF COLORADO. (*a*)
(In tons of 2000 lb.)

Country.	1900	1901	1902	1903	1904	1905	1906	1907
Clear Creek	2,497	1,945	1,641	1,726	1,981	1,631	1,439	1,832
Hinsdale	4,089	3,705	3,107	230	521	446	442	470
Lake	31,300	28,180	19,725	18,177	23,590	26,424	23,918	17,032
Mineral	7,476	5,260	4,646	4,300	6,673	5,940	7,443	6,490
Ouray	4,739	3,952	2,131	1,675	1,022	2,674	2,861	1,803
Pitkin	13,726	16,375	12,487	16,635	9,441	10,987	8,781	6,957
San Juan	8,789	7,736	3,850	3,485	4,644	3,223	2,070	6,213
Others	8,921	6,813	5,565	4,529	5,901	6,531	6,038	5,696
Total	82,137	74,056	53,152	50,757	53,773	57,856	52,992	46,493

(*a*) As reported by the State Commissioner of Mines.

Idaho.—The production of lead in this State showed a large decrease in 1907, owing chiefly to the curtailment by the Federal Mining and Smelting Company in the second half of the year. As in previous years, the bulk of the output of Idaho was derived from the Cœur d'Alene, where the Bunker Hill & Sullivan Mining and Concentrating Company, the Federal

Mining and Smelting Company, the Hecla Mining Company, and the Hercules mine are the principal producers. But little of the lead ore produced in Idaho is smelted in the State, it being shipped chiefly to works in Colorado, Utah and Nebraska, while a small quantity is shipped to Pittsburg, Penn., and Newark, N. J. The Panhandle Mining and Smelting Company, at Ponderay, treats a small quantity of ore, receiving supplies from adjacent States and from British Columbia.

(By Stanley A. Easton.) Returns from all the large producers of the Cœur d'Alene show an output in 1907 of 217,756,499 ℔. of lead, 6,882,743 oz. silver and 6,984,724 ℔. of copper. A careful estimate of the production from unreported sources added to these figures gives a total yield from the district during the year of 220,256,499 ℔. of lead, 6,945,343 oz. silver, 6,984,724 ℔. copper, and not less than 4,500,000 ℔. of zinc. All the copper came from the Snowstorm mine and all the zinc came from the Success (formerly the Granite). The mineral products of the Cœur d'Alene district, other than lead, are important and valuable, but the district is famous for its lead output, which is worth more than the combined value of all the other products many times over.

From January to June 4, 1907, nearly one-half the year, payment for lead was based on the New York price of 6c. per pound, a high-water line in lead prices, and freight and treatment charges on these ores were never lower. This condition, although cars for shipments were hard to get and operating difficulties of every kind interfered, resulted in large earnings and great activity. From 6c., on June 4, the New York price of lead fell away to 3½c. at the close of the year. By a careful estimate, it is determined that 61 per cent. of the year's lead was produced during the first six months and 39 per cent. from the last six months, indicating that the decline in yield, compared to preceding years and compared to the total yield of the country, is directly due to market and not to mine conditions. This fact is more strongly established by a comparison of the production of the last three months, when the lead price was lowest, with either of the first two quarters of the year.

The development accomplished by all the mines during 1907 is uniformly reported to have given very satisfactory results. The immediate outlook is such that the present resources of the district can maintain its past output and importance with no assistance from new discoveries. During 1907 no new sources of lead added appreciably to the yield. Several properties, at present in an incipient shipping stage, may help the production during 1908, but their output is disregarded in summarizing the prospects for the immediate future.

(By Robert N. Bell.) The development of lead ore outside of the Cœur d'Alene, notably in Lemhi and Blaine counties, received considerable attention in 1907; some important developments were made which

indicate that the production from these sources will become larger. At Bellevue in the Wood River district, considerable concentrating ore was developed by the Idaho Consolidated Mining and Development Company along the Minnie Moore contact; a 250-ton mill is being built to treat this ore, half the capacity of which embraces the Sutton-Steele dry method of concentration.

Another important lead-ore development was made at the Crœsus mine, three miles west of Hailey, on Wood river, where a mill of 250 tons per day capacity will very likely be erected during 1908. At the old Muldoon mine, 25 miles east of Hailey, a quantity of high-grade concentrating lead ore was developed and the machinery for a large milling plant was ordered. These and other mills of the Wood River district will give a daily crushing capacity of 1000 tons; from present indications these three new plants will be ready for operation by the fall of 1908. The lead ores of the Wood River district are invariably rich in silver and also carry some gold. It seems likely that within another year this old district will again produce considerable lead and silver. Several of the larger orebodies are rich in zinc and arrangements are being made to produce a clean zinc concentrate.

In Lemhi county the Gilmore mine made important discoveries. This property shipped 40 cars of 50 per cent. ore during 1907; much high-grade ore was developed in the lower levels. The principal ore-shoot is about 125 ft. long by 12 ft. wide, and contains an average of 35 per cent. lead and about 18 oz. silver per ton. The mine is developed to a depth of 380 ft. by a vertical shaft; the ore at this depth is still completely oxidized, consisting of a soft mixture of lead carbonate and brown oxides of iron and manganese, a combination which makes a desirable smelting ore.

In the same vicinity two other mines, the Lemhi Union and the Leadville, which have just begun to produce, shipped about six cars of ore each. At the Lemhi Union mine the ore closely resembles that of the Gilmore mine and is found under similar conditions; at the Leadville mine, the ore occurs near a contact of blue Carboniferous limestone and quartz porphyry. The ore of this property carries a small percentage of antimony and consists of a blackish steel galena and yellowish-gray carbonate ore. The crude-ore shipments averaged 55 per cent. lead with 45 oz. silver and $2 gold per ton. The district in which this mine is situated is an extensive one, with many indications of lead-silver and copper ores and flat dipping contacts of blue Carboniferous limestone and shale and quartz porphyry, somewhat resembling Leadville, Colo.

Iowa (By James H. Lee).—The year 1907 was devoted more to prospecting and developing than to production. There has been a great impetus given to the lead mining industry during the last two years and numbers of new prospects have been opened up in addition to the reopening and

development of old workings. Some of these latter had been abandoned after the extraction of the richer ore or after the water level had been reached, but improved mining methods and machinery have made deeper workings possible and a profitable industry is now assured.

As many of the mines yield both lead and zinc ore it is difficult to classify these or to separate the industry into its component parts. In this connection, the statement of Prof. T. C. Chamberlain, made over twenty-five years ago, may be cited here as showing his insight into conditions the existence of which has been proved by the development of the last year. He stated that in the middle and lower portions of the Galena limestone, where lead has been found at higher horizons, it is altogether probable that mixed lead and zinc ores occur, and the more recent deeper workings of old mines have shown that in many cases at least this is exactly the situation.

The mining industry centers at Dubuque and most of the large producers are situated within or near this city. There are over seventy prospects and mines in this district and among these are twelve important lead and zinc producers. In addition there are fifteen which will be producing before June 1st of the current year. These various producers sold 850,000 pounds of lead during 1907, and the fact that during the first six weeks of the present year more lead and zinc was shipped than during the entire year of 1907 augurs very well for the future of the industry.

Kentucky.—Lead ore occurs in connection with zinc ore in the vicinity of Marion; also in Owen and Gratz counties, about 50 miles from Cincinnati. The lead deposits of northern Kentucky, which are apparently of but little importance, were described by R. B. Brinsmade in *Eng. and Min. Journ.*, Apr. 6, 1907.

Missouri and Kansas.—The lead production of these States showed a large increase in 1907, owing chiefly to the greater activity at Bonne Terre and Flat River, Mo. The Federal Lead Company curtailed production during the second half of the year, but the St. Joseph Lead Company and the St. Louis Smelting and Refining Company continued operations in full force almost to the end, when the St. Joseph reduced to one-half capacity, but only for two or three weeks. As in 1906, the bulk of the production of southeastern Missouri was made by the smelters at Collinsville and Alton, Ill., and at Herculaneum, Mo. A comparatively small quantity of Missouri ore was shipped to Newark, N. J.

(By H. A. Wheeler.) The output in 1907 of the southeastern Missouri lead district is estimated at 98,000 short tons, approximating $10,150,000 in value. This is the largest in its history, as it exceeds the banner output of 1906 by 11 per cent., notwithstanding that production was heavily curtailed after the panic. The Flat River and Bonne Terre camps in St. François county as usual not only produced almost this entire tonnage,

but also the increase was entirely derived from this source. These camps furnished about 94 per cent. of the output, while the Madison county mines that center about Fredericktown supplied about 5 per cent., and the outlying counties of Washington, Jefferson and Franklin produced about 1 per cent.

The ore produced in Washington, Jefferson and Franklin counties is still derived from innumerable shallow diggings worked on the leasing system. Some efforts were made in 1907 to test the deep ground under the shallow mines in Washington county, but the financial stringency brought the prospecting to a standstill before tangible results were obtained, although considerable encouragement was met. When it is considered that for more than 70 years the lead production of St. François and Madison counties was derived exclusively from similar small, shallow deposits, it is surprising that greater effort has not been made to prospect the other shallow lead camps. It is true that the base of the Bonne Terre limestone, in which the large bodies of disseminated ore occur, is much deeper in Washington, Jefferson and Franklin counties, so that the drilling must be from 800 to 1200 ft. deep as compared with 100 to 800 ft. in St François and Madison counties. It is also true that during the first 20 years after the disseminated lead was discovered by drilling, the holes rarely exceeded 200 ft. in depth, and it is only recently that drilling has been carried to the depths of 500 to 800 ft. Yet what is today probably the finest mine in the district is one of the recent discoveries made at a depth of 500 ft., and two shafts are now being sunk to orebodies lying more than 700 ft. deep. Since the large, deep orebodies of St. François and Madison counties are found to be intimately associated with the shallow lead deposits, it will be remarkable if the richer and more numerous shallow deposits of Washington county are not similarly underlaid with the disseminated orebodies. It is certainly an attractive field for the courageous prospector as the low price of lands in Washington county that have been large producers of shallow lead offers a very handsome profit if intelligent drilling discovers the deeper deposits that geological evidence so strongly indicates.

The past year opened auspiciously, but at the end the output was reduced by most of the companies until it was only about 50 per cent. of the capacity of the district. Wages were partially reduced from the abnormally high level that prevailed in 1907, but no doubt a lower wage will shortly become universal and the district will again enjoy its old reputation of being one of the lowest labor markets in America. The miners' union will also probably be a disappearing factor, for with the large number of idle men who are now willing to work for anything they can get, it will be an easy matter to cull out the labor agitators, while 90 per cent. of the men will be only too glad to withdraw from the union.

The absorption of the smaller companies made further progress in 1907 and 90 per cent. of the production now rests with two powerful interests. During the prosperous period the large companies did considerable prospecting, especially on lands held under option. Several of the options were exercised and the large acreage held by the two prominent interests was considerably increased. Nearly 100 diamond-drills were at work until Sept. 1, which is the largest number in the history of the district; but all this work has been stopped and the drills housed.

One of the important events during 1907 was the completion of the mill which the Federal Lead Company has built on its Central tract. The building is of steel and the mill consists of six 400-ton units, giving a total capacity of 2400 tons. It is said to have cost $1,250,000. It is equipped with a fine independent crushing and sampling department and the storage bins have a three-days capacity. The mill is operated by electricity furnished from three Curtis turbine generators of 500 kw. each.

The St. Joseph Lead Company still holds its prestige of being the largest producer of the district; its No. 1 mine is over 40 years old. While no new shafts were sunk in 1907, its plants were greatly improved in capacity and efficiency and Bonne Terre, which the company monopolizes, has been made a model mining camp. The new No. 13 shaft at Bonne Terre was equipped with an electric hoist and is now an important producer. Nos. 11 and 12 shafts on Big river are now able to keep the new 1500-ton mill supplied with plenty of ore and a well-built town, called Leadwood, has grown up near the No. 12, or Hoffman shaft. The smelter at Herculaneum, 30 miles north of Bonne Terre, was greatly improved in capacity and efficiency. Two large, modern furnaces have replaced the old small, round stacks and mechanical feeding has superseded hand feeding. The Savelsberg pot-roasting process is now used for roasting the ore and matte and a marked economy has resulted over the Freiberg hand roasters; the new plant consists of five roasting pots. The Hill and Gumbo shafts were shut down in October; in December the production was cut in half by putting the employees on half time.

The Doe Run Lead Company completed the new shaft on the Mitchell tract and re-opened the old abandoned shaft at Flat River station that was flooded 17 years ago; the latter was recently found to be dry, the adjacent mines having drained it. The ore is still hauled 10 miles south to the mill at Doe Run, which is a 1200-ton plant. A large, modern concentrator will be built at Elvins to treat the ore from the Mitchell tract. The Doe Run company took an option on the Union Lead Company's tract, found a rich orebody and purchased the property. The company also purchased the tract belonging to the Columbia Lead Company; the No. 2 shaft and 300-ton mill have been overhauled and are again producing.

These two acquisitions added about 1000 acres to the company's extensive holdings at a cost of about $1,000,000.

The Desloge Lead Company operated its mill at full capacity throughout 1907. This company was the first to reduce wages.

The National Lead Company completed its No. 5 shaft at Esther and it is now an active producer. The mill is being reconstructed on lines that should materially increase its efficiency. The Hancock jig is replacing the plunger jigs; Huntington mills are being installed to grind the middlings finer and the number of tables and vanners in the slime department is to be increased. The Richards classifier will be used. Electric haulage has replaced mules underground.

The Federal Lead Company completed its new Central mill, which is the largest concentrator in the Mississippi valley. The old 400-ton Central mill has been torn down and the old Central smelter has been dismantled. The No. 8 shaft has been completed on the 40-acre tract in Flat River, while No. 9 and No. 10 shafts on the old Missouri Lead Fields tract were being sunk to an orebody that is over 700 ft. deep.

The Madison county output in 1907 was less than in 1906, as the principal producer, the North American Lead Company, has gone into the production of copper, nickel and cobalt and produces only a small amount of lead as a by-product. A $250,000 smelter has been completed, that is, equipped with an electrolytic refining plant. The ore assays 4 to 7 per cent. copper. The other mines of Madison county, the Madison (formerly the Catherine), the Hudson Valley and the Mine la Motte, were actively operated during 1907, but labor shortage interfered with a full output until near the end of the year.

As the Madison Lead Company has enlarged its plant and as the Mine la Motte is adding a new 500-ton mill, the Madison County output should be considerably larger in 1908.

(By Jesse A. Zook.) The shipments of lead ore from the Joplin district in 1907 amounted to 41,742 tons, valued at $2,898,404, against 39,189 tons in 1906. The production of lead ore in this district is made in connection with zinc ore, and generally rises and falls according to the output of the latter. Production of both ores in 1907 was stimulated by a high range of prices during the first three-quarters of the year. In the first two weeks of January the price for lead ore was $87, the next week $87.50, then two weeks at $88, then $88.50, $88.25, back to $88, then $86, $85 for four weeks, $84 one week, $82 for five weeks, then $83 the last week of May and $83.50 the first week of June, and the remainder of the month teetered up and down, $75.50, $78, $74, and closing at $76. The first week in July it was cut $20 per ton, opening the month at $56, up to $58, $62 and closing July at $64. The first week of August it was $62, then $64.50, $65, dropping to $62 the last two weeks. September opened at $61, and the next three

weeks it was $54. All of October the price rested very steadilv at $55, dropping the first week of November to $51.50, the next week to $51, then $48, $46, $44, opening December at $42 and dropping to $40.

PRODUCTION OF LEAD ORE IN THE JOPLIN DISTRICT.

(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1894........	32,190	1897....	30,105	1900....	29,132	1903....	28,656	1906	39,189
1895........	31,294	1898....	26,687	1901....	35,177	1904....	34,362	1907	42,065
1896........	27,721	1899....	23,888	1902....	31,625	1905....	31,679	....	....

PRICE OF LEAD ORE AT JOPLIN.

(Per ton of 2000 lb.)

Year.	1899	1900	1901	1902	1903	1904	1905	1906	1907
Highest......	$55.00	$56.50	$47.50	$50.00	$60.50	$62.00	$80.00	$87.00	$88.50
Average......	51.34	48.32	45.99	46.10	54.12	54.80	62.12	77.78	68.90

Nevada.—The increase in the lead production of this State in 1907 was due chiefly to the Richmond-Eureka Mining Company, of Eureka, which during the summer and early fall was shipping as high as 200 tons per day of ore averaging about 3½ per cent. lead and 2 to 3 oz. silver per ton. When the production of lead was curtailed by the United States Smelting, Refining and Mining Company, late in the fall, the mines at Eureka were closed. Outside of Eureka, mines at Hamilton, White Pine county, were small producers.

Utah.—The production of lead in this State decreased in 1907 owing to curtailment by the mines and smelters. Early in October the United States Smelting Company requested its clients to reduce shipments of lead ore, while the American Smelting and Refining Company pursued the same policy. Under normal conditions, the Murray plant of the American Smelting and Refining Company produces about 4000 tons of base bullion per month, while the United States company produces about 3000 tons per month. The Murray plant has eight furnaces, each capable of treating 200 tons of charge per day, while the United States company has six furnaces which treat 170 tons of charge per furnace per day. The Tintic Smelting Company, a new concern, was engaged in 1907 in erecting a lead smelting plant in the Tintic district.

Wisconsin.—The production of lead ore in the Wisconsin district, which includes the northwestern corner of Illinois, and the region in the vicinity of Dubuque, Iowa, increased in 1907 along with the increase in zinc ore production, which was natural, inasmuch as zinc ore and lead ore occur in close association in this district.

LEAD PRODUCTION OF THE WORLD. (*a*)

(In metric tons.)

Year.	Australasia.	Austria.	Belgium.	Canada.	Chile.	France.	Germany.	Greece.	Hungary.
1897......	22,000	9,860	17,023	17,698	370	9,916	118,881	16,468	2,527
1898......	67,000	10,340	19,330	14,477	13	10,920	132,742	19,193	2,305
1899......	87,600	9,736	15,700	9,917	171	15,981	129,225	19,059	2,166
1900......	87,100	10,650	16,865	28,654	14	15,210	121,513	16,396	2,030
1901......	90,000	10,161	18,760	23,452	455	21,000	123,098	17,644	2,029
1902......	90,000	11,300	19,504	11,478	(*e*)500	18,817	140,331	15,668	2,243
1903......	141,446	12,162	22,263	9,070	71	23,258	145,319	16,093	2,057
1904......	118,979	12,645	23,470	19,000	17	18,000	137,580	14,320	2,104
1905......	(*c*)104,639	12,968	22,885	25,650	Nil.	24,100	152,590	13,729	2,146
1906......	(*d*) 93,000	14,846	23,765	24,560	(*e*) 50	25,600	150,741	12,308	1,925
1907......	(*d*) 97,000	(*fd*)15,400	(*d*)25,800	21,571	....	(*d*)23,000	142,571	(*d*)13,800	(*g*)

Year.	Italy.	Japan.	Mexico.	Russia.	Spain.	Sweden.	United Kingdom. Foreign Ores.	United Kingdom. Domestic Ores.	United States.	Totals. (*h*)
1897...	22,407	1,737	71,637	450	189,216	1,480	13,312	26,988	179,369	721,167
1898...	24,543	1,705	71,442	241	198,392	1,559	23,239	25,761	207,271	798,615
1899...	20,543	1,989	84,656	322	184,007	1,606	17,571	23,929	196,938	820,873
1900...	23,763	1,877	63,827	221	176,600	1,424	10,738	24,762	253,204	854,407
1901...	25,796	1,806	94,194	156	169,294	988	19,639	20,361	253,944	892,861
1902...	26,494	1,644	106,805	225	177,560	842	9,113	17,987	254,489	926,870
1903...	22,126	1,728	(*b*) 94,181	106	175,109	678	14,900	19,958	250,992	951,517
1904...	23,475	1,803	(*e*)103,000	90	185,862	589	6,888	19,838	274,132	961,792
1905...	19,077	2,255	(*e*) 87,393	700	185,693	576	(*e*) 5,000	20,646	292,519	972,496
1906...	21,268	(*d*)4,000	(*e*) 66,610	907	185.470	(*e*) 600	(*e*) 4,000	22.693	315,831	968,174
1907...	(*d*)22,900	(*e*)3,500	(*e*) 72,000	(*e*) 100	(*d*)185,800	(*e*) 700	(*e*) 4,000	(*d*)20,000	317,568	964,910

(*a*) From official reports of the respective countries, except the United States and as otherwise specified in these foot-notes. (*b*) Net exports. (*c*) Commercial statistics of Julius Matton, London. (*d*) As reported by the Metallgesellschaft, Frankfurt am Main. (*e*) Estimated. (*f*) Including Hungary. (*g*) Included with Austria. (*h*) The totals are somewhat too high, because of certain duplications which it is impossible to eliminate.

Lead Mining in Foreign Countries.

Australia.—Among the lead-producing countries of the world, Australia, or more specifically New South Wales, ranks fourth. In 1907 its output showed an important increase. It is chiefly derived from Broken Hill, which as a district is surpassed only by the Cœur d'Alene and Bonne Terre-Flat River in the United States.

PIG LEAD PRODUCTION OF AUSTRALIAN SMELTING WORKS. (*a*)

(In tons of 2240 lb.)

Works.	1903	1904	1905	1906
Broken Hill Proprietary Co.	61,375	68,513	67,062	55,892
Sulphide Corporation	15,680	23,094	22,246	21,033
Smg. and Ref. Co. of Australia	7,500	10,599	502	....
Tasmanian Smelting Co.	6,800	7,800	9,000	8,900
Fremantle Smelter	250	5,053	2,104	2,850
Queensland	3,795	2,046	2,422	2.461
Totals	95,400	117,105	103,336	91,141

(*a*) As reported by Julius Matton of London

LEAD PRODUCTION OF NEW SOUTH WALES. (a)
(In tons of 2240 lb.)

Lead.	1903	1904	1905	1906	1907
Base bullion ..	92,293	106,038	93,182	79,925	79,870
In ore exported	29,706	59,507	69,044	58,683	111,830
Totals......	121,999	165,545	162,226	138,608	191,700

(a) According to the official statistics of New South Wales.

Canada.—British Columbia continues to be the source of nearly all the lead produced in the Dominion. The production of this Province in 1907 was approximately as follows: Consolidated Mining and Smelting Company of Canada, Trail, 10,850 tons; Hall Mining and Smelting Company, Nelson, 3160 tons; all other smelters (chiefly the Sullivan Group Mining Company, Marysville), 5320 tons; exported in ore to the United States and Europe, 4000 tons; total, 23,330 tons. The production in 1906 was 26,389 tons. The chief producing mines in 1907 were the St. Eugene and the Sullivan, both in the East Kootenay. The lead furnaces of the Hall Mining and Smelting Company were blown out on Sept. 16 and remained idle during the remainder of the year. The Dominion bounty on lead was quiescent from Apr. 25, 1906, to Dec. 2, 1907, when fractional payments began to be made, the bounty being in effect only when the price of lead at London is below £16 per long ton. The total production of lead in Canada in 1907 was 23,783 short tons.

Germany.—The production of this Empire decreased about 5 per cent. in 1907 as compared with 1906.

GERMAN IMPORTS AND EXPORTS OF LEAD.
(In metric tons)

Year.	Metal.					Ore.				
	1903	1904	1905	1906	1907	1903	1904	1905	1906	1907
Imports.............	52,440	61,388	78,528	71,040	74,973	67,573	83,807	92,667	89,979	137,861
Exports.............	30,243	23,169	32,515	27,039	27,708	1,270	1,312	1,496	1,916	1,296

Mexico.—During the last five years the production of lead in this Republic has been very irregular. Having fallen off largely in 1906 it jumped up again in 1907, but is still far short of the maximum on record. The famous mines of Sierra Mojada are no longer what they were and on the whole Mexico is waning in importance as a lead-producing country, although it will doubtless continue to make an important production from many widely scattered districts, which will maintain a great smelting industry.

Spain.—The production in 1907 was 185,800 tons. The Carolina district has developed to such an extent during the last two or three years

that it is expected soon to be the leading lead producing district in Spain. The Penarroya company and other companies are contemplating important undertakings in the district. The production at present amounts to about 6000 tons of ore per month, of which 70 to 75 per cent. is sulphide, assaying 80 per cent. lead; while the remainder is carbonate ore assaying 55 to 60 per cent. lead. The richness of the veins is said to be increasing with depth.

United Kingdom.—The output of lead ore in the United Kingdom during 1906 was 30,795 long tons of dressed ore and the amount of lead obtainable by smelting was 22,335 tons, containing 147,647 oz. of silver. The corresponding figures for 1905 were 27,649 tons of dressed ore, 20,646 tons of lead, and 163,399 oz. of silver. There were 65 mines in operation, and in addition there were a few places where lead ore to the extent of 569 tons was won from open work quarries. The largest producers are the Mill Close mine in Derbyshire, Rhossemor in Flint, Van mine in Merroneth, Foxdale in the Isle of Man, Lead Hills in Lanark, and Greenside in Westmoreland.

The Lead Markets in 1907.

New York.—The consumption of lead, like that of all other metals, had assumed unprecedented proportions at the opening of the year 1907. Manufacturers had difficulty in supplying the wants of their customers and business was flourishing in all branches. The greatest proportion of the increased consumption was furnished through the enormous demands for the manufacture of sheets for the chemical industry and lead-covered cables. Whatever lead could be furnished by the smelters found a ready market at the price then ruling of 6c. New York—the highest figure in a period of 30 years. Prices doubtless would have been driven still higher if it had not been for the fact that the bulk of the entire production was furnished to the manufacturers under average contracts.

This situation continued into March, when the convulsions of the money market began to enforce the policy of retrenchment which became more and more extensive as time passed. It was in the larger industries requiring heavy amounts of capital that the effect was first felt, and it developed that while the outlet to the small consumers requiring white lead, lead pipe and mixed metals continued unabated, the effect of diminished sales was first felt through the falling off in the demand for chemical sheets and lead-covered cables. Nevertheless, prices were nominally maintained, because the largest producer had a regular outlet to its subsidiary concerns. Other important sellers, however, began to have difficulty in placing their product, and during May started their policy of forcing sales by shading what was known as the official quotation, which was reduced early in June to 5.75c. New York, without, however, altering

the basic conditions; the only result being that outside producers again shaded their prices. A repetition of these conditions was experienced when on July 3 the official price was reduced to 5.25c. New York. The lower market did not stimulate the demand, and the general pressure to realize brought about a lowering in the quotation to 4.75c. New York, early in September.

The severe panic which occurred in October forced a number of producers to try to find a market for their stocks regardless of price. A state of utter demoralization followed and it became evident early in November that the quantities available had grown so large that the official quotations were absolutely ignored, and it was then evident that the market was bound to become an open one. This was accelerated by large sales of Missouri lead at considerably lower prices than the nominal quotations.

In the *Engineering and Mining Journal* of Nov. 23 it was said: "Last week we stated that, for reasons then given, our quotations for lead would hereafter be those of the open market—that is, the prices of large quantities on actual sales—and not the contract prices of the chief producer, as heretofore. That producer has accepted the situation. The American Smelting and Refining Company announced last Monday that it withdraws its official quotations and in future will compete in the open market. By this action the artificial situation which has existed hitherto is at an end, and more normal conditions can be expected to prevail henceforth, since prices are again to be regulated by the law of supply and demand."

What was said then was true of the market subsequently. The moment the bars were down the quotations reflected the actual situation by declining from day to day, and by the end of November lead was freely offered at 4.10c. New York, and 3.95c. St. Louis. Even at these low prices consumers persisted in a hand-to-mouth policy. The situation was aggravated through the average contracts for fixed quantities, which were considerably larger than were required under the new conditions, so that a great many consumers were placed in a position where they were powerless to take advantage of the low prices. Sellers, being thus deprived of a market, tumbled over each other to book what small business turned up from time to time.

The downward rush continued into late December, when the market reached 3.25c. St. Louis, 3.40c. New York. Prices as low as these had not been established in over 10 years; they marked, however, the end of the protracted decline; both consumers and speculators became attracted to the metal and bought up large quantities, with the result that prices quickly advanced to about 3.40c. St. Louis, 3.55c. New York. At these figures the market stood at the end of the year.

AVERAGE MONTHLY PRICE OF LEAD PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*
1897....	3.04	3.28	3.41	3.32	3.26	3.33	3.72	3.84	4.30	4.00	2.96	3.70	3.58
1898....	3.65	3.71	3.72	3.63	3.64	3.82	3.95	4.00	3.99	3.78	3.76	3.76	3.78
1899....	4.18	4.49	4.37	4.31	4.44	4.43	4.52	4.57	4.58	4.58	3.70	4.64	4.47
1900....	4.68	4.68	4.68	4.68	4.18	3.90	4.03	4.25	4.35	4.35	4.58	4.35	4.37
1901....	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.15	4.33
1902....	4.00	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.069
1903....	4.075	4.075	4.442	4.567	4.325	4.210	4.075	4.075	4.243	4.375	4.218	4.162	4.237
1904....	4.347	4.375	4.475	4.475	4.423	4.196	4.192	4.111	4.200	4.200	4.200	4.600	4.309
1905....	4.552	4.450	4.470	4.500	4.500	4.500	4.524	4.665	4.850	4.850	5.200	5.422	4.707
1906....	5.600	5.464	5.350	5.404	5.685	5.750	5.750	5.750	5.750	5.750	5.750	5.900	5.657
1907....	6.000	6.000	6.000	6.000	6.000	5.760	5.288	5.250	4.813	4.750	4.376	3.658	5.325

London.—The lead market in 1907 was without special feature until the second half of the year, when the price of lead suffered from the same depression which was experienced in the other branches of the metal industry. At the end of January, Spanish lead was quoted at £19¾; at the end of February, £19½@19⅝; at the end of March, £19⅝; at the end of April, £20 $\frac{3}{16}$; at the end of May, £19⅞@20½; at the end of June, £19¾@21, according to time of delivery; at the end of July, £19½@19¾; at the end of August, £19⅝@18¾; at the end of September, £21½ for spot, and £18¾ for futures; at the end of October, £18½@17½; at the end of November, £16@15⅞; at the end of December, £14.

AVERAGE MONTHLY PRICE PER 2240 lb. OF LEAD AT LONDON. (*a*)
(In pounds sterling.)

Year	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1897......	11.717	11.708	11.562	11.787	11.837	11.912	12.250	12.679	13.650	13.575	13.100	12.600	12.367
1898......	12.508	12.362	12.650	13.062	13.700	13.437	12.950	12.800	12.800	13.050	13.412	13.100	12.983
1899......	13.375	14.350	14.150	14.375	14.146	14.283	14.385	14.733	15.267	16.179	17.096	16.883	14.933
1900......	16.296	16.542	16.612	16.733	16.900	17.225	17.533	17.633	17.667	17.596	17.229	16.233	16.987
1901......	15.925	14.667	13.379	12.421	12.275	12.342	12.150	11.692	11.954	11.600	11.267	10.533	12.521
1902......	10.567	11.617	11.508	11.596	11.600	11.271	11.233	11.121	10.892	10.746	10.717	10.754	11.261
1903......	11.304	11.708	13.225	12.404	11.800	11.437	11.383	11.146	11.167	11.108	11.108	11.179	11.579
1904......	11.558	11.592	12.037	12.254	11.754	11.521	11.667	11.737	11.787	12.187	12.892	12.775	11.983
1905......	12.875	12.462	12.296	12.658	12.762	13.000	13.608	13.958	13.950	14.679	15.337	17.050	13.719
1906......	16.850	16.031	15.922	15.959	16.725	16.813	16.525	17.109	18.266	19.350	19.281	19.609	17.370
1907......	19.828	19.531	19.703	19.975	19.688	20.188	20.350	19.063	19.775	18.531	17.281	14.500	19.03[illegible]

(*a*) The statistics for 1897–1905 are from the report of the Metallgesellschaft, Frankfurt am Main. Those for 1906 and 1907 are from the *Engineering and Mining Journal*.

White Lead, Red Lead, Litharge and Orange Mineral.

The demand for white lead, as well as for the oxides, during 1907, was in excess of that of 1906, and the plants of the consolidated interests and also those of the independent corroders, were employed to almost their full capacity. Prices which were advanced, just prior to our last review, to a basis of 6⅝c. for dry white lead and 7¼c. for lead in oil, were maintained on that basis until September, 1907, with concessions of ⅛ to ¼c., here and there, as the result of aggressive competition. As a

whole, however, the demand so far kept pace with the supply that there was no necessity for breaking prices, and the concessions referred to resulted chiefly from energetic competition at certain local points and were not general. Early in September there was a reduction of $\frac{1}{2}$c. on all pigments, following a decline of $25 per ton in pig lead which began with a break of $5 in June followed by two further reductions of $10 within the next 60 days. The steady shrinkage in the value of pig lead since then, amounting to fully $20 per ton, was met by a further reduction of $\frac{1}{4}$c. in the prices of white lead and the oxides on Dec. 16. Manufacturers are still producing from the higher cost metal, but the present disparity between the metal and its products is too wide to admit of much firmness in the latter and still greater concessions are reported to have been made privately on dry white lead to paint manufacturers who are disinclined to enter into contracts, in view of the probability of another reduction in the publicly quoted price, early in 1908.

The demand for oxides was especially active throughout 1907, and red lead was relatively stronger than any of the other pigments, by reason of its heavy consumption as a structural paint, as well as in the industries where it is employed for other purposes. Litharge was freely used in all of the ordinary channels of consumption, and even the demand for electrical purposes showed no appreciable shrinkage until near the close of the year, in spite of the reported curtailment of activity in that industry.

By reason of contracts entered into on the basis of prices prevailing before the advance in December, 1906, manufacturers of paints obtained a large share of their supplies at figures which gave them a better margin of profit than the difference in the card prices of dry lead and lead in oil would have afforded. Beyond the nominal narrowness of this margin, there was nothing in the course of the market to justify the apprehension referred to in this review last year as to the policy of the combined smelting and corroding interests toward the paint manufacturers who, although large consumers of lead, are using other pigments in still larger quantities in the manufacture of competing paints. It would appear that each branch of the merged interests is being operated with reference to its own profits, and in view of the competition which the smelting as well as the corroding branch has to face, there is nothing to encourage fear of a more complete or more aggressive monopoly in the near future than exists today.

One of the most interesting features of the present situation is the uncertainty as to the effect which the reduced cost of lead will have upon oxide of zinc, the prominence of which, as a competitor of white lead in the manufacture of mixed paints, has been gained through a wider difference than there is at present in the cost of the two pigments. The New Jersey Zinc Company makes its price for the ensuing year about

Nov. 1, and thus far it shows no inclination to modify the contract price made three months ago.

The net earnings of the National Lead Company in 1907 were $2,942,245, against $2,499,632 in 1906. According to the official report of the company for 1907, the volume of business for the year exceeded that of any other in its history, notwithstanding the paralyzed condition of trade in November and December. The experience of 16 years teaches that the business of this company is less susceptible to the fluctuations of trade than many others and does not suffer in the same measure when the general volume of trade shrinks. At the time of preparation of the report (about April, 1908) a comparison so far in 1908 with the unprecedented volume of business done at the same time in 1907 showed a shrinkage of 13½ per cent.

E. J. Cornish, president of the Carter White Lead Company of Chicago and Omaha, at the sixth annual convention of the New York State Retail Hardware Association in Buffalo, N. Y., Feb. 20, 1908, said: There is surprisingly little in the technology of corroding that can be called established. Learned authorities have made many mistakes and advanced theories that have led corroders into many expensive errors. In the manufacture of white lead today, much is empirical. The tendency of corroders is to follow safe, approved customs, rather than to embark upon proposed improvements that may prove deceptive.

In the manufacture of white lead only the purest metal lead is used. It must not contain over 0.003 per cent. of copper, iron, zinc or bismuth, 0.005 per cent. of antimony or 1 oz. of silver to the ton. If any of these ingredients is present in excess of the amount mentioned it will manifest itself in low percentage of corrosion or defective color of the finished product. Extreme whiteness indicates perfect corrosion. The whiter the lead, the more value it has as a pigment.

Several hundred patents on processes or improvements in processes for making white lead have been taken out in this country and Europe. They

PRODUCTION OF LEAD PIGMENTS IN THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897	7,798	$744,709	105,804	$9,522,360	8,591	$773,190	477	$76,320
1898	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200
1899	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200
1900	10,098	1,050,192	96,408	9,910,742	10,462	1,067,124	825	100,650
1901	13,103	1,448,550	100,787	11,252,653	9,460	979,586	1,087	224,667
1902	11,669	1,262,712	114,658	11,978,172	12,755	1,299,443	867	138,349
1903	12,300	1,385,900	112,700	12,228,024	12,400	1,326,800	1,000	168,000
1904	13,938	1,672,569	(a) 126,336	13,896,913	12,487	1,248,691	1,125	168,681
1905	16,269	1,919,767	(a) 122,398	12,068,443	12,643	1,422,616	1,000	120,000
1906	13,693	1,874,448	(a) 123,640	15,234,297	13,816	1,890,050	2,927	421,488
1907	13,370	1,778,717	111,409	12,254,990	14,769	1,624,553	815	123,917

(a) The output of "sublimed white lead," a mixed sulphate and oxide of lead, is not included in 1904-07.

may be divided into three classes: (1) The old Dutch or stack process. Probably four-fifths of the white lead made is by this process. (2) Chamber processes, in which the conditions of the stack are sought to be reproduced in chambers. The Carter process has been the most successful of this type. Probably more Carter white lead is sold than any other brand in the world unless the Dutch Boy has succeeded in unifying the several brands of the National Lead Company. (3) Precipitation processes. There are two plants in the United States using these processes, both of which are reasonably successful.

IMPORTS OF LEAD PIGMENTS INTO THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1896	1,543,262	$47,450	1,183,538	$52,400	51,050	$1,615	1,359,651	$51,077
1897	1,386,070	46,992	1,101,829	48,988	60,984	1,931	1,486,042	67,549
1898	682 449	25,780	506,739	24,334	56,417	2,021	795,116	37,745
1899	776,197	30,479	583,409	30,212	55,127	3,614	1,141,387	58,142
1900	549,551	25,532	456,872	28,336	77,314	2,852	1,068,793	61,885
1901	485,466	19,369	384,671	21,226	49,306	1,873	977,644	52 409
1902	1,075,839	37,833	506,423	25,320	88,115	2,908	997,494	49,060
1903	1,152,715	40,846	453,284	24,595	42,756	1,464	756,742	36,407
1904	836,077	30,115	587,383	33.788	44,541	1,500	766,469	37,178
1905	704.402	26,553	597,510	34,722	117,759	4,139	628,003	31,106
1906	1,093,619	50,741	647,636	41,233	87,230	3,737	770,342	42,519
1907	679,171	35,959	584,309	37,482	90,475	4,386	615.015	37,799

THE COST OF PRODUCING LEAD IN THE UNITED STATES.

Throughout the decline in the price of lead during the second half of 1907 strenuous efforts were made to secure a restriction of production. The American Smelting and Refining Company and the United States Smelting, Refining and Mining Company discontinued the receipt of lead-bearing ore except such as they were bound to take under previously existing contracts, but even in those cases they urged the producers to restrict their shipments in their own interest. Two subsidiaries of the American Smelting and Refining Company, viz., the Federal Mining and Smelting Company of the Coeur d'Alene and the Federal Lead Company of Missouri, curtailed their outputs. On the other hand, the two great independent producers, viz., the Bunker Hill & Sullivan of the Coeur d'Alene and the St. Joseph-Doe Run of southeastern Missouri, both resisted all persuasion to cut down their production. However, about the middle of December, the St. Joseph-Doe Run bowed to the adverse conditions of the market and sent orders to the mines to restrict their output 50 per cent. At this time the price for pig lead was about 3.60c. New York, or say 3.50c. St. Louis. It might be inferred that this was uncomfortably near the actual cost of production under present conditions, but subsequently orders were given to resume production at

the normal rate on Jan. 1, 1908. Five or six years ago Missouri lead could be laid down at St. Louis at 2.25c. per lb., but since then the cost of mining and milling in southeastern Missouri has risen immensely owing to repeated increases in the wages for labor and corresponding decreases in the rate of efficiency per man.

On the other hand, the cost of mining and milling in the Coeur d'Alene has remained nearly stationary, or even has decreased, the wages for labor having continued unchanged, while steady improvements have been made in the mining and milling practice. In the case of these mines the cost of producing lead is greatly dependent upon the silver content of the ore and the price for silver. The reports of the Federal Mining and Smelting Company, which is the largest single producer of lead, not only in the Coeur d'Alene but also in the United States, for the years ending Aug. 31, give the figures in Table I.

TABLE I.

	1905.	1906.	1907.
Tons ore mined	664,830	874,332	888,950
Average lead (*a*)	6.64%	7.21%	6.72%
Average silver, oz. (*a*)	4.05	4.48	4.15
Tons concentrate (*b*)	85,205	130,855	130,373
Ratio	7.8 :1	6.7 :1	6.8 :1
Oz. silver	2,689,867	3,920,884	3,689,298
Average per ton	31.57	29.96	28.30
Tons lead	44,137	63,029	59,746
Average	51.8%	48.17%	45.83%
Net profit	$1,242,698	$2,685,300	$2,532,250
Dividends paid	1,098,896	1,647,457	1,917,741

(*a*) Average yield, not average assay. (*b*) Includes mill concentrate and shipping ore.

The report of the Federal Mining and Smelting Company gives no data as to the cost of production, but a rough approximation may be made on the basis of average prices for silver and lead during the corresponding periods. Assuming that the company sold its ore on the same basis as other large producers in the Coeur d'Alene, the total proceeds and costs to the company were approximately as shown in Table II.

TABLE II.

	1905.	1906.	1907.
Total proceeds	$4,755,300	$7,345,804	$7,260,105
Less freight and smelting	1,363,280	2,093,680	2,085,968
Net value	3,212,020	5,252,124	5,174,137
Net earnings	1,242,698	2,685,300	2,532,250
Direct operating cost	1,969,322	2,566,824	2,641,887
Add development account	100,000	100,000	100,000
Total cost of production	2,069,322	2,666,824	2,741,887
Deduct silver value	1,508,684	2,410,642	2,388,452
Net cost of lead	560,638	331,245	353,435
Cost of lead per lb	0.70c.	0.29c	0.33c.
Cost of mining and milling per ton of crude ore	$3.11	$3.05	$3.08
Av. price of silver	58.82c.	64.72c.	68.15c.
Av. price of lead	4.45c.	5.42c.	5.79c.

In the computations of Table II an addition of $100,000 per annum has been made for development account. In the fiscal year ending Aug. 31, 1907, the company charged off $300,000 for this purpose, nothing having been thus written off in the two years previous. It is probable that this allowance is insufficient. Moreover, experience has amply proved that the net earnings from operation are not wholly applicable to the payment of dividends, a surplus being required by the company for application to new construction and extraordinary expenses. Consequently the actual cost of production to this company is higher than the above figures indicate.

However, on their basis, the cost of mining and milling per ton of crude ore was $3.11 in 1905, $3.05 in 1906, and $3.08 in 1907, while the cost per pound of lead was 0.70c., 0.29c. and 0.33c. respectively. It appears that while the cost for lead decreased rather largely from 1905 to 1906 the cost of mining and milling decreased only a trifle. The difference is explainable by an increase in the grade of the ore. This appears from the statistics which show that in 1906 the crude ore raised from the mines yielded an average of 7.21 per cent. lead and 4.48 oz. silver per ton, while in 1905 the figures were 6.64 and 4.05 respectively. Moreover, the value of silver increased largely from 1905 to 1906 and this has a highly important bearing on the mines of the Coeur d'Alene. Thus if the figures of the Federal Mining and Smelting Company for 1907 were computed on the basis of 50c. per oz. for silver, instead of 68.15c., it would appear that the cost of a pound of lead would have been approximately 1c. instead of 0.33c.

Of course it will be understood that these estimates mean that 1c. or 0.33c., as the case may be, must be received for a pound of lead in the Coeur d'Alene for the mining company to come out even. The cost on the basis of delivery at New York is another matter. This again is a fluctuating affair, depending upon the grade of the concentrate, the profit expected by the smelter, and other factors, but allowing a reasonable profit to the smelter, the cost of freight, smelting and refining on concentrate containing 45 to 50 per cent. lead may be fairly estimated at 2.33 to 2.5c. per lb. of lead. It may be inferred, therefore, that under the conditions of the Federal Mining and Smelting Company, with silver at 50c. per oz., the cost of delivering lead at New York is at least 3.33 to 3.5c. per lb., but with proper allowances for amortization of mining and milling plant, etc., the actual cost must be close to 4c. per lb. With silver at 67½c. per oz., on the other hand, the cost is 2.65 to 2.8c. without allowance for amortization, and probably about 3.3c. with it.

The cost of production to the Bunker Hill & Sullivan company figures out in much the same way as the Federal, but more favorably, because of the higher grade of its ore and its lower cost of mining and milling.

In the year ending May 31, 1907, this company mined 336,630 tons of ore, yielding 40,169 tons of lead and 1,645,719 oz. of silver. The average yield of the ore was about 12 per cent. lead and 5 oz. of silver per ton. The direct operating expenses were $665,379; the total expenses, including new construction, exploration, litigation, taxes and insurance, and in fact all charges was $934,657. The average price for silver during this period was 67.497c. per oz., at 95 per cent. of which the company realized for silver $1,055,268, which was more than $100,000 in excess of the total cost of production; consequently, the property would have been profitable if nothing at all had been received for its lead production. If, however, the price for silver had been only 50c. per oz., the amount realized for the silver product would have been less than the total expenses and the lead product would have cost about 0.2c. per lb. However, even then the total cost of this lead delivered at New York would have been only a little more than 2.5c. per lb. The Bunker Hill & Sullivan product is undoubtedly the cheapest large supply of lead in the United States at the present time.

The three companies considered in this discussion—the St. Joseph, Federal Mining and Smelting, and Bunker Hill & Sullivan—furnish nearly 40 per cent. of the domestic production of lead. The two districts of which they are representative—the Coeur d'Alene and Flat River-Bonne Terre—furnish nearly 60 per cent. Consequently the market for lead is always determined largely by the conditions in these districts.

Recent Improvements in Lead Smelting.

By H. O. Hofman.

Introductory—Physical Properties—Alloys.

New Publications.—L. S. Austin published a book entitled "Metallurgy of the Common Metals"; *Mining and Scientific Press*, San Francisco, Cal., 1907, 407 pp., $4. This book contains valuable chapters on lead smelting (pp. 306-334) and lead desilverizing (pp. 352-363). "History of Lead in the United States," by W. R. Ingalls,[1] contains a chronology of lead mining in the United States in which the leading events of the history of the metal (discovery, mining and treatment) and the operations of the lead mining and smelting companies are given; the facts are given in the chronological order and are accompanied by a few remarks. The earliest record of lead mining and smelting is that of Falling Creek, Va., 1621.

Analyses of Refined Lead.[2]—The accompanying table shows some

[1] *Trans.* A. I. M. E., 1907, XXXVIII.
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, p. 1179.

analyses of refined lead made at the Technical Analysis Department of Osaka, Japan.

BRANDS OF REFINED LEAD.

	Selby, Cal.	Trail, Brit. Col.	England.	England, Chemical Lead.	Broken Hill, N. S. W.	England, Enthoven.
Pb	99.9579	99.9890	99.9762	99.9693	99.9853	99.9851
Ins.	0.0040	Trace	Trace	Trace	Trace	Trace
Bi	0.0300	None	0.0046	Trace	None	0.0048
Cd	Trace	None	0.0002	0.0007	Trace	Trace
Ni	0.0001	Trace	Trace	0.0003	None	Trace
Co	None	None	Trace	Trace	Trace	Trace
Ag	0.0010	0.0025	Trace	0.0020	0.0009	0.0015
Mn	0.0008	None	0.0003	None	None	None
Cu	None	0.0003	None	0.0097	None	None
Sb	None	None	0.0137	0.0149	0.0108	0.0160
Sn	0.0004	0.0007	None	None	0.0004	None
As	0.0024	0.0020	0.0090	0.0002	None	None
Zn	0.0003	0.0002	Trace	Trace	0.0001	Trace
Fe	0.0027	0.0053	0.0039	0.0029	0.0025	0.0026

Structure of Lead.—W. Campbell,[1] in an extended illustrated paper on the structure of metals and the influences of mechanical and thermal treatment in general, takes up, among other metals, the macro- and micro-structure of lead. The surface of a bar of lead often shows characteristic dendritic forms parallel to the longer axis (etching brings out the granular structure) and etching a section through a bar shows the direction of the crystals, usually at right angles to the cooling planes, the crystals at the bottom and the sides being longer than those at the top. Mechanical treatment gives rise to slip-planes and distortion of crystals; tempering of rolled lead at 20 deg. C. causes a molecular rearrangement resulting in the formation of large crystals; ordinary electro-deposited lead shows a fern-like structure, while that obtained by the Betts process is so dense that it can be rolled into plates.

J. Cartaud and F. Osmond[2] give a number of photomicrographs of lead and some of its alloys in their paper covering the changes metals undergo when passing from the liquid to the solid state.

Volatility of Lead.—H. Moissan and T. Watanabe[3] distilled alloys of silver-copper, silver-tin, and silver-lead, and found that these four metals can be arranged as to their volatility in the following order: lead, silver copper, tin.

Powdery Lead.—J. W. Bailey patented[4] an apparatus for producing lead in the form of dust or powder resembling very much a Carr disintegrator in which liquid lead poured slowly is converted into dust, to be drawn off by an exhaust fan.

[1] *Metallurgie*, 1907, IV, 801, 825.
[2] *Rev. de Métallurgie*, 1907, IV, 820.
[3] *Compt. rend.*, 1907, CXLIV, 16.
[4] U. S. Patents, Nos. 846,384, March 5, 1907; and 864,443, August 27, 1907.

Lead-Aluminum.—H. Pecheux[1] determined the freezing points of some lead-aluminum and bismuth-aluminum alloys by means of a platinum and platinum-iridium, and a copper-nickel pyrometer. The data for the lead-aluminum alloys with 92 per cent. aluminum are 643 and 645 deg. C. respectively; with 94 per cent. aluminum 648 and 652 deg. C.; with 96 per cent. aluminum 637 and 635 deg. Centigrade.

Lead-Calcium.—Harkspill[2] prepared alloys of lead and calcium by introducing small pieces of calcium into a bath of lead chloride; the reaction was very violent. A simpler method is the electrolytic decomposition of molten calcium chloride, using a lead cathode. The author believes that there exists a compound, $Pb_3\,Ca_2$, which melts at 775 deg. C. and has a a specific gravity of 7.6.

Lead-Iron.—E. Isaac and G. Tammann[3] have studied the relations of lead and iron and settled by definite tests, what has been generally believed, that the two elements have no affinity whatever for each other.

Lead-Nickel.—A. Portevin[4] studied the alloys of lead and nickel. The freezing-point curve shows that with lead melting at 327 deg. C., there is formed a eutectic mixture with 0.07 per cent. nickel, freezing at 323 deg. C.; between 0.07 and 7 per cent. nickel, the alloys are homogeneous, but between the temperatures 323 and 1365 deg. C., crystals of nickel separate out; between 7 and 60 per cent. nickel, the alloys are perfectly liquid above 1365 deg. C., but at 1365 deg. C., nickel again crystallizes out; lastly, between 60 and 100 per cent. nickel, this metal separates out at temperatures ranging from 1365 to 1484 deg. C. (melting point of nickel), leaving behind an alloy with 7 per cent. nickel. The conclusion is that lead and nickel are partly miscible in the liquid but unmiscible in the solid state.

Lead-Selenium.—H. Pélabon[5] made a study of the alloys of lead and selenium. The latter raises the freezing point of lead; thus, pure lead solidifying at 325 deg. C., lead with 2 per cent. selenium will freeze at 745 deg. C., with 4.3 per cent., at 830 deg.; at 1065 deg. C. a maximum is reached with 27.65 per cent. selenium, which corresponds to PbSe. The freezing points then fall to 673 deg. C., a point corresponding to 45 per cent. selenium, and then remain constant; increasing the content of selenium above 45 per cent. causes two layers to form, a lighter one of pure selenium, and a heavier one consisting of a solution of selenium and PbSe.

Lead-Silver and Silver-Tin.—G. J. Petrenko[6] did some work on lead-silver and silver-tin alloys. Lead-silver alloys have already been studied by Heycock-Neville[7] and Friedrich[8].

---

1 *Compt. rend.*, 1906, CXLIII, 397.
2 *Compt. rend.*, 1906, CXLIII, 227.
3 *Zeit. anorg. Chem.*, 1907, LV, 58; *Metallurgie*, 1907, IV, 817.
4 *Rev. de Métallurgie*, 1907, IV, 814.
5 *Compt. rend.*, 1907, CXLIV, 1159.
6 *Zeit. anorg. Chem.*, 1907, LIII, 200; *Metallurgie*, 1907, IV, 836.
7 *Philosophical Trans.*, 1902, LV, 122.
8 *Metallurgie*, 1906, III, 396.

Other Lead Alloys.—Pouchine[1] determined the electromotive force of many alloys in solutions of acids and alkalies and drew conclusions as to their constitutions. Among others he found that lead and tellurium in a N/10 lead nitrate solution showed the existence of the compound PbTe which gave a solid solution with lead, but not with tellurium; that there existed no definite compounds of lead with copper, silver, bismuth, antimony nor arsenic, and that a solid solution was formed only with antimony; that zinc and silver in a N/10 zinc sulphate solution gave Zn_6Ag, Zn_4Ag, Zn_2Ag, ZnAg and $ZnAg_2$ (doubtful); that zinc and gold give Zn_6Au, Zn_2Au, ZnAu, and a solid solution between Zn_6Au and gold.

Ternary Alloys of Copper-Silver-Lead.—K. Friedrich and A. Leroux[2] investigated the eutectic point of the ternary alloy of copper, silver and lead. First they retraced the freezing point curves of copper-silver, lead-copper and lead-silver. The copper-silver curve differs from that of Osmond, but agrees with that of Heycock-Neville who, however, did not trace the eutectic line completely. The eutectic point was found to lie at 778 deg. C., and the eutectic to consist of 28 parts pure copper and 72 parts of a solid solution of silver with 6 per cent. copper. The lead-copper curve agrees with that of Heycock-Neville as does the lead-silver curve. The ternary eutectic consists of Cu, 0.5 per cent., Ag 2 per cent. and Pb 97.5 per cent.; freezes at 299 deg. C. or 0.5-1.0 deg. lower than the freezing point of the lead-silver eutectic with Ag 3 per cent., Pb 97 per cent. The paper contains 47 diagrams and 34 photomicrographs.

Commercial Lead Alloys.—L. Burrows[3] treats in a popular article of the principal lead alloys which are used mainly for solder, bearing-metal and type-metal. He emphasizes the necessity in our days of specifying the composition of an alloy, as a commercial name covers many sins, intentional and otherwise.

An editorial[4] refers to tea-lead, i. e., the lead lining of tea chests, which is 0.005 to 0.010 in. thick and forms an important article of commerce. It is made by pressing molten lead between two flat stones covered with several layers of coarse rice paper. The loss in melting tea-lead alone is 10 to 15 per cent. by weight; this loss is reduced by plunging tea-lead into a bath of lead.

Fluxes[5] are very helpful in diminishing the losses in melting down base metals and their alloys. A flux must dissolve the oxide formed in melting and fuse at a lower temperature than the metal. Thus rosin is a common flux for lead; its melting point is also sufficiently high to prevent it catching fire; other fluxes of similar character are more expen-

[1] *Rev. de Métallurgie*, 1907, IV 926.
[2] *Métallurgie*, 1907, IV 293.
[3] *Metal Industry*, 1907, V, 9.
[4] *Brass World*, 1907, III, 339.
[5] *Brass World*, 1907, III, 363.

sive. The same is the case with waxes and fats. Sodium has a cleaning effect upon lead, but coats the surface with caustic soda which absorbs water. Sal ammoniac is not beneficial. An addition of 0.1 per cent. arsenic in the form of an alloy (as 10 per cent., Pb 90 per cent.) clears the surface of the lead, but hardens the metal and makes it sluggish.

An editorial[1] discusses the soft solders in the market. These consist usually of lead and tin, special varieties sometimes containing bismuth, cadmium, or zinc. The three kinds of solder usually sold contain Sn 66 per cent., Pb 34 per cent. (which has a low melting point); Sn 50 per cent., Pb 50 per cent. (which is used for most purposes, but solidifies suddenly upon cooling); and Sn 34 per cent., Pb 66 per cent. (which is best for wiping a joint as it passes through a pasty stage in solidifying). On account of the common use of scrap in preparing solder, this often contains some impurities. Thus 2 to 5 per cent. antimony is not unusual; it makes the alloy look well, but renders it less fluid, less adhesive and more corrodible than the pure alloy. Bismuth is sometimes added to lower the melting point, but not often on account of the high cost; the presence of the smallest amount of zinc spoils the solder, as does a similar amount of aluminum; the addition of 0.00X per cent. phosphorus in the form of phosphor-tin is beneficial; a larger amount is harmful.

Constitution of Lead Compounds.—K. Friedrich[2] investigated the freezing point curves of Ag_2S-Cu_2S and PbS-Cu_2S. Silver sulphide with 99.6 per cent. Ag_2 S, solidifying at 835 deg. C., and cuprous sulphide with 99.6 per cent. Cu_2S, fusing at 1121 deg. C., appear to form a solid solution; the curve has a depression reaching 677 deg. C. as a minimum, but there is no break and the microscopical examination does not reveal any eutectic structure. The transformation of Ag_2S, known to occur at 175 deg. C., could not be detected. Lead sulphide, with 87.1 per cent. lead, melting at 1114 deg. C. and cuprous sulphide, form an euctetic mixture of the composition PbS, 49 per cent., and Cu_2S, 51 per cent., which freezes at 540 deg. C. The investigation proves the non-existence of the compounds $9Cu_2S$ 2PbS, $3Cu_2S$ 9PbS, $9Cu_2S$ 4PbS and $3Cu_2S$ 2PbS.

The same author[3] studied the relations of PbS-FeS and PbS-Ag_2S. The PbS, melting at 1114 deg. C. and FeS, melting at 1187 deg. C., form a eutectic mixture which freezes at 863 deg. C. and consists of PbS, 70 per cent., FeS, 30 per cent., the eutectic line extending from ordinate to ordinate. PbS, and Ag_2S, melting at 835 deg. C., also form a eutectic mixture which freezes at 630 deg. C. and is composed of PbS, 27 per cent., Ag_2S, 77 per cent.; the eutectic line does not reach the ordinates, showing that near these there is a tendency to form solid solutions. At 175 deg. C. the diagram shows a second line parallel to the eutectic line, extending from

[1] *Brass World*, 1907, III, 23.
[2] *Metallurgie*, 1907, IV, 671.
[3] *Metallurgie*, 1907, IV, 479.

100 to 75 per cent. Ag_2S; this represents a transformation in the solid Ag_2S.

Fournet Series.—P. Schütz[1] has re-investigated the affinities of some of the leading metals for sulphur. The original Fournet series of 1834, it will be remembered, is: copper, iron, cobalt, nickel, tin, zinc, lead, silver, mercury, gold, arsenic, antimony. The order of Schütz is manganese, copper, nickel, iron, tin, zinc, lead. The influences of calcium and barium sulphides upon decomposition was tested by adding these compounds to charges of lead sulphide and iron. They were that in the presence of calcium sulphide the reaction was so violent that some lead was lost by volatilization; with barium sulphide the reaction proceeded quietly, and the loss of lead was small. In both cases the yield of metallic lead was reduced on account of the larger amount of matte formed. If with $PbS+Fe$ the yield in lead was 81.88 per cent., in the presence of calcium sulphide it was 45.03 per cent., and of barium sulphide 77.3 per cent. These facts have mainly a theoretical interest, as both $CaSO_4$ and $BaSO_4$ are readily decomposed, in the presence of carbonaceous matter at about 1100 deg. C., into CaO, or BaO, and SO_3; slags from lead and copper blast furnaces rarely contain any alkaline sulphides.

Lead Oxide.—F. O. Doeltz and W. Mostowitsch[2] determined the melting point of PbO at 906 deg. C.; later researches by Mostowitsch fix it at 883 deg. C.

W. Stahl[3] calculates that the following metallic oxides are decomposed by heat at atmospheric pressure at the temperatures given: $2\ Ag_2O=2\ Ag_2+O_2$, 383 deg. C.; $4CuO=2Cu_2O+O_2$, 1679 deg.; $2Cu_2O=2Cu_2+O_2$ 1935 deg.; $2CuO=Cu_2+O_2$, 1775 deg. (1886); $2PbO=Pb_2+O_2$, 2348 deg.; $2NiO=Ni_2+O_2$, 2751 deg.; $2\ ZnO=Zn_2+O_2$, 3817 deg. While these dissociations temperatures, with the exception of Ag_2O, are not reached in metallurgical carbon-furnaces, the data give an idea of the relative stability of these oxides.

F. O. Doeltz and C. A. Graumann[4] determined the reducibility of PbO by means of solid carbon at different temperatures. They found that at 500 deg. C. PbO was not de-oxidized, but that reduction was noticeable between 530 and 555 deg. C., moderately strong between 550 and 563 deg., very decided at 600 deg. and still more so at 700 deg.

W. Mostowitsch[5] investigated the constitution of lead silicates. His work replaces the earlier work of Grenet[6], Simmons[7], and Manihot-Kieser[8]. The material used was SiO_2, 99.97-per cent. pure, and PbO prepared in

---

[1] *Metallurgie*, 1907, IV, 659,694.
[2] *Metallurgie*, 1907, IV, 289.
[3] *Metallurgie*, 1907, IV, 682.
[4] *Metallurgie*, 1907, IV, 420.
[5] *Op. cit.*, 1907, IV, 648.
[6] Babu, L. "Traité de Métallurgie Générale." Béranger, Paris, 1904, I., 500.
[7] *Journ.*, Chem. Soc., 1903, LXXXIII, 1,449.
[8] Liebig *Ann. der Chem.*, 1905, CCCXLII 356.

two different ways, also chemically pure; the melting point of PbO was fixed at 883 and 884 deg. C. The series of eight tests ranged from 6 $PbO:SiO_2$ (PbO, 95.68; SiO_2, 4.32) to $PbO:SiO_2$ (PbO, 78.68; SiO_2, 21.32). They show that PbO and SiO_2, as far as ascertained, do not form any chemical compounds as has been held, but combine between 700 and 800 deg. C., mostly below 750 deg. C., to a glass or solution which in the liquid state has the property of dissolving PbO when the O ratio of $PbO:SiO_2$ exceeds that of 1:2. This dissolving power increases with the temperature; when the latter sinks, the excess of PbO falls out of solution and is accompanied by a rise in temperature. The mixtures, $6PbO:SiO_2$, $5PbO:SiO_2$, $4PbO:SiO_2$, $3PbO:SiO_2$ and $2.5PbO:SiO_2$, become as fluid as water below 800 deg.; C. $2PbO:SiO_2$ requires 850 deg. C.; $4PbO:3SiO_2$ over 880 deg.; and $PbO:SiO_2$ over 940 deg. C. to become pourable. All these silicates are readily soluble in dilute nitric and hydrochloric acids, and acetic acid with a separation of silica. $PbO:SiO_2$ is completely soluble in hot dilute ammonia or caustic potash, with the formation of Na_2SiO_3 or K_2SiO_3 and the respective plumbates. The silicates are readily decomposed; even water attacks them. By heating in a current of hydrogen at 350 to 600 deg. C., the PbO of a silicate is first reduced to Pb_2O and then to Pb; the reduction of pure PbO begins between 290 and 300 deg. C. Gases like CO and CH_4 begin to show a reducing effect at about 500 deg. C. The formation of lead silicate at as low a temperature as 700 to 800 deg. C. is important in the roasting of galena, as the formation of silica hinders the sulphatization in the hand-reverberatory and pot-roasting furnaces. Near the flue end of the hand-reverberatory furnace, where the temperature is below 700 deg. C., PbS is largely converted into PbSO; in fact at Mechernich,[1] with its two-hearth, hand-reverberatory, roasting furnaces, the ore on the upper hearth loses only 1 to 2 per cent. sulphur, as the PbO formed is in part sulphatized by the SO_3 set free on the lower hearth.

Lead Ores.

Purchasing Lead Ores.—L. S. Austin[2] publishes the schedule of prices of the American Smelting and Refining Company, in 1906, for the ores of Clear Creek and Gilpin counties, Colo.; he plots the prices to be paid for lead ores on the neutral and flat scales, and for the lead values of both scales; he then shows what anomalies there exist in the general plan and indicates what steps may be taken in apportioning different ores so as to bring them into a higher class that they may command a better price. This plan of plotting prices and values is very helpful for forming a clear idea of classification.

---

[1] *The Mineral Industry*, 1905, XIV, 404 (Guillemain).
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, 226.

Smelting Practice.

Smelting in the Harz Mountains.—K. Waldeck issued a pamphlet[1] on silver-lead smelting in the Harz Mountains and travels over familiar historical ground. The leading points of general interest had already been given in his thesis.[2] In the preface of the present paper the introduction of the Huntington-Heberlein process at Clausthal and Andreasberg, and of the Savelsberg process at Lautenthal, are mentioned; also the new zinc desilverizing plant at the last place, but the text is silent regarding these improvements and other minor ones.

Smelting at Malfidano, Italy.[3]—These works treat zinc-lead ores from their own mines. In the lead department there are four hand-reverberatory roasting furnaces, in which the temperature at the fire-bridge is raised to frit the charge, and three water-jacketed blast-furnaces with eight tuyeres, producing 40 tons base bullion in 24 hours. This is desilverized in 35-ton kettles by the Parkes process.

Smelting in the Joplin District.—D. Brittain[4] discusses the metallurgical apparatus that has been used and the results obtained in smelting lead and zinc ores in the Joplin district. For lead-smelting there have been in operation log-furnaces, air-furnaces, hearth- and blast-furnaces. The log-furnace consists of flat stones placed on edge on a slanting floor to the shape of a V; a charge consisting of wood and lead ore is ignited, natural draft furnishes the necessary air for combustion, and the lead liberated trickles down the inclined floor. Details of some of the air- hearth- and blast-furnaces show the former and present practice. The latest plant is that of the Petraeus Company, built in 1902. The equipment consists of four single, water-back ore-hearths, a blast-furnace, a bag-room 100x68 ft. containing 680 bags, each 30 ft. long and 2 ft. 9 in. diameter, made of cotton cloth with 44 threads to the inch both ways. A charge consisting of 7000 lb. galena (77 to 82 per cent. Pb), 1 bushel Kansas coal and 1 bushel lime is smelted by 2 men in 7 hrs., for which each receives $2.20; the gray slag assays Pb, 35 per cent.; Fe,167 per cent. The blast-furnace is charged with galena, carbonate ore, gray slag and ignited blue fume; the slag produced averages: SiO_2, 22 to 28 per cent.; CaO, 20 to 24 per cent.; Fe, 24 per cent.; Zn, 7 per cent.; Pb, 0.7 per cent. The ore-hearth and blast-furnace leads are melted down in a kettle of 9000 lb. capacity, poled and molded. The gases from the ore-hearths and the blast-furnace are drawn by a 90-in. Buffalo exhaust fan through a flue, 6x4 ft. in the clear, before they enter the bag house. No white fume is made.

Smelting in Eureka, Nevada.—W. R. Ingalls[5] in a general article upon

---

[1] "Streifzüge durch die Blei- und Silber-Hütten des Oberharzes," Knapp, Halle, 1907, pp. 68, pl. 3.
[2] *The Mineral Industry*, 1902, XI, 445.
[3] *Compt. rend.*, 1907, 274.
[4] *Eng. and Min. Journ.*, 1907, LXXXIV, 862.
[5] *Eng. and Min. Journ.*, 1907, LXXXIV, 1051.

the silver-lead mines at Eureka, Nev., reviews the smelting industry. It began in 1869 and increased so fast that in 1870 there were already 14 blast-furnaces in operation. The industry fell more and more into the hands of two large companies, the Eureka Consolidated and the Richmond Consolidated. The output increased until 1880; since then the production has declined; in 1890 the Richmond, and in 1891 the Eureka, plants were closed down; in 1874 the Richmond company erected a steam-Pattinson desilverizing plant, the only one that has been in operation in the United States. The paper contains illustrations and descriptions of some of the old furnaces.

Smelting in Salt Lake City, Utah.[1]—There are two silver-lead smelting plants near Salt Lake City, that of the American Smelting and Refining Company at Murray, and that of the United States Smelting, Refining and Mining Company at Bingham Junction. The Murray smelter, which was built in 1901, has undergone several changes since it was discussed in these pages.[2] The present notes supplement the former statements. The sampling mill is provided with Gates breakers and Vezin samplers; the finishing sample represents 1-625th of the original. The ores are bedded on an uncovered cement floor carrying 2-in. partitions, 8 ft. high, which inclose spaces 40 to 50 ft. wide by 60 to 80 ft. long, holding 1200 to 2500 tons ore. The ore-beds are made up according to the prevalence of some leading component, such as S, SiO_2, CaO, FeO, Pb. The original 12 hand-reverberatory roasting furnaces are used in part only, and then mainly for desulphurizing lead matte which is not treated by the Huntington-Heberlein process. The matte, crushed to $\frac{3}{8}$-in. size, is charged every four hours in 4200-℔. lots; the furnace treats 12.6 tons in 24 hr. reducing the sulphur from 22 to 4 per cent.; the roasted matte is stored in brick-lined, steel bins, cooled by sprinkling, and discharged into cars to be conveyed to ore-beds. Of the 20 Brückner cylinders, 8.5x22 ft., some are still in operation for the treatment of sulphide ores low in lead. A charge of 24 tons is roasted in 48 hr., the sulphur being reduced to 4 per cent., with a coal consumption of 16.67 per cent., and the service of one man for 11 furnaces. In 1905 the Huntington-Heberlein process was introduced and has greatly changed the former practice of roasting. The new plant contains five Godfrey roasters and one row of Huntington-Heberlein pots, with room for another row, served by a 30-ton overhead traveling crane. The Godfrey roasters are 26 ft. diameter; the hearth is cast-iron, covered with brick; the capacity of a furnace per shift is 8 to 12 tons ore with a sulphur-reduction from 18 or 25 to 8 or 12 per cent., and a fuel-consumption of 3500 to 4500 ℔. coal in 24 hr. The Huntington-

---

[1] W. R. Ingalls, *Eng. and Min. Journ.*, 1907, LXXXIV, 527,575; R. B. Brinsmade, *Mines and Minerals*, 1907, XXVIII, 216.
[2] *The Mineral Industry*, 1905, XIV, 396.

Heberlein pots[1] are spherical and of standard size, i.e., 9 ft. in diameter, and have long, heavy trunions supported by open bearings; these rest on a circular concrete wall which encloses the pot; the top of the wall extends about 1 ft. above the brick working-floor. A pot is covered with a steel hood which can be moved up and down the inclined flue, telescope-fashion, by means of a pulley and counterweight. The flues from the pots end in an elevated, circular, steel flue, with V-shaped bottom, which also receives the gases from the Godfrey roasters through steel pipes, two to a furnace, placed on opposite sides of the feed-hopper.

A pot receives a charge of about nine tons which it treats in about 12 hr. There are two classes of charges: Raw ore, high in sulphur, with SiO_2, 40 per cent., and FeO, 20 per cent.; and roasted ore, assaying about 12 per cent. sulphur, with SiO_2, 10 per cent., and FeO, 20 per cent.; the lead is kept down to 20 per cent. and the zinc to 10 per cent.; the loss in zinc in the operation is 1.5 per cent. and less. In charging, 1.5 in. coal ashes is spread over the grate; this is followed by hot, dry, Godfrey "special roast" (8 per cent. sulphur); then comes one ton of "primer," hot, dry, Godfrey "common roast" (12 per cent. sulphur); finally seven tons of a moistened mixture of raw and roasted ore with 20 per cent. sulphur, prepared by hand in an old pot and dumped from it by means of the crane. When the surface has been leveled, the blast at 12 oz. pressure is turned on and the hood put in place. The sulphur is reduced to 4 per cent. in 8 to 14 (av., 12) hours, the only attention required during the blow being the poking-down of blow-holes. The top of a blown charge contains a small amount of unfused fine material which is imperfectly desulphurized, the rest forms a solid cake. The hood is now transferred to one side, the pot raised out of its pit, tilted to pour off the fines onto the floor, which go to the next charge on top of the primer, replaced in its normal position, carried to the breaking platform at the end of the building, inverted, and the cake of the charge dumped on the floor laid with closely set steel rails when it breaks. Large dumps are raised by the crane and dropped again; pieces small enough to permit handling go into a 10x20-in. Blake crusher discharging onto a double, wire-rope conveyer with steel buckets; this raises the material and discharges it into steel railroad cars running on a trestle and delivering into hopper-shaped bins near the ore beds. The blast-furnace department has remained unchanged, except that a No. 10 rotary blower has been added to supplement the work of the four original piston-blowing engines. The eight furnaces are fed mechanically. The charging car, filled below the bedding floor, is raised electrically on one of two inclined trestles (placed at right angle to the line of furnaces) to an electric transfer car, the track of which runs parallel to the line of

[1] *The Mineral Industry*, 1906, XV, 527.

furnaces. The charge is dropped upon the gable-shaped spreader of a furnace and falls the hight of 6 ft. A furnace 48x168 in., using about 12 to 15 per cent. of Huntington-Heberlein material, now puts through in 24 hr. about 200 tons charge, of which 80 per cent. is ore, instead of 160 tons before the advent of the Huntington-Heberlein process. Further advantages of the Huntington-Heberlein material are, the reduction of the waste slag and of matte in the charge, and the diminution of the amount of flue-dust formed. The handling of base bullion is unchanged. It remains to be mentioned that the kettle-dross is liquated on a perforated, V-shaped, iron trough before it goes back to the blast-furnace charge. The matte, roasted as stated above, is smelted in one of the eight blast-furnaces, the crucible of which has been filled. The resulting matte, with 40 to 50 per cent. copper, goes to the Garfield plant of the same company. The waste slag contains 0.6 per cent. lead and 0.15 per cent. copper. The gases from the blast-furnaces pass through a cylindrical steel flue with V-shaped bottom to a masonry flue, and thence, since 1907, into a bag-house. The steel flue is, coated inside with a graphite paint and protected outside by a roof. The original concrete flues proved unsatisfactory, as they leaked under the combined action of heat and sulphurous gases; they are being replaced by masonry having the form of a catenary arch which requires no buck-stays nor abutments. The arch, 9 in. thick, is laid in two courses of stringers with headers every sixth row and a 3-in. expansion space every 20 ft. The brick flue is 2000 ft. long and has cross-section of 250 sq.ft. The bag-house, erected at a cost of $150,000, not for saving metal but to do away with the smoke which injured the surrounding agricultural land, is a building 100x216 ft.; it contains 4160 bags 30 ft. long by 18 in. diameter, grouped in four sections, each of which has a steel chimney with top 175 ft. above the floor. The gases, at a temperature of 50 to 70 deg. C., are taken from the flue by an 18x6-ft. fan which has a capacity of 250,000 cu.ft. gas per min., requires 125 h.p., and delivers the gas at a pressure of 1 in. water. The flue dust is molded in the Chisholm, Boyd & White machine to briquets 4 in. diameter and 2 in. hight.

The United States smelter, started in 1902, has two sampling departments. In the oxide mill there are four Snyder mechanical samplers.[1] The ore, broken to 2.5 in. in a Blake crusher, is elevated to the top of the mill, passed through one sampler which takes out 1-5 as sample; this then passes through the other three, being crushed by rolls between the samplers. The final mechanical sample (1-625th of the original) is repeatedly mixed and halved in a riffle down to 5 lb., dried, ground in a sample-grinder, halved, bucked through 80-mesh with silver ore and 120-mesh with gold ore, when five bottle-samples ore prepared. The sulphide mill

[1] *The Mineral Industry*, 1900, IX, 440.

has the same general arrangement as the oxide mill; the first sampler takes out ¼ and the other three 1-10th, so that the final mechanical sample represents 1-4000th of the original ore. The cars bringing the ore arrive on elevated trestles; on either side are the bedding-bins with run-ways on top of the divisions for handling the ore. The ores are roasted in 16 hand-reverberatory furnaces, 16 by 70 ft., having 14 rabbling- and one fire-door on a side; 4000 lb. ore- or 5000 lb. matte-charges are fed every 4 hr.; the coal consumption is 35 per cent. of the charge; there are three men on a shift. The roasted ore, which is fritted and retains 4 to 5 per cent. sulphur, is drawn into pots; these, when cool, are dumped into a car on a depressed track to be hauled by an electric locomotive over the bins of the smelting department. There is an experimental plant of "blast-roasters" designed by C. Robinson. Such a roaster consists of a low, rectangular, cast-iron shaft with fixed sides, sliding front- and back-doors, and a grate for the bottom. The charge is introduced from the top, blown as in the Huntington-Heberlein process, and the progress watched from the top through doors. When blown, the front and back doors are opened and the cake of sintered charge pushed out by a ram, as is the coke in a horizontal retort-oven. The ores are smelted in six blast-furnaces, served by Connersville rotary blowers. The hearth of a furnace is 45x160 in., there are ten 4-in. tuyeres on a side; the crucible is inclosed by an elliptical iron shell; there are six flanged-steel water-jackets on a side and one on each end; the hight of the shaft from tuyeres to downcomer is 13 ft. and to feed floor 24 ft. The great hight above the charge gives room for three cross-rails which act as distributors for the charge dumped through two longitudinal slots which are closed with steel doors by means of a lever. The charging-car of five-ton capacity, standing on a depressed track, receives its charge from two-wheel buggies through openings in the bedding floor, is hauled electrically up an incline to the feed-floor on a track which has a gage equal to the length of a furnace, runs over the tops of the furnaces, is emptied, and returns to its former place after having passed over a U-shaped loop. The bottom of the car is V-shaped and has drop-doors. A furnace puts through 170 tons charge in 24 hours., or 3.4 tons per sq.ft. of hearth area. Matte and slag are tapped into an oval, movable fore-hearth; the slag overflows into a Nesmith waste-slag pot with Devereux tap. An electric locomotive hauls the slag to the dump; the shells are collected in a 30-ton steel car on a sunken track to be hauled up a trestle to hoppered charge-bins. The lead from the siphon-tap is collected in two-wheel, cast-iron pots, trammed to the casting-house, collected in three 30-ton kettles, skimmed, siphoned into pig or into anode molds for the company's Betts electrolytic-desilverizing plant at Grasselli, Ind. The kettle dross is liquidated in a reverberatory furnace and returned to the blast-furnace. The lead matte is tapped into

flat Rhodes molds, crushed, roasted and concentrated to 40-per cent. copper matte in a blast-furnace. The blast-furnaces have, per shift: 18 men on the bedding floor, one man on each of the two charging-cars, and one man on the feed-floor; on the furnace floor are two motor-men and 10 men for handling matte and slag; further, each blast-furnace has a tapper, a helper and a pot man. The gases pass into a comparatively short main flue of masonry with catenary-arch roof and self-cleaning (i.e., gable-shaped) floor, then into the bag-house (cost $100,000) put into operation in 1907. This is a brick building 192x58x60 ft. with steel floor 12 ft. above the ground. It is divided into five compartments by brick walls reaching from floor to roof; each compartment under the steel floor is divided by cross-walls into three subdivisions. The steel floor has 18-in. nipples placed at 2-ft. centers, which receive 2080 to 2200 bags, 30 to 32 ft. long and 18 in. diameter. The gases, having a temperature of about 80 deg. C., are drawn from the main flue by an electrically driven fan of 150,000 cu. ft. capacity per min., with an expenditure of 90 h.p., and are delivered at a pressure of 1 in. water into a horizontal steel main from which 15 branches extend under the steel floor into the 15 compartments. The filtered gases from each compartment pass through a branch pipe into the horizontal main lying outside on the ridge-pole and ending in a steel stack. Taking 2200 bags, 18 in.x30 ft., gives 310,860 sq.ft. filtering area, or 50,000 sq.ft. per furnace, or 300 sq.ft. per ton of charge. The collected dust, about 1 per cent. of the charge, contains 30 to 35 per cent. arsenic, probably due to the arsenical ores of Eureka, Nev.; the mode of treatment has still to be worked out.

Smelting at Trail, B. C.—J. M. Turnbull[1] gives a brief description of the smelting works of the Consolidated Mining and Smelting Company of Canada, at Trail, B. C. These were erected in 1896 to treat copper ores, enlarged in 1898 to work lead ores, and again increased in 1906. The new sampling mill for copper ores has a No. 8 crusher and Vezin sampling machines. It handles 150 tons ore in 24 hours. The lead ore sampling division has a Blake crusher and Brunton samplers; it handles 30 tons ore per hour. Ores are roasted in two O'Hara furnaces, 9x95 ft., and 12x97 ft., four 26-ft. Heberlein, circular, mechanical-roasting furnaces and 24 Huntington-Heberlein converting-pots. There are three copper blast-furnaces 43x263 in. smelting 300 tons in 24 hours and two small-size ones to be enlarged; further, two lead blast-furnaces 45x140 in. and 45x160 in. respectively at tuyeres, treating 125 to 150 tons charge in 24 hours. The blowing plant consists of three No. 7, one No. 7½, and one No. 8 Connersville blower, one No. 8, one No. 9 and one No. 9½ Root blower, each driven by a separate motor. There are extensive dust chambers and two stacks, the highest being 185 ft. The flue dust is

[1] *Can. Min. Journ.*, 1907, XXVIII, 421.

briquetted with 5 per cent. lime in two Chisholm, Boyd & White briquet presses.

Smelting in Mexico.—T. D. Murphy[1] in a general article on the mines of El Doctor, Querétaro, Mexico, outlines the smelting and refining operations. The ores are smelted in a water-jacket blast-furnace, 36x42 in., with Arents siphon-tap; blast of 12 to 15 oz. pressure is furnished by a No. 4½ Baker blower driven by a 15-h.p. Westinghouse motor. The furnace puts through in 24 hours 40 tons charge with 12 per cent. charcoal and 8 per cent. coke. The base bullion is very rich in antimony, assaying Sb, 33 per cent.; Pb, 62 per cent.; Ag, 233.3 oz.; and Au, 0.17 oz. The matte assays Pb, 10 to 12 per cent.; Ag, 30 oz. An average slag analysis shows SiO_2, 33; FeO, 28; CaO, 23; ZnO, 2.5; Al_2O_3, 10; Pb, 0.5 per cent.; Ag, 2 oz. The base bullion is shipped to Germany at present, as pay is received for the antimony. Formerly it was cupelled in a German cupelling furnace, when 30 to 35 tons were cupelled in seven days with ½ cord of wood per ton of lead; the yield in silver was 95 per cent., and in lead 85 per cent.; the cake of crude silver was refined in crucibles to 0.998 fineness. The matte with lead 10 to 12 per cent., and over 30 oz. silver, is heap roasted and returned to the blast-furnace as basic flux.

Smelting in Australia.—G. D. Delprat[2] describes the smelting and refining works of Port Pirie, N. S. W. The ore-dressing plant furnishes two products to the smelter. Concentrates assaying Pb, 55; Zn, 10 per cent.; Ag, 26 oz.; and slimes assaying Pb, 18; Zn, 17 per cent.; Ag, 17 oz. The slimes are sintered by heap-roasting[3] near the ore-dressing plant. The mode of operating is as follows: The slimes are run out from the settling tanks onto the ground to form a layer 9 to 12 in. thick, are air-dried for several days and cut with spades into blocks; the blocks are further air-dried four days, until hard enough to permit handling, loaded on trucks and trammed to the sintering plant. The blocks are built into heaps with brick channels along the bottom; the sides of a heap are plastered with slimes, and draft-holes are left open at the top; a heap is 200 to 250 ft. long, 20 to 22 ft. wide, 6½ft. high, and holds 1000 to 1500 tons. A fire is kindled in the brick channels and kept going for one or two days to start the roasting. A heap burns 10 to 15 days and requires 1.5 per cent. wood. When burnt, the outside cover is removed, mixed with water and used as plaster for another heap. Heap-roasted slime forms a hard, porous clinker, well suited for the blast-furnace. Slimes before roasting assayed: Pb, 17; Zn, 16; S, 12.5 per cent.; Ag, 17.5 oz.; and after roasting: Pb, 14; Zn, 12.5; S, 7 per cent.; Ag, 15.8 oz. The loss

---

[1] *Min. and Sci. Press*, 1907, XCV, 241.
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, 317, 517.
[3] *The Mineral Industry*, 1903, XII, 253.

in lead is considerable, but is compensated for by the cheapness of the operation; 100,000 tons of slimes are treated annually in this way.

The concentrates go to the smelting works direct. Here they are mixed with 14 per cent. fine shells, 45 per cent. fine iron ore and 9 per cent. silicious ore, and rough-roasted in five Ropp furnaces, each of which treats 100 tons charge in 24 hours. Each roaster discharges into a 3-ton iron bin from which the ore is lowered into 1-ton trucks which are raised to the feed-floor of the Huntington-Heberlein battery of 17 converters, each of 8 tons capacity. The hot ore is dumped into a pot, the hood lowered and the charge blown four hours. The pot is tipped and the charge dropped upon a floor on which are placed four cast-iron cones; the large pieces are further broken by spalling. The blast-furnace department has 13 furnaces with Arents siphon tap, of which eight are in operation. A furnace is 62x212 in. at the tuyeres, the working hight is 20 ft., the total hight 24 ft. 9 in. There are on a side 11 cast-iron water-jackets each with a tuyere-opening; the tuyeres do not protrude into the furnace, the large distance of 62 in. between the tuyeres being probably made possible by the coarseness of the charge, which is composed of sintered slimes, 1000; converted concentrates, 2000; raw concentrates, 200; foul slag, 800; iron ore, 1050; limestone, 550; total, 5600 ℔.; coke, 840 ℔. The gases are drawn off through an oblong Darby tube, 4x8 ft., which reaches 3 ft. into the furnace. It is claimed that taking off the gases at the center counteracts the formation of wall accretions. The blast for eight furnaces is supplied by eight Green blowers, No. 8, discharging into a single main 60 in. diameter; the eight furnaces take 75,000 cu. ft. air per min., under a pressure of 30 oz. In 24 hours a furnace puts through 120 tons of charge containing 17 per cent. lead, produces 25 to 30 tons base bullion, and recovers 95 per cent. lead, 98 per cent. silver and 100 per cent. gold. The slag has the following average composition: SiO_2, 25; FeO, 33; MnO, 6; CaO, 12; ZnO, 13; Al_2O_3, 6; S, 3; Pb, 1.5 per cent. In the zinc desilverizing plant a charge of 36 tons of base bullion is melted down in eight hours in a reverberatory furnace, with hearth built into an iron pan, and drossed. The charge makes 3000 ℔. dross, which assays: Pb, 77.5; Cu, 8; Fe, 1; Zn, 1.2; S, 4.3; As, 0.7; Sb, 0.7; insoluble, 2.1 per cent.; and goes to the blast furnace. The coal consumption is 2 per cent. on the base bullion. The drossed lead is run into a second reverberatory furnace similar to the first and freed from arsenic and antimony in 16 hours by heating, cooling and skimming twice. The skimmings assay Pb, 74; Sb, 10.7; As, 1 per cent. The coal consumption is 3 per cent. on the base bullion. At given periods the accumulated skimmings are desilverized by melting down in a reverberatory furnace, in 6-hour periods, charges of 2800 ℔. skimmings, 120 ℔. fine coal and 120 ℔. fine coke, tapping the reduced lead every 12 hours and the desilverized skimmings every 24

hours. The furnace requires 10 per cent. coal. The lead goes to the softening furnace; the skimmings are melted in a cupola (450 lb. skimmings, 250 lb. blast-furnace slag, 100 lb. coke) for antimonial lead (Pb, 78; Sb, 20; As, 1.5) which goes to market, and waste slag. The desilverizing kettles have a capacity of 40 tons softened base bullion; the outer diameter including rim is 9 ft. 11 in., the depth 3 ft. 6½ in., the radius of the circular part 4 ft. 6½ in., the thickness 2 in. They are provided with Howard stirrer and press. The first zinc addition is 200 lb.; the resulting gold crust is used twice over with fresh kettle-charges and finally assays Ag, 200 oz.; Au, 6 oz.; Zn, 10 per cent.; Pb, 89 per cent. The second zinc addition is about 750 lb.; the resulting silver crust. freed from excess lead in the press, weighs about one ton and assays Ag, 3000 oz.; Zn., 20 per cent.; Pb, 70 per cent. The third and last zinc addition weighs 700 lb., cleans the kettle and is used over again as usual. A kettle requires 3.4 per cent. coal for a complete operation. The desilverized lead is drawn into a refining reverberatory furnace where it is heated, cooled and skimmed twice, which requires 4 per cent. coal; the refining skimmings go to the ore blast-furnace. The refined lead is run into a market kettle and thence drawn off into molds; it contains 99.995 per cent. lead and forms 83.3 per cent. of the original base bullion. The distillation of the crust is carried on in gas-fired furnace with Siemens regenerative chambers for superheating the secondary air. The furnace, shown

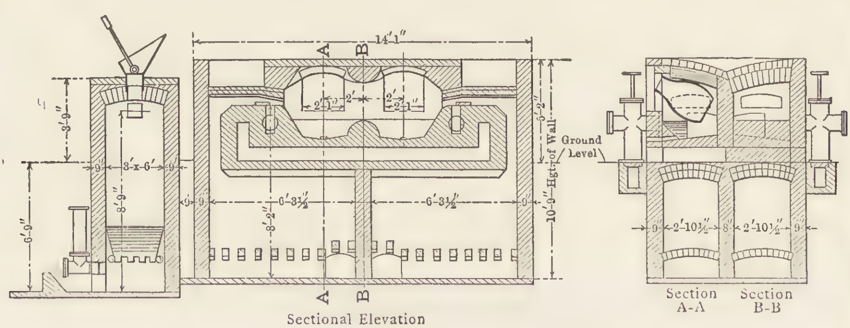


FIG. 1.—GAS FIRED RETORT FURNACE.

in the accompanying drawings, is double, contains two graphite retorts on a side, or four in all; a retort holds a charge of 1000 to 1200 lb. of crust which is worked in four hours with a consumption of 15 per cent. coal. The rich-lead from the gold crust is molded, stacked until 35 tons have accumulated, melted down in a kettle, drossed, liquated with a Howard press, giving 15,000 to 20,000 lb. dross. This is worked off in a concentrating cupel, with hearth made of cement and sand (4:1) and a capacity of 0.5 ton charge, to an enriched lead (with 17,000 oz. Ag,

and 550 oz. Au) and litharge (with 40 to 50 oz. Ag, and 0.1 oz. Au); the cupellation is completed on a finishing hearth to doré bullion with 30 to 35 thousandths gold to be parted. The liquated lead, from which the above gold dross has been taken, assays Ag, 225 oz., and Au, 7 oz. It is treated with 700 lb. zinc to free it from gold and is then desilverized in the usual way. The gold crust is treated in the same manner as described above. The silver crust upon distilling gives 82 per cent. rich lead with 3500 oz. silver, 3 per cent. dross, and 15 per cent. metallic zinc. The rich lead is cupelled to crude silver and this refined in a new cupel to bullion, 998.5 fine. The litharge goes to the ore blast-furnace. The refined silver is melted down in graphite crucibles in charges of 2200 oz., when copper is added to bring the fineness down to 996 (the standard for export bars) and finally cast into bars weighing 1020 oz. The weekly product is 100,000 oz. The doré silver is parted by the sulphuric acid process in the usual way, using cast-iron dissolving and settling kettles and water-cooled cast-iron crystallizing tanks. The silver sulphate crystals, after washing and drying, are reduced to metal by fusing with charcoal in a crucible (a method long ago abandoned in this country on account of the losses in silver). The residual gold, after the usual boiling with sulphuric acid, etc., is treated with hydrochloric acid, washed, dried and cast into bars which are 992 fine.

Smelting in the Ore-hearth.—E. T. Perkins[1] treats of mining and smelting at Granby, Mo. The ore-hearth treatment is briefly the following: The galena, assaying 82 to 85 per cent. lead, is smelted in five jumbo furnaces. In 10 hours is worked a mixture consisting of 2400 lb. galena, 1200 lb. white fume (Pb, 73 per cent.), 400 lb. blue fume (Pb, 66 per cent.), 1 bushel lime with 1000 lb. coal. Four men work on a shift, two being before the fire in 15- to 20-min. periods. The products are lead, 85 per cent.; white fume, 9.5 per cent.; blue fume, 3 per cent.; and slag, 2.5 per cent. (Pb. 40 per cent.). The gases, drawn by a 6-ft. American blower, pass first through a steel settling chamber (blue fume), then through 200 ft. of steel pipe and goose-necks into the bag-house which contains 640 bags 18 in. diameter and 30 ft. long. The collected white fume is burnt over and charged back into the furnace. The men are expected to recover as metal 65 per cent. of the lead in the galena, 50 per cent. in the blue fume, and 60 per cent. in the white fume; they receive for this \$2 a day and ¼c. premium for every additional pound of lead recovered; they average over \$3 per day.

Roasting and Reaction Processes.—R. Schenck and W. Rassbach[2] have investigated in an ingeniously contrived apparatus the chemical processes

---

[1] *Eng. and Min. Journ.*, 1907, LXXXIV, 388.
[2] *Metallurgie*, 1907, IV, 455.

that are possible by the interaction of Pb, PbS, $PbSO_4$, PbO and SO_2, i.e., of substances formed in the roasting and reaction processes of lead smelting. Under the guidance of the phase rule these may be grouped to form four equations:

(1) $PbS + PbSO_4 = 2\,Pb + 2SO_2$. This equation is reversible; equilibria were found at 609 deg. C. with a tension of SO_2 of 30 mm. of mercury; at 655 deg. with 155 mm.; at 700 deg. with 442 mm.; at 723 deg. with 735 mm. The action of PbS upon $PbSO_4$ begins at 550 deg. C. and increases in velocity with the temperature, but is governed by the pressure of sulphur dioxide.

(2) $PbS + 2PbO = 3Pb + SO_2$. This equation is also reversible; equilibria were found at 692 deg. C. with a tension of SO_2 of 6 mm. of mercury; at 755 deg. with 38 mm.; at 800 deg. C. with 99 mm.; at 847 deg. with 544 mm.; at 870 deg. with 830 mm. The action of PbS upon PbO begins at 650 to 660 deg. C. and becomes very decided at 700 deg. C.; at 800 deg. C. PbS begins to become volatile, and the volatilization increases rapidly with the temperature. The progress of the reactions upon cooling down is to some extent retarded by the property lead has of dissolving PbS, and by the fact that the substances are not as intimately mixed as at the start, the lead being on the bottom and covered by PbO, which prevents the SO_2 from acting.

(3 and 4) $PbS + 3PbSO_4 = 4PbO + 4SO_2$ and $Pb + PbSO_4 = 2PbO + SO_2$. These equations show no equilibria even up to a pressure of SO_2 of one atmosphere. The components begin to act at 550 deg. C. and the reaction proceeds irreversibly.

It will be noticed that reactions were studied under pressures of pure sulphur dioxide ranging from 6 to 830 mm. of mercury. In furnace practice there is no pressure of sulphur dioxide; on the contrary there is a slight vacuum due to the draft. In a furnace charge, also, one has to deal not with pure sulphur dioxide as in the experiments, but with air charged with carbon dioxide and small amounts of sulphur dioxide, not over 7 per cent., the partial pressure of which is low. Lastly, sulphur dioxide, the gaseous phase, being withdrawn in the furnace as soon as formed, we have to deal with three phases instead of four; hence the presupposed equilibria and reversible reactions have no existence and the first two equations in furnace-work proceed only in one, the usual direction from left to right. The experiments, while of great scientific interest, do not seem to have any application in practice. As to reaction No. 3 with reference to the Huntington-Heberlein process, the air forced through the pot under pressure drives off the sulphur dioxide set free as soon as formed, and thus assists in the quickness of the roast besides increasing the temperature which favors an acceleration of the process. Equation No. 4 appears to be doubtful under working conditions.

Smelting in the Reverberatory Furnaces.—A. E. Swinney[1] describes with drawings of the reverberatory smelting and cupelling furnaces the details observed in working a charge of gold slime (Au, 15.5 per cent.) from a cyanide plant, by the Tavener process[2] on a bath of lead. The lead bullion was calculated to weigh 32,000 oz. and to assay 10 per cent. gold; the cupelled gold bullion was remelted in crucibles and cast into bars 857 to 862 fine.

Pot-Roasting.—W. M. Hutchins[3] in discussing in a general way some of the reactions taking place in lime-roasting, lays stress upon the decomposition of $CaSO_4$, formed in the Huntington-Heberlein and added to in the Carmichael Bradford processes, by the formation of silicate, as in the case with the decomposition of $PbSO_4$. He states his belief that when $CaSO_4$ and PbS are heated together, the PbS is oxidized by the $CaSO_4$, which is reduced to CaS and later re-oxidized by the blast. The experiments of Doeltz[4] have shown that the endothermic reaction, $PbS+CaSO_4=PbSO_4+CaS$, does not take place and that on the contrary there is a tendency for the reaction to go from right to left.

E. Prost[5] has published a general article on lime-roasting of galena which is a summary of some of the leading papers that have appeared so far and have already been discussed in these reviews.

W. R. Ingalls[6] states that the St. Joseph Lead Company at Herculaneum, Mo., has installed the Savelsberg process with a five-pot plant, while the Federal Lead Company at Alton, Ill., has introduced the Huntington-Heberlein process which replaces the Scotch hearth during the four hot months of the summer.

According to Mack and Scibird[7] Cripple Creek fines are added in Colorado to the Huntington-Heberlein charges instead of lime, to keep them from fusing, and with equal success.

H. O. Hofman, R. P. Reynolds and A. E. Wells[8] carried on some laboratory experiments in lime-roasting a galena concentrate from Idaho with reference to the Savelsberg process. As converters they used clay crucibles for charges weighing about 1 kg., and a cast-iron slag-pot for charges of 20 kg. In the crucible-converters two series of charges were worked; in the first, a singulo-silicate slag was made the basis; in the second, various sub-silicates containing, however, similar amounts of limestone as the first. The conclusions arrived at were: That in lime-roasting a silicious galena concentrate low in blende, charges containing a wide range of CaO and SiO_2 and little FeO can be blown successfully; that a sin-

---

[1] Inst. Min. and Met., 1906-07, XVI, 51. *Eng. and Min. Journ.*, 1907, LXXXIII, 608.
[2] *The Mineral Industry*, 1903, XII, 251.
[3] *Eng. and Min. Journ.*, 1907, LXIII, 201.
[4] *The Mineral Industry*, 1905, XIV, 403.
[5] *Rev. Univ. des Mines.* 1907, XVIII, 303.
[6] *Eng. and Min. Journ.*, 1908, LXXXV, 24.
[7] *Min. and Sci. Press*, 1907, XCV, 751.
[8] *Trans.* A. I. M. E., 1907, XXXVIII.

gulo-silicate charge with limestone equal to 16 per cent. of the weight of the ore gives most satisfactory results as regards the physical condition of the product, the elimination of sulphur and the loss of lead and silver; that the same is true with a sub-silicate; that a low pressure blast is a better desulphurizer and causes less loss than a high pressure blast. G. A. Packard (*loc. cit.*), in discussing the paper, gives some results he obtained when working along similar lines.

A. Savelsberg patented[1] a process of pot-roasting sulphide ores with limestone (or a substance which after having been heated swells up in contact with water), dumping the roasted charge, disintegrating it by wetting down, mixing again with limestone and repeating the pot-roast. That is, if you have failed to eliminate a sufficient amount of sulphur with one roast, you may try again.

A. Lotti[2] patented a process for roasting and agglomerating lead or copper sulphides. Lead sulphide is stirred into a given quantity of slag tapped from the blast-furnace which produces a spongy mass, sulphurous fumes being given off at the same time. While still red-hot, air is forced through the sponge which roasts off much of the sulphur, the charge becoming heated. After 1 to 3 hours, the mass is said to have become completely desulphurized and converted into a solid block which, broken up, forms a good blast-furnace material.

Blast-furnace Site.—H. Lang[3] discusses the level vs. the terrace site for a smelting plant and emphasizes again the advantages of the level site, especially in relation to modern large-size works which treat such amounts of ore that handling by mechanical appliances has become an absolute necessity. L. S. Austin[4] contends that with plants of medium tonnage, say 400 tons per day, the terraced site and industrial locomotive have many advantages as against the level site and the full-size, broad-gage locomotive advocated by Lang; he cites the terraced Washoe plant of the Anaconda Copper Mining Company, which handles 13,000 tons of materials per day with compressed-air engines in a most satisfactory way.

Handling Materials.—E. H. Messiter[5] gives with three photographic reproductions and plan drawing a brief outline of the system of ore handling in operation at the smelter of the Greene Cananea Company, Cananea, Sonora, Mexico, which appears to be very economical. Ores are dumped into steel bins underneath which motor-drawn feed-cars travel and deliver their contents to 30-in. Robins belt conveyers; these rise at an angle of 20 deg., discharge either into a gyratory crusher or a second conveyer leading to the sampling mill. The rejected ore is conveyed by belts to an

[1] U. S. Patent No. 870,690, November 12, 1907.
[2] U. S. Patent No. 847,017, March 12, 1907.
[3] *Eng. and Min. Journ.*, 1907, LXXXIII, 565.
[4] *Ibid.*, p. 726.
[5] *Min. and Sci. Press*, 1907, XCIV, 539; XCV, 528.

overhead structure which travels over the bedding floor and distributes the ore uniformly over piles 400 ft. long. Another machine removes the ore from the beds, attacking all parts of the cross-sections, and conveys the mixture to feeding bins or the furnaces themselves.

The T. M. Park[1] automatic loader, manufactured by the Railway Materials Company, of Chicago, is in operation at the Murray plant of the American Smelting and Refining Company, for loading iron ore into one-ton charging buggies. The machine consists of a heavy steel truck carrying an inclined steel plate with travelling scraper-arms for gathering the ore from the ground, and a short, wide conveyer belt for transferring the ore to the car. The truck is manipulated in a manner similar to a steam roller; the power for the two motors is taken from any convenient circuit, one serving for propelling the machine and operating the scraper, the other for driving the conveyer. Working without a conveyer, a buggy is loaded with iron ore in 10 minutes.

Blast-furnace Working.—L. S. Austin[2] has edited some personal letters of the late T. S. Austin relating to lead metallurgy, and has given the world at large the benefit of them. The letters deserve careful study; some of the leading opinions are brought together here, arranged, however, somewhat differently from the original. The letters deal mainly with blast-furnace work.

There are a few remarks about roasting. With regard to the constituents of the charge, pyrite helps in roasting arsenical and blendic ores; lead tends to agglomerate the charge; silica decomposes sulphates. As to furnaces, the Brückner cylinder requires close watching and careful management, neither of which it gets in ordinary circumstances. Smelting —In the interval of time during which these letters were written, the working hight of blast furnaces had been raised from 12 or 14 ft. to 20 ft. The increased hight, besides keeping the throat cool, has the effect of keeping the charge tight, and this in turn causes the gases to remain longer in the lower part of the furnace and to exercise an increased chemical effect; high furnaces, further, do not require as careful feeding as low ones. Mechanical feeding, besides being cheaper than hand-feeding, at least in large plants, has the advantage that the operation is independent of the laborer, who is liable to become careless, especially in the early hours of the morning.

The size of charge particles has a decided influence upon the running of the furnace. Thus, fine ores cause slow running, liability to clog at the tuyeres, irregularities in general, and high losses in silver. It is advisable to sinter fine ores, the extra expense being more than counterbalanced

---

[1] *Eng. and Min. Journ.*, 1907, LXXXIII, 1189.
[2] *Min. and Sci. Press*, 1907, XCIV, 252, 341, 537, 762: XCV, 59.

by the advantage of having a coarse charge. Lead in charge—10 per cent. is a low figure, but does good work, other things being equal.

The question of slags has received more careful study in the practice of lead-smelting than in any other branch of non-ferrous metallurgy. The characteristics of the "half-slag" with SiO_2, 30; FeO, 40; CaO, 20; total, 90 per cent. are: Dropping from the spout, it shows a slight thread; when rising in the pot, more or less concentric rings are seen, and radiating lines (usually four) when the pot is about full; a big pot as well as a ladle-sample will show crystals in the central part when broken (an excess of FeO tends toward large crystals while excess of CaO favors a granular structure); the crystalline structure of the normal slag extends to the edge (if it stops short of the edge and this has a reddish tinge there is an excess of SiO_2); if the cavities in the central part show no crystals, but are smooth and hard looking, there is an excess of FeO in regard to the SiO_2; a smooth top on a big pot indicates an excess of CaO. The "three-quarter slag" with SiO_2, 33; FeO, 33; CaO, 23; total, 89 per cent., is metallurgically about as good as the half-slag; it is usually more economical as to fluxes, but requires a little more fuel. In calculating a charge, zinc oxide and alumina may be advantageously figured as replacing lime, which in its turn covers any barium and magnesium oxides that are present. Thus, a slag SiO_2 33; FeO, 33; $ZnO+Al_2O_3$, 17; $CaO+MgO+BaO$, 17 per cent., does good work. On the other hand, A. Raht makes his slag run SiO_2 and FeO each 33 to 35 per cent., calculates his Ca(MgBa)O as about half the SiO_2, and neglects the rest. In many instances this gives the same results as Austin's modification of the standard three-quarter slag. The "whole, or one-to-one slag" with SiO_2, 35; FeO, 27; CaO, 28; total, 90 per cent., is difficult to fuse, runs slowly and is liable to give a hot top.

The manner of figuring in zinc oxide and alumina has just been referred to. It is safe to assume that one-half of the zinc in the charge goes into the slag. While basic slags favor the scorification of zinc oxide, they are commercially unfavorable, as silicious ores, on account of their abundance, usually command a higher smelting charge than basic ores. Taking $CaO=56$ and $Al_2O_3=103$, it is safe to assume that 1 lb. CaO can be replaced by 2 lb. Al_2O_3, considering the usual amount of Al_2O_3 present in lead blast-furnace charges.

The percentage of fuel in a charge is a point which has not received the attention it deserves. A charge with metallic sulphides requires less fuel than one made up of oxides. A good general rule is to avoid any excess of fuel and to keep on the "ragged edge of reduction," as this gives large tonnage, long campaigns, good furnace conditions and good metallurgical work; the slag will assay 0.5 to 1.0 per cent. lead, and be fluid and hot, the volume of smoke issuing from the throat will be small, and gray or dark

in color, and the lead output will be good. Over-reduction, i.e., using too much fuel, is accompanied by no end of disturbances. There is danger of iron and copper sows; of making much speiss in the presence of arsenic; the crucible-content becomes sticky; the furnace tightens and speed is lessened; slags run slowly, although they are hot; coke appears at the tap-hole, and white smoke and overfire at the top; slags contain too little lead, viz., 0.3 to 0.4 per cent. In under-reduction, i.e., using too little fuel, the speed is high; the slags, while fluid, are cold and look heavy; and run 1 per cent. lead and over: a large volume of thick, grayish-white smoke arises from the throat; and the yield in lead is bad. The correction is made by adding, say with a slag of 2 per cent. lead, 4 per cent. more fuel and then reducing this gradually when the results of the correction have been noticed. If the fuel is to be reduced when the furnace is doing fair work, it is wise to decrease by not more than 1 per cent. at a time. In running a new charge, it is advisable to use a little extra fuel in order that the assays and physical appearances may form clear guides in making any necessary changes.

The volume and pressure of blast require careful regulation. As a rule the furnace ought to get as much air as it can stand without the top getting hot and the crew being overcrowded with the increased production. When once the volume of air has been settled upon, it is advisable to keep this constant and to make changes in the charge to correct any evils that may occur. However, with over-reduction, an increase of blast gives speed and keeps the charge for less time in contact with the carbon; while with under-reduction a diminution of blast will help matters, as the charge remains longer in contact with carbon. The aspect of the crucible-content also furnishes a means of recognizing the amount of reduction that is going on. Thus, if the lead appears drossy, this may be due to under-reduction, to copper from over-reduction, or to a lack of sulphur. If due to under-reduction, the slag will show high assays and the furnace the other characteristics given above.

The lead content of the matte under normal conditions varies from 12 to 15 per cent.; if it is higher, addition of scrap iron assists in lowering it. In concentrating roasted lead matte in the blast-furnace, it is advisable to have a high reduction in order to recover all the lead that can be obtained. Further, the whole melted content of the crucible is advantageously tapped into a suitable overflow-pot in order to insure cleaning out of the bottom of the furnace. Speiss is very liable to be formed with a "quarter slag" containing SiO_2, 28; FeO, 50; CaO, 12; total, 90 per cent. In order to keep the speiss fluid, it is necessary to change to a slag that runs higher in silica and in lime. Wall accretions disturb regular work; a palliative is to run slowly with an increased amount of fuel; the only corrective is to bar them down. Austin prefers blowing out the furnace and

performing a clean piece of work instead of barring down in the usual way. The composition of blast furnace gas should form an indication of the chemical process in the furnace; it will differ however with the character of the charge, which may be made up of oxide or sulphur mineral. Under ordinary working conditions, where both classes of mineral are present, the furnace is doing good work when CO_2: $CO=2:1$. Losses in lead and silver are dependent upon many factors. A good recovery is 95 per cent. lead, and 98 per cent. silver; with 180- to 200-oz. base bullion the slag will run: Pb, 0.6 to 0.8 per cent. and Ag, 0.6 to 0.8 oz.; and with 100-oz. bullion about 0.5 oz. As a general rule it may be held that high-silica slags give high silver losses unless the furnace is run slowly. With a half-slag, for example, silica 33 per cent. and over would be high, 32 to 33 per cent., medium, and 30 per cent. low; the low-silica slag will give the smallest loss. However, nobody will lower the silica-content unless forced to do so. The loss in silver is greater when non-ferrous galena is smelted with a dry silver ore than when lead and silver are closely associated; the latter class of ores admits of faster driving than the former. The presence of copper matte reduces losses, but matte too high in copper gives the troublesome copper-sponge. In separating matte from slag, a reverberatory furnace does excellent work, with slags containing little impurities; with zinky slags, the zinc troubles resulting from a mushy mass separating out between matte and slag occur readily with a one-to-one slag, but disappear with a three-quarter slag. As to the operation of the Parkes process, an interesting fact noticed in 1887, but recorded here for the first time, is that at the Germania (Salt Lake City) works, gold-losses began to appear in the refining plant when gold-crust was made as distinct from silver-crust. They disappeared when no distinction was made between the two, and only doré silver produced, as is the general practice to-day in this country.

Effect of Size of Charge.—L. S. Austin[1] records an interesting experience of G. B. Lee in 1896 at the Union smelting works, at Leadville, Colorado, where in blast-furnace smelting the lead went into the wall-accretions instead of collecting in the crucible. This was because the smelting zone extended too far upward, although the lowest permissible amount of fuel (14.5 to 15 per cent. Starkville coke with 20 per cent. ash) was being used. The remedy was found by making the charge four times as heavy as usual. There were fed into the furnace, 43x130 in. at tuyeres, at a time 600 ℔. of coke followed by 4000 ℔. of charge and 1500 ℔. of slag. The reason for the improvement in the working appears to be that with a small-sized charge the fuel becomes evenly distributed through the charge and thus favors the creeping-up of the heat. Reduction in the size of charge-components also proved helpful.

[1] *Min. and Sci. Press*, 1907, XCIV, 61.

Calculation of Charges.—R. Chauvenet[1] has published a series of papers on the calculation of furnace charges to which the reader is referred.

Mechanical Feeding.—E. H. Messiter[2] patented three mechanical charging devices which are to distribute evenly over the throat of the furnace the charge received in a stationary hopper.

Patching of Water-jacket.—Bromide[3] describes the successful patching of a ragged hole about 3x5 in. on the inner side of a steel water-jacket with simple means. The hole was enlarged to 4x8 in. and the edges were beveled with a file. Into the patch, a piece of plate 6x10 in., there was

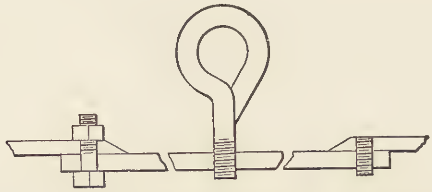


FIG. 2.—BAR WITH EYE SECURED IN THE PATCH

secured a 0.5-in. bar with eye which was to serve as handle, and as anchor for clamping. The patch was placed against the inside of the jacket and clamped; a row of holes slightly smaller than 0.5 in. was drilled through the overlapping iron and the jacket, $\frac{3}{8}$-in. bolts wound with thin wire were inserted through the hand-hole, drawn through the drilled holes and supplied with nuts. The patch was loosened, a thick lute distributed under the overlapping edge and the nuts then drawn down. The bolts were then withdrawn one at a time, a tapering thread was cut in each hole, a plug inserted and driven hard with a pipe-wrench. The plugs were cut off flush with the jacket and then the eye in the center of the patch. Trimming the surface with a file and rubbing with emery-cloth gave the desired smooth surface.

Fore-hearth.—An editorial[4] states that the zinky mush which is liable to accumulate in the fore-hearth, used for separating matte and slag, and to cause trouble because it cannot be tapped and often has to be raked out, can be decomposed by feeding scrap iron into the hearth. This decomposes the zinc sulphide and forms iron sulphide, while the liberated metallic zinc boils off and burns, showing the well-known white fumes.

Waste Slag.—H. Lang[5] describes the disposal of waste slag at the works of the Compagnie du Boleo, Lower California, where it is used for con-

---

[1] *Min. Reporter*, 1907, LVI, 56, 76, 96, 148, 170, 190, 212, 264, 378, 396.
[2] U. S. Patents Nos. 840,573, 840,574, 840,575, Jan. 8, 1907.
[3] *Eng. and Min. Journ.*, 1907, LXXXIII, 344.
[4] *Electrochem. and Met. Ind.*, 1907, V, 40.
[5] *Eng. and Min. Journ.*, 1907, LXXXIII, 1240.

struction of a breakwater. The slag is tapped into a tank resting on a flat-car; this is hauled by a locomotive to a convenient point where blocks are cast by running slag into pits made in the dirt alongside the track, and hooks are inserted to allow raising the blocks by a travelling crane and transferring them to the end of the breakwater. It is suggested that the molds be made prismatic, or better still that the slag be cast *in situ* where it is wanted.

Two other papers of similar purport are those of Smith[1] and Hixon[2]. L. S. Austin[3] describes with illustrations the slag-casting machine constructed by B. N. Bennetts for the Tacoma Smelting Company, Tacoma, Wash., which is in successful operation. While devised for a copper blast-furnace, the idea is of course applicable to lead smelting.

Smelter Fumes.

Earlier Investigation.—P. Frazer[4] publishes with short abstracts a bibliography of 46 titles on the injuries to vegetation by furnace-gases. Here only the effects of smelter gases and fluedust will be briefly touched upon. The leading investigations upon this special phase, until recently, have been those of Schertel-Schröder[5] and Reuss,[6] and the only treatise covering the whole smoke question is that of E. Haselhoff and G. Lindau.[7] The effect of furnace gases upon plants resembles that of decay in the autumn; the leaves, imperfectly developed, become pale, bleach, soon assume a dirty autumn coloring, and wither; in woods the effects are seen first in patches here and there; these unite later on to form barrens. Evergreens show a behavior similar to deciduous trees. The withering effect is due mainly to the direct contact of sulphur dioxide, and to a smaller extent of trioxide, with the leaves and needles, and can be valued by the percentage of sulphur these contain in excess of similar plants in the same region not exposed to the fumes. The sensitiveness of plants to sulphur dioxide and trioxide varies. Soil quickly converts H_2SO_3 into H_2SO_4 and neutralizes this, hence there is no direct action of these two acids upon the roots. The damage done by sulphurous gases depends upon the amount of dioxide present, the hight at which it is discharged into the air, the direction of the wind and the moisture of the air. In England the law says that the gases discharged into the air may not contain over 9.2 grains trioxide per cubic meter; they rarely exceed one-

[1] *Min. and Sci. Press*, 1907, XCVI, 205.
[2] *Ibid.*, 1907, XCVI, 553.
[3] *Min. and Sci. Press*, 1907, XCIV, 282.
[4] *Trans.*, A. I. M. E., 1907, XXXVIII.
[5] "Freiberger Jahrbuch," 1884, 93
[6] Two pamphlets, Jäger & Sohn, Goslar, 1893 and 1896.
[7] "Die Beschädigung der Vegetation durch Rauch," Bornträger, Leipzig, 1903.

third that amount. In Prussia, gases may not contain over 0.02 per cent. dioxide by volume. Needles and twigs of pines near Freiberg (Saxony) from healthy unaffected trees showed 0.162 per cent. sulphuric acid; in areas where injury was just observable to the eye the percentage increased to 0.210 to 0.300; a higher grade of injury gave 0.3 to 0.5 per cent.; a still greater injury 0.5 per cent. and over. The idea that high chimneys lessen the amount of damage to vegetation is realized only to a moderate degree. Reuss draws a circle covering the region to be investigated with the source of smoke as the center, divides the area into 8 sectors and examines these. The injury of sulphurous gases is increased by dampness and lessened by dryness. Contact of plants with fluedust has no injurious effect; any metallic poison found in the plants is due to soluble salts having been formed in the soil and taken up by the plants; its effect is only very slight. Needles and twigs of pine trees at Freiberg showed in 100,000 parts: Pb, 5 to 50; As, 3.3 to 14.3; SO_2, 62 to 120.

Salt Lake Valley.—W. C. Ebaugh[1] studied the effects of smelter smoke upon the Salt Lake Valley with its four smelting plants. Examining the the atmosphere, he found[2] that for 60 per cent. of the total time the air was free from sulphur dioxide, that for 11 per cent. it contained one part dioxide in 1,000,000 parts air; for 20 per cent., from two to four parts; for six per cent., four to six parts; for two per cent., seven to ten parts; and for 1 per cent. of the time, over 10 parts. These figures show that the injurious fumes do not cover the whole region at one time, but that they shift and act more or less upon the different parts of the whole area. The amounts of sulphur dioxide found in the air are exceedingly small, as the gases from lead blast-furnaces contain 0.20 to 0.50 per cent. of it by volume; from copper roasting and blast-furnaces combined, 2.50 per cent.; from converter stacks, 0.20 to 1.20 per cent.; from copper reverberatory smelting furnaces, 0.01 per cent. and less. The researches of Schröder, Schertel and Reuss were confined to pines, firs and other conifers. The author's experiments were made with alfalfa, sugar beet and other plants with succulent leaves. They showed that these required a number of applications of air charged with the amounts of sulphur dioxide found in the district, in order to produce any harmful effects. Spraying sugar beet, by means of an atomizer, with solutions containing 2.91, 2.33, 1.75 and 1.40 grams SO_2 per liter caused no harmful effects; watering, however, was accompanied by corrosion.

Pouring solutions with 1.17 to 0.85 grams SO_2 per liter had a similar effect; weaker solutions were harmless. The effect upon alfalfa was weaker than upon the beet. Spraying and pouring water with less than

---

[1] *Journ.*, Am. Chem. Soc., 1907, XXIX, 951.
[2] *The Mineral Industry*, 1905, XIV, 419.

1.38 gram H_2SO_4 per liter has no deleterious effects; here also alfalfa proved to be more resistant than the sugar beet. The action of sulphurous gases upon the plants investigated is less harmful than was presupposed by their effects upon conifers. Experiments with fluedust gave results which are just the opposite of what is usually accepted, in that its effects were found to be very deleterious. The accompanying table gives some analyses of fluedust.

ANALYSES OF FLUEDUST.

No.	Source	H_2O %	SO_3 sol. in H_2O %	SO_3, Total %	Fe sol. in H_2O %	Fe, Total %	Cu sol. in H_2O %	Cu Total %	Insol. %	Pb %	As %	Zn %
3	Mass of dust broken from stack.	4.1	17.7	33.9	3.6	16.7	1.6	4.2	28.5	0.8	0.83	0.15
4	Flue near roasting furnace.	0.1	6.7	20.6	0.2	8.1	tr.	2.2	31.4	16.3	1.31	4.40
8	Flue from copper blast furnace.	0.7	2.3	31.8	0.0	33.0	–	3.7	27.4	2.4	0.47	0.95
1	From end of long flue-system or from cold tube inserted into flue.	2.0	15.7	26.8	3.3	15.2	1.6	5.1	21.6	6.0	9.75	0.65
2		0.7	14.2	27.3	3.6	18.3	0.5	3.5	22.9	10.7	6.14	0.95
5		0.3	6.1	21.0	0.8	10.3	–	1.0	18.0	19.7	13.52	4.25
6		0.9	6.9	24.4	0.7	9.4	–	0.3	16.6	24.1	10.26	1.25
7		3.3	17.8	29.0	4.2	9.6	0.8	1.5	15.8	15.5	6.00	1.95

Samples of fluedust exposed to water were found to be very hygroscopic; they absorbed in 12 hours 1.5 to 16.2 per cent. water; in 24 hours, 2.8 to 21.8; in 48 hours a little more. The hygroscopic power was found to be roughly proportional to the content of sulphur trioxide and therefore to the corrosive power. Spraying sugar beet and alfalfa with water containing 3, 2, 1, and 0.5 per cent. dust caused corrosion; dusting with a mixture of soil and 20 per cent. dust corroded the leaves severely, while with 10 per cent. no harmful effects were noticed; i.e., the corrosive effect of the dust may be neutralized by the soil. Testing the injured beets and alfalfa showed that the injured beets yielded about as much sugar as the uninjured, and that the alfalfa had good value as fodder. Therefore the harm done to sugar beets and alfalfa by smelter smoke is not as great as is usually held.

W. D. Harkins and R. S. Swain[1] investigated the determination of arsenic and other solid constituents in smelter smoke, and studied the effects of high stacks and of large condensing flues. The present paper is the first instalment. Their work was carried on at the Washoe smelter, from the stack of which 1,733,500,000 cu.ft. gas, under standard conditions, pass off into the air in 24 hours. It travels at a velocity 52.88 ft. per sec., carrying with it As_2O_3, 59,270 lb.; Sb_2O_3, 4320 lb.; Cu, 4340 lb.;

[1] *Journ.*, Am. Chem. Soc., 1907, XXIX, 970.

Pb, 4775 lb.; Zn, 6090 lb.; $(Fe, Al)_2O_3$, 17,840 lb.; Bi, 880 lb.; Mn, 180 lb.; SiO_2, 10,260 lb.; SO_3, 447,600 lb.; and SO_2, 4,636,000 lb. These data show that the extended system of large-sized flues collects much copper, but allows considerable amounts of arsenious oxide to escape. The collected fluedust assayed Cu, 4.64 to 8 per cent., and As_2O_3, 26.06 to 7.14 per cent. The effects of an increased hight of stack have been that the arsenic is spread over a greater area and that the sulphur dioxide is more dilute when it reaches the ground. Samples of gas taken 2½ miles away gave 0.003 to 0.130 milligrams As_2O_3 per cu.ft. Of the arsenious oxide in the smoke, 89 to 92 per cent. is soluble in water; of that settled on the grass, 84 to 93 per cent, is soluble in the digestive fluid of the cow; as a rule the arsenious oxide on the grass is less soluble in water than that of the smoke, as rain washes away the more soluble portions.

Absorption by Vegetation.—J. K. Haywood[1] studied several instances of the injury to vegetation and animal life by smelter fume. At Redding, Cal., it was found that the leaves of 80 per cent. of the injured trees examined contained more sulphur trioxide than the uninjured; at Ducktown, Tenn., the figure was 94 per cent.; at Anaconda, Mont., 92 per cent. In regard to arsenic, the grasses near Anaconda showed the following amounts: Two miles northeast, 78 parts in 1,000,000; four miles northeast, 21 parts; 10 miles northeast, 53 parts; three miles east, 32 parts; six miles east, 42 parts; 6½ miles west, 42 parts. Cattle eating the grass will consume 3 to 10.9 grains arsenic per day.

Discussion of Remedies.—L. S. Austin[2] reviews in a general paper the articles by Ebaugh and Haywood, just discussed, and evaluates the remedies so far proposed to ameliorate the evil.

Fumes in Alkali Districts.—F. W. Traphagen[3] in a brief general discussion of the smoke problem gives it as his belief: (1) That smelter smoke in the alkali waste lands surrounding Anaconda is not only harmless, as metal salts are precipitated, but may even be beneficial as the sulphurous acids neutralize the alkali; (2) that the gases coming to the ground are so dilute as to be harmless; (3) that the crops on the land were not damaged; (4) that only insignificant amounts of arsenic could be detected in animals said to be "smoked." Five tables of analyses of soils and air are given in the paper. The same author[4] reviews the recent litigations of smelter fume and other damages from tailings and tailing-waters.

Defense of Smelters.—A correspondent[5] writing about smelter fume brings out two interesting legal points frequently invoked by the defendant

---

[1] *Journ.*, Am. Chem. Soc., 1907, XXIX, 998; also *Bulletin* No. 89, Bur. of Chemistry, U. S. Dept. of Agriculture.
[2] *Min. and Sci. Press*, 1907, XCV, 649.
[3] *West. Chem. and Met.*, 1907, III, 46.
[4] Amer. Mining Congress, 9th session, Denver, Colo., 1906, Papers and Addresses, p. 132.
[5] *Min. and Sci. Press*, 1907, XCV, 90.

in a suit; they are, that the smelter is the greatest benefit to the greatest number, and that the location of the smelter is the one best suited for the industry. These points appear to have found application in the refusal to grant an injunction against the Washoe smelter,[1] it being held that to close the plant would be a greater damage to the farmers than its continuance as now operated, as the company was able to pay and respond for all real damages sustained, and that the location was best suited for the mines and for the sale of the farm produce.

Disposition of Fumes.—H. Lang[2] treats of a new method of disposing of smelter fume. To smelting plants so situated as not to be in a position to manufacture sulphuric acid, the only way out of the difficulty has been to erect high chimneys and discharge the fumes at such an elevation into the atmosphere that when they come to the ground they will be so diluted as to be harmless (the experience at Freiberg shows that this is not always successful). The new idea is to convey the fume to a distant sterile unsettled region. The example of Alston Moor, England, is cited, where smoke was conveyed in large conduits by natural draft for a distance of five miles, and the proposition is made to use small conduits and employ power. Such a flue would be of wood and A-shaped; the earth would form the bottom, the side boards would be battened on the outside. Such a conduit of 24 sq.ft. crosssection would cost $11,000 per mile, and it would convey per min. 25,000 cu.ft. of air cooled to 50 deg. C. by having passed through steel dust-chambers and flues. At every mile would be installed a Sturtevant fan driven electrically from a parallel line of wire. At the lower end, the flue would be doubled so as to admit cutting out one side for cleaning, as here most of the dust would fall out, while cleaning doors spaced at short intervals would be sufficient for the single flue. A cylindrical steel flue of No. 8 sheets, 5 ft. diameter, would have an efficiency equal to a triangular one 8 ft. high and 6 ft. wide, but it would cost $15,000 per mile. There would be required 32 h.p. for either form. A flue of brick work or concrete, although more expensive than steel, may be considered for a permanent plant.

Estimation of Dust in Air.—Gemünd[3] describes the John Aitken apparatus for estimating dust in air, and suggests its application for estimating the amounts of dust and soot contained in furnace gases. The functioning of the apparatus is based upon the fact that the water vapor of saturated air can be fully condensed only if particles of dust are present which form nuclei for the vapor to collect upon. There is an ingenious device for counting the droplets of water and the included particles of dust. The mode of operating is to dilute with pure air the charged gases to

---

[1] *Eng. and Min. Journ.*, 1907, LXXXIV, 750.
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, 1227.
[3] *Braunkohle*, 1907, VI, 30.

be examined, to saturate the mixture with water-vapor, and then, cooling it suddenly, to count the fine droplets of water and dust.

Restrictions in Germany.—A. Gradenwitz[1] discusses the smoke question in Germany. The regulations for gases passing into the open are: That escaping gases shall not contain over 5 per cent. SO_2; that the chimney through which they are discharged shall be at least 333 ft. high; that a sworn officer make daily analyses (Orsat apparatus); that apparatus be provided for making complete analyses; that records be kept showing amounts of ore treated, percentage of sulphur in raw and roasted ore, and amounts of lime used in absorbing apparatus.

English Requirements.—E. Walker[2] treats of the smelter-fume question in England. The paper deals more with zinc and copper than with lead plants; the maximum sulphur dioxide allowed to escape into the air is 1.5 grains per cubic foot of gas.

Collecting Lignite Dust.—W. Henkel[3] describes a new principle for separating out and collecting the large amounts of dust formed in briquetting lignites in Germany. The idea may find application in the collection of fluedust.

New Processes.

Economy of Fuel.—E. Dedolph[4] patented a process for the treatment of mixed sulphide lead and copper ores in which the ore is ground, mixed with finely divided fuel (sawdust), roasted to oxidize the sulphur, consuming the fuel only partially, and smelted. The ore thus prepared is expected to require about 30 per cent. less fuel than if it were smelted raw.

Electrolytic Sulphide Reduction.—E. S. Anderson[5] patented a process for the electrolytic reduction of copper or lead sulphides. The electrolyte is silicon fluoride. The tank contains two carbon electrodes, one (A) is in contact with sheet copper, the other (B) forms one side of a pocket for the ore while a diaphragm (a wooden frame covered with canvas) forms the other. The current at first passes from A to B; by cathodic reduction the sulphur of the ore is converted into hydrogen sulphide which passes off; upon reversing the current the lead, or the copper, is dissolved at B and deposited upon A.

Treatment with Sulphur Dichloride.—C. E. Baker and A. W. Burwell[6] patented a process for treating zinc-lead sulphides in which the finely-pulverized ore is treated in a heated revolving barrel with SCl_2, when

---

[1] *Eng. and Min. Journ.*, 1907, LXXXIV, 311.
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, 1134.
[3] *Braunkohle*, 1906–07, V, 683.
[4] U. S. Patent No. 870,668, November 12, 1907.
[5] U. S. Patents No. 846,642, March 12, 1907; 862,871, Aug. 13, 1907.
[6] U. S. Patent No. 841,102, Jan. 15, 1907.

fused lead and zinc chlorides are formed and sulphur vapor is set free. The vapor is recovered; the fused chlorides are electrolyzed with graphite electrodes when lead, resp. zinc, is deposited on the cathode, and the chlorine set free at the anode is utilized again in preparing sulphur dichloride.

Electrolytic Decomposition of Galena.—Another process is that of O. Fronck.[1] E. F. Kern and H. S. Auerbach[2] carried on some laboratory tests upon the electrolytic decomposition of galena at an elevated temperature. Introducing the subject, they review some of the processes carried out or proposed, enumerate the conditions an electrolyte ought to fulfil to meet requirements, give results of melting tests of a number of fluxes. record their results and draw conclusions; to all of which the reader must be referred.

Preparation of Lead Compounds.

Lead Oxide.—C. P. Townsend[3] patented a process for producing lead oxide, electrolytically. In the cell, anode and cathode are separated by a diaphragm of parchment paper; the anolyte is a 10-per cent. solution of sodium acetate, the catholyte a 2.3-per cent. solution of sodium hydroxide. The temperature is 60 deg. C.; the current density 20 amperes per sq.ft. of electrode surface; the potential difference 2.1 volts. The electrodes are of sheet lead; the anode department is large, the cathode department small. While electrolyzing, yellow crystals of lead oxide appear in the anode department. They remain suspended in the electrolyte, and are separated out by filtering and dried at 100 deg. C. Their physical properties differ from the lead oxide produced in the dry way, in that they darken and disintergrate when exposed to light, become hydrated in the presence of water, and change into ordinary lead oxide at a red heat.

Basic Lead Sulphate.—Chevalier[4] proposes to blow air through molten galena and thus to produce a new white paint, $Pb_3S_2O_9$, probably a mixture of lead oxide and sulphate, the properties of which resemble that of the normal sulphate.

Lead White and Lead Chromate.—E. D. Chaplin[5] patented a process for producing lead white and lead chromate electrolytically.

Lead Arsenate.—C. D. Vreeland[6] patented an electrolytic process for producing lead arsenate.

---

1 U. S. Patent No. 845,868, March 5, 1907.
2 *Sch. Mines Quart.*, 1907, XXIX, 63.
3 U. S. Patent No. 867,320, October 1, 1907.
4 *Rev. Chimie Industrielle*, 1907, XVIII, 197.
5 U. S. Patents No. 871,161 and 871,162, November 19, 1907.
6 U. S. Patent No. 870,915, November 12, 1907.

Lead Pigments.—Patented process by E. A. Sperry.[1] Discussion of zinc-pigment by E. W. Buskett,[2] L. S. Hughes,[3] and P. Barker.[4]

Desilverization.

Pattinson Process.—L. S. Austin[5] discusses the Tredinnick modification of the steam-Pattinson or Luce-Rozan process referred to already in these reviews.[6] The improvement, it will be remembered, consists in returning to the original Pattinson principle of having a kettle for each product, and then of supporting each kettle by a hydraulic ram so that it can be raised when silver-lead is to be drawn off, and lowered when this is to be received. Each of the series of twelve 50-ton pots is gas-fired and surrounded by a brick-lined iron casing. The stock of lead needed for the plant is 300 tons, the rich-lead assays 300 oz., the market-lead 0.5 oz. silver per ton. In view of the fact that Missouri lead with 2 to 3 oz. silver per ton is being desilverized down to $\frac{1}{8}$ oz. by the Parkes process with one zinking, at the works of the St. Louis Smelting and Refining Company, Colinsville, Ill., the recovered silver paying for the cost and the profit lying in the higher grade of lead produced (Pb,99.99 per cent.), it does not seem unreasonable to expect that with this new modification of the original Pattinson process, by means of which 12 separations are said to be possible in eight hours, instead of five in two hours with the Luce-Rozan process at Eureka, Nev., Missouri lead may be desilverized and purified with profit, even if more silver and impurities remain in the Pattinson than in the Parkes lead.

Parkes Process.—In the accompanying Engraving, Fig. 3, is represented in two vertical sections the oil-fired Faber du Four furnace in use at the desilverizing works of the American Smelting and Refining Company, Maurer, N. J.[7] The trunnions from which the furnace is swung, and their supports, are not shown. The retort resting on a central pillar is protected from oxidizing effect of the oil-flame by a carbon shield which is part of an old retort; the bottom of the furnace is tamped with brasque, which slopes from back to front and serves to catch any leakage of rich lead; at the side is the Billow burner; the oil flame travels around the retort and the products of combustion pass off through a flue in the roof. In some furnaces the cast-iron bottom is convex instead of straight, as shown in the figure, in order to furnish additional space for the free development of the flame.

---

[1] U. S. Patent No. 867,435, October 1, 1907.
[2] *Eng. and Min. Journ.*, 1907, LXXXIII, 760.
[3] *Ibid.*, LXXXIV, 356.
[4] *Min. Reporter*, 1907, LVI, 217.
[5] *Min. and Sci. Press*, 1907, XCIV, 89.
[6] *The Mineral Industry*, 1900, IX, 457; 1902, XI, 851.
[7] *Eng. and Min. Journ.*, 1907, LXXXIII, 84.

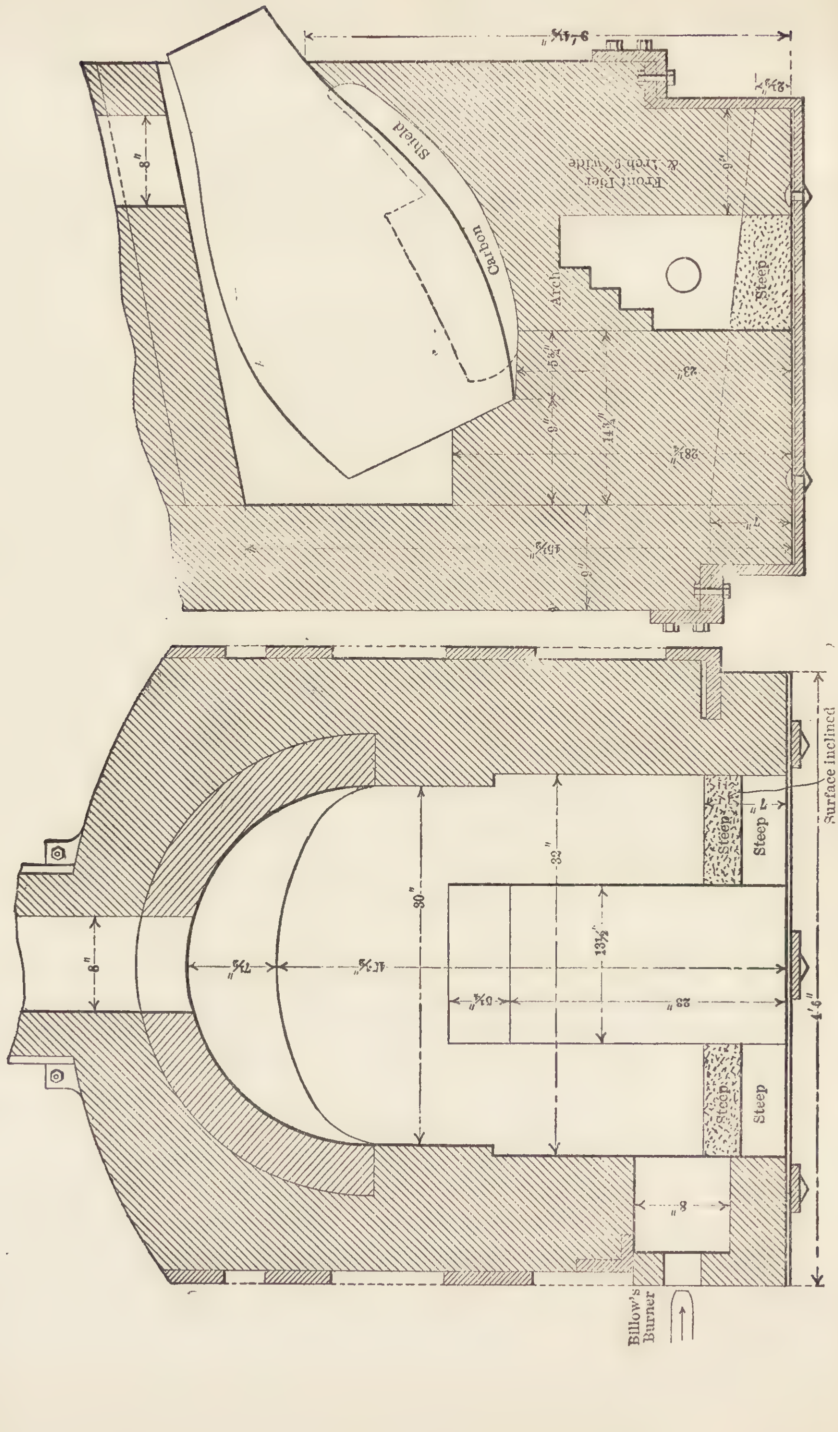


FIG. 3.—FABER DU FAUR FURNACE.

Cupellation.—T. K. Rose[1] gives some facts about the losses of gold by volatilization. Moissan[2] found that gold boiled at about 2530 deg. C. and lost as much as 50 per cent. of its weight in a few minutes. Rose[3] in 1893 had proved that gold began to become volatile just below 1100 deg. C., that the loss amounted to 0.02 per cent. at 1200 deg. C., with a charge of 1200 oz.; also, that standard copper-gold was more volatile than pure gold. It is now shown that the volatilization is governed by temperature, exposed surface of metal, and time; is increased by the passage of a current of air over the surface, also by the presence of impurities. The ordinary loss in a common melting furnace rarely exceeds 0.01 per cent. with a charge of 1200 oz. Dust-chambers have been erected in the mints of London, Sydney and Philadelphia[4] and have collected much gold.

Betts Process.—A. G. Wolf[5] published a paper on the electrolytic refinery of the Consolidated Mining and Smelting Company at Trail, B. C., which uses the Betts process. The plant was erected in 1902,[6] enlarged in 1904 and 1906, and now has 240 tanks with a daily capacity of 70 tons refined lead. There are four divisions, the melting, the electrolytic refining, the treatment of slimes, and the parting of doré silver, to which belongs the working-up of copper-sulphate solution. The base bullion (Ag,100 oz.; Au,1 oz.; As,0.2 per cent.; Sb,0.8 per cent.; Cu,0.25 per cent.) is melted down in two 50-ton kettles, drossed, cast by means of a pump into anodes ($1\frac{1}{2}$x$31\frac{1}{4}$x26 in.) having the usual lugs and weighing 300 lb. The anodes are transferred by a hand-crane from the molding car to a straightening table and then placed upright on cars to be run to the tank room. This is 205x50 ft., has 240 tanks forming a pair of double rows; the tanks are arranged in series, the electrodes in parallel. The electrolyte flows from tank to tank in each row. A tank, 84 in. long by 30 in. wide by 41 in. deep, is made of 4-in. Douglas fir, excepting the partition between a pair which is 5 in. thick. The planks have a dove-tail groove, are coated with a $\frac{1}{4}$-in. layer of asphalt (melting point 40 to 50 deg. C.) and are tied by iron rods. The sloping floor is covered with P. & B. roofing paper; this is coated with P. & B. paint and covered with a $\frac{1}{4}$-in. layer of asphalt; the drainage goes to launders ending in a settling-tank. The electrolyte contains 8 to 11 per cent. SiF_6, 4.5 to 5.5 per cent. Pb, and 0.5 lb. glue per ton of lead refined; the temperature is 30 deg. C. The cathodes are sheets of lead 26 in. wide by 36 in. long by $\frac{3}{16}$ in. thick, and weigh 20 lb. Cathode lead is melted in a 10-ton kettle; the cathodes are cast by ladling lead into a horizontal trough

[1] "Annual Report of the Royal Mint," London, 1906 p. 60; *Eng, and Min. Journ.*, 1907, LXXXIV, 297.
[2] *Compt. Rend.*, 1905, CXLI, 977.
[3] *Journ.*, Chem. Soc., 1893, LXIII, 714.
[4] Report of the Director of the Mint, 1906, p. 30.
[5] *West. Chem. and Met.*, 1907, III, 83.
[6] *The Mineral Industry*, 1902, XI, 453; 1903, XII, 266; 1905, XIV, 421.

hinged to an inclined (15 deg. from horizontal) plate, turning over the trough and discharging the liquid lead upon the plate. The tanks, holding 20 anodes and 21 cathodes, are charged and discharged by a 5-ton Niles crane. The area of an immersed anode is 11 sq.ft.; each cathode is suspended from a copper cross-bar 1x0.5 in. The current density is 14 amperes per sq.ft. cathode area; the difference in potential between anode and cathode is 0.3 volt. An anode is corroded in eight days, a cathode is exchanged every four days. In discharging, the cathodes are removed to washing tanks, freed from adhering electrolyte and slime, transferred to the melting room, melted down in a 50-ton kettle where the lead is pumped into 80- to 100-lb. molds placed in a semicircle. An average analysis of refined lead shows: Ag, 0.0014; Cu, 0.0003; Fe, 0.0013; Sn, 0.301; Sb, 0.306; Zn, 0.001; Au, Mn, As, Ni, Co, Cd, Bi, absent; Pb, by difference 99.9962 per cent. After the cathodes have been removed, the anodes with their 1.5-in. layer of adhering slime go to the washing tanks, are scraped and brushed and return in the form of 15-per cent. scrap to the bullion melting kettles. A tank is re-charged after allowing 15 to 20 min. for the slime to settle. Every 24 days a tank is cleared from slime. The slime from the settling tanks assays: H_2O, 14.5; Au, 0.1; Ag, 17.1; Cu, 9.5; Sb, 25.91; As, 5.96; Pb, 14.5. It is transferred in copper cars to one set of washing tanks (6x8x6 ft.) of concrete painted with asphalt, washed with water stirred by steam, settled, etc., transferred to a second and third set of tanks (8 ft. by 3 ft. 8 in. by 2 ft.), removed to a filter bed (8x8x2 ft.), covered with a perforated hard-rubber sheet, filtered with steam-suction, transferred to a drying car and then run into the dryer, which consists of a hood 6 ft. wide by 13 ft. 10 in. long having a fire-place and sliding doors to admit two cars. The dried slime is mixed with soda, melted down in a water-jacket reverberatory furnace lined with magnesia brick, and thus converted into bullion, 956 to 960 fine, which is cast into bars weighing 38 lb., to be parted with sulphuric acid. The copper sulphate plant treats the liquors resulting from the decomposition of silver sulphate by means of metallic copper. The electric generator of the plant is a 440-k.w. machine of the Canadian General Electric Company. It is proposed to construct a plant for preparing silicon fluoride, and for recovering antimony electrically from the slime.

Electro-Deposition of Lead.—R. S. Snowdon[1] found that in precipitating lead electrically from an acetate solution a smooth deposit could be obtained by using a rotating cathode speeded to about 2500 r.p.m. with a current density of 0.5 ampere per sq. decimeter, but not more. Using one gram gelatine per liter electrolyte, a cathode density of 1.5 ampere per sq. decimeter still gave an adherent deposit.

---

[1] *Journ. Phys. Chem.*, 1906, X, 500.

Refining of Doré Silver.—C. W. Lee and W. O. Brunton[1] have put into practical operation the results of the researches of Rose[2] for refining low-grade, precious-metal bullion by forcing air through it with a clay tube while holding it molten in a crucible. Thus cyanide bullion with Au,647; Cu, 163; Zn, 13; Ag, 88; Pb, 74; Ni, 2; total, 640; was brought to Au, 819; Ag, 107; total, 819 fineness by pumping air through the molten charge for two hours. In a similar way 1360 oz. cyanide bullion assaying Au, 778.1; Cu, 71; Ni, 9.2; Ag, 62.6; Pb, 47.1; Zn, 7.1; was refined by a few hours treatment to Au, 816; Ag,166; total, 872 fineness. A drawing of the apparatus and a description of the mode of operating are given in the original.

[1] *Journ.*, Chem., Met and Min. Soc. of South Africa. 1906-07, VII, 358; 1907, VIII, 26, 121.
[2] *The Mineral Industry*, 1905, XIV 420.

LIMESTONE.

The limestone industry in the United States has grown to enormous proportions. In addition to the amount consumed for metallurgical purposes and in the manufacture of lime, immense quantities are used for building and roadway material, in the manufacture of cement and as a base for chemical manufactures. It would be difficult to collect statistics of production of the limestone used for all purposes; only the production of that used as flux will be dealt with in these paragraphs.

Flux.—Limestone as a flux is used chiefly in the smelting of iron, copper and lead. The pig iron production in the United States in 1907 was 25,781,361 long tons. For the production of 2240 lb. of pig iron there is required an average of 1150 lb. of limestone. This indicates a consumption of 14,824,000 short tons of limestone for iron flux. The average value of this limestone may be reckoned at 45c. per ton.

The bases for estimates in connection with lead and copper smelting are less well established. The bulk of the non-argentiferous lead is smelted from ore having a calcareous gangue and requires no lime flux. The total production of desilverized lead of domestic origin in 1907 was 222,997 short tons. It may be assumed that an average of 10 tons of ore had to be smelted to obtain one ton of lead, which indicates that there were 2,229,970 tons of ore smelted. We believe that these figures approximately represent the magnitude of the silver-lead industry in the United States, being too low rather than too high. Also, a certain quantity of ore of foreign origin is omitted. In present American lead-smelting practice, the average requirement for lime flux is about 16⅔ per cent. of the tonnage of ore smelted, indicating a consumption of about 371,661 tons of limestone for lead-smelting flux in 1907, the average value of which may be reckoned at 90c. per ton.

In connection with copper smelting, we estimate that 658,924,725 lb. of copper derived from districts other than Lake Superior during 1907 was obtained from ore yielding an average of 100 lb. of copper per ton, which indicates about 6,589,250 tons of ore smelted. Reckoning the consumption of lime flux at 8 per cent. of this tonnage, we arrive at 527,140 tons, the average value of which may be reckoned at 90c. per ton.

The above estimates, which are admittedly rough, together with those

for 1905 and 1906, are summarized in the accompanying table, the values being based on the prices per ton given in the above paragraphs.

CONSUMPTION OF LIMESTONE FLUX IN THE UNITED STATES.

Items.	1905		1906		1907	
	Sh. tons.	Value.	Sh. tons.	Value.	Sh. tons.	Value.
Consumed for iron flux	13,220,000	$5,949,000	14,552,000	$6,548,400	14,824,000	$6,670,800
Consumed for lead flux	357,000	321,300	379,300	341,370	371,661	334,495
Consumed for copper flux	521,000	468,900	554,839	449,355	527,140	474,426
Total	14,098,000	$6,739,200	15,486,139	$7,339,125	15,722,801	$7,479,721

Limestone in the United States.

Short notes on the limestone industry in several States are given in the following paragraphs.

Michigan (By A. C. Lane).—The limestone quarries which are furnishing limestone for portland cement and chemical uses produced a large tonnage in 1907. The Sibley and Anderdon quarries near Detroit, the quarries around Alpena and Little Traverse Bay, and two quarries which find high-grade limestone in the Niagara, west of Trout Lake junction in the Upper Peninsula, all had a prosperous year.

Missouri (By E. R. Buckley).—The limestones of this State are many of them remarkable for their freedom from impurities. For this reason they are used extensively in the manufacture of quicklime. Where shale of the desired composition occurs near by they are most desirable for use in the manufacture of portland cement. There are now 35 lime plants and three portland cement factories operating in the State. In addition to these there are two portland cement factories in process of construction and at least three more projected. The operating plants are situated near St. Louis, Kansas City and Hannibal. They have a combined capacity of about 6,000,000 bbl. per annum.

The limestone quarries at Carthage have developed rapidly during the last year, and the stone is now being used for many new purposes, especially ornamental. The stone has a beautiful white color and its thoroughly crystalline texture renders it well adapted for many purposes for which other building stones in the State are not suited.

New York (By D. H. Newland).—The limestone produced in New York during 1907 is valued at $3,182,447. The marble industry was specially active in 1907, and the production, valued at $1,571,936, has probably never been exceeded in the State. The stone quarries are distributed among all the counties practically, while they yield nearly every kind of material for building, construction or ornamental purposes.

West Virginia (By G. P. Grimsley).—This State ranks fifth in the production of limestone flux, with the promise of a rapid development in this industry in the next few years. Limestone is used in small quantities as a flux in cupolas where the pig iron is melted for use in bessemer converters, and on a large scale in the reduction of iron ores in the blast furnace. Limestone for use in the manufacture of open-hearth steel, low in silica and phosphorus, is found in the eastern part of the State, at Martinsburg. This rock carries on an average, 98 per cent. lime carbonate, 0.5 per cent. silica, and 0.02 per cent. phosphorus.

There are several quarries operating in the neighborhood of Martinsburg, Parkersburg, Bunker Hill, near the main line of the Baltimore & Ohio Railroad, and the Cumberland Valley division of the Pennsylvania Railroad. Ballast and lime quarries are operated in the vicinity of Martinsburg, Bakerton, Engles Siding, Millville, and in the southern part of the State; also on the Chesapeake & Ohio Railroad at Frazier, near Fort Spring. There is also a quarry and crushing plant about eight miles above Morgantown, on the Morgantown & Kingwood Railroad.

Wyoming (By H. C. Beeler).—The production of limestone in Wyoming for 1907 was 25,000 tons, which was shipped to the beet sugar factories of Colorado, from the vicinity of Hartville, in northern Laramie county. Similar ledges exist throughout the State, and many new quarries are being opened up.

LITHIA.

The lithium minerals of commercial importance are spodumene, lepidolite and amblygonite. Mining is carried on only in California and South Dakota, although deposits are known to exist in Maine and Connecticut. The principal deposits of California are situated in San Diego county, and few of them are worked steadily. Heretofore the bulk of the California product has been shipped to Germany for chemical treatment, but a movement is now on foot to erect a chemical plant within the State for the manufacture of lithia compounds.

STATISTICS OF LITHIUM ORE AND SALTS IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year.	Production. (*b*)		Imports. (*c*)			Production. (*b*)		Imports. (*c*)	
	Tons.	Value.	Pounds.	Value.		Tons.	Value.	Pounds.	Value.
1902......	1,245	$25,750	21,216	$22,951	1905......	79	$1,412	*Nil.*	....
1903......	1,115	23,425	5,596	3,669	1906......	383	7,411	*Nil.*	....
1904.......	577	5,155	19	48	1907......	530	11,000	60	$100

(*a*) Statistics of the U. S. Geological Survey. (*b*) Ore. (*c*) Lithia salts.

The principal development in the industry during 1907 was the opening of several mines in Pennington county, S. D., where spodumene, the silicate of lithium, is abundant. Interest centered chiefly in and around the camp of Keystone. The largest producers were the Etta mine, which shipped 200 tons of spodumene; and the Provident Mining Company, which produced about 240 tons of amblygonite. The Etta mine is under lease from the Harvey Peak Tin Mining and Milling Company, while the Provident company owns the property which it operates.

MAGNESITE.

The magnesite deposits of California, the only ones commercially utilized in the United States, showed little or no change in 1907 from what they did in the previous year. Two new deposits became productive during 1907, one in Sonoma and the other in Tulare county, but their combined output was small. Owing to litigation no mineral was shipped from Madeira mine No. 2, situated on Austin creek, Sonoma county, although several hundred tons were accumulated on the dumps. The California Magnesite Company was incorporated to operate a deposit near Winchester, Riverside county, but no production was recorded from this source during the year.

STATISTICS OF MAGNESITE IN THE UNITED STATES.
(Tons of 2000 lb.)

Year.	Production. (a)		Imports.		Consumption.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897	1,143	$13,671	(b)	....	....	....
1898	1,263	19,075	16,039	$134,130	17,302	$153,205
1899	1,280	18,480	20,807	(e) 174,779	22,087	193,259
1900	2,252	19,333	28,821	(e) 216,158	31,073	235,491
1901	4,726	43,057	33,461	(e) 250,958	38,187	294,015
1902	2,830	20,655	49,786	373,928	52,616	394,583
1903	1,361	20,515	54,776	461,399	56,137	481,914
1904	2,850	9,298	38,704	286,828	41,554	296,126
1905	3,933	16,221	74,374	638,619	78,307	654,840
1906	4,032	40,320	90,396	863,492	94,428	903,812
1907	....	....	99,008	875,359	....	....

(a) Reported by the State Mining Bureau of California. (b) Not reported. (e) Estimated.

Most of the magnesite in California comes from the deposits in Tulare county, near Porterville, where it is mined very cheaply by quarrying, and whence can be shipped by rail to San Francisco at low cost. Calcining furnaces have been built at that point, the product being sold to paper manufacturers who use it in wood-pulp manufacture. The manufacturers of carbon dioxide obtain their crude magnesite from both Tulare and Sonoma counties, and they also sell some of the calcined material to the paper-makers. The deposits in Alameda county, which extend into Santa Clara and Stanislaus counties, are utilized by a company which has a manufacturing plant in Oakland, where some building material and brick are made, but only on a small scale.

Experiments are being made with calcined magnesite as a base for tiles, which, if successful, may cause a greater demand for the material. There are many known deposits of the mineral in California, but generally speaking, only those are utilized which are near the railroad lines, where cheap transportation may be obtained. The consumption is too limited to admit of many deposits of this mineral being advantageously worked.

Market and Prices.—The consumption of magnesite produced in California is confined to the Pacific Coast, as freight rates east are prohibitive when the material must meet the competitive price of the Austrian and Grecian magnesite. The crude may be considered as worth about $3 per ton, or less. The calcined is worth less than formerly; at the close of 1907 about $12 per ton.

Quotations in New York on Grecian magnesia were subject to but little variation during 1907. Crude was steady at $7@8 per 2240 lb. up to the end of the year, when the price was advanced about $1 per ton. During January and February calcined was quoted at $30@35 per 2000 lb.; in March an advance was made to $30@40 per ton, at which price the year closed. These quotations were for comparatively small quantities; on large contracts calcined magnesite can be bought much more cheaply, say for $14@16 per 2000 lb. for the inferior quality. Bricks and domes brought from $160@200 per M., according to quality, f.o.b. Pittsburg. Magnesium chloride opened the year at 90c. @ $1.25 per 100 lb., dropped to 80c. @ $1.15 in April, and closed the year at 80c. @ $1. Sulphate was quoted at 90c. @ $1.25 per 100 lb. early in the year, closing at 90c.@$1.

Magnesite in Foreign Countries.

Africa.—Although this country is not yet a factor in the magnesite market, work is being pushed in the Transvaal and the industry promises to become of importance. There is an extensive deposit on the railroad running from Delagoa bay to Johannesburg, between the stations Kaap-Muiden and Malelane, and another in the vicinity of Salt Creek. Samples from these localities are said to compare favorably with the Grecian magnesite, carrying even smaller amounts of lime, clay and iron. Wood is plentiful and coal can be laid down from the mines at Middleburg for about $2.40 per ton.

Greece.—The principal deposits are in the province of Eubœa, where the industry is largely in the hands of the Société des Travaux Publics et Communaux. This company recently absorbed the Société d'Entreprises. The deposits are about 3 or 3.5 km. from the sea and are very advantageously situated. The occurrence of the deposit is such that mining can be carried on with very little capital. The material is shipped raw or calcined. Although the magnesite is amorphous rather than crystalline,

it is in great demand and prices per metric ton during the summer of 1907 were as follows: 93 per cent., 30 fr. ($6); 95 per cent., also material calcined at 600 deg. C., 90 fr. ($18); calcined at 1000 deg. C., 111 fr. ($22.20). Completely calcined material (1600-1700 deg. C.) was quoted at 130 fr. ($26) and upward.

Italy.—According to a United States consular report, this country produces annually about 1800 tons of sulphate of magnesium, 600 tons of calcined magnesia and 625 tons of magnesite. The Kingdom exports about 850 tons of sulphate of magnesium and imports 950 tons of chloride of magnesium, exporting about one-half of the latter amount. The domestic production appears to suffice for the home consumption and is capable of increase to meet the demand. Italy would not seem to offer a market for foreign magnesia products. The deposits are situated in the province of Turin.

Venezuela.—Consul Thomas P. Moffat, of La Guaira, reports a recent concession granting the exclusive privilege, for a period of 25 years, of exporting magnesite found on private lands in the island of Margarita. The government is to receive one bolivar (19.3c.) on each ton exported. It is estimated that the annual output will be from 12,000 to 15,000 tons. Work has been commenced on several properties, and shipments will probably be made soon. A private contract is said to have been made between the grantee and an American company for the entire quantity to be exported.

THE PRINCIPAL SUPPLIES OF MAGNESITE.
(In metric tons.)

Year.	Austria-Hungary. (*a*)	Germany. (*d*)	Greece. (*e*)	India. (*e*)	United States. (*e*)
1897	(*c*)	53,086	11,311	(*f*)	1,038
1898	(*c*)	50,114	14,829	(*f*)	1,146
1899	(*c*)	60,910	17,184	(*f*)	1,161
1900	(*c*)	67,988	17,277	(*f*)	2,043
1901	(*b*)40,236	67,732	13,410	(*f*)	4,286
1902	(*b*)53,467	58,947	27,103	3,597	2,567
1903	(*b*)69,058	60,834	25,657	838	1,234
1904	(*b*)53,781	65,142	44,828	1,193	2,585
1905	(*b*)92,359	87,585	43,498	2,645	3,568
1906	....	81,481	64,424	1,861	3,658

(*a*) Exports. (*b*) Calcined magnesite. (*c*) Previous to 1901 magnesite was included with other minerals not elsewhere specified. (*d*) Chloride and sulphate of magnesium. (*e*) Crude magnesite. (*f*) Not reported.

MANGANESE.

The manganese products of the United States are supplied chiefly from manganiferous iron and zinc ores, chiefly the former. The mining of true manganese ores is of secondary importance. The minerals of manganese are of widespread occurrence, but in the majority of cases the deposits are pockety or the mineral carries too large a percentage of some objectionable element, such as phosphorus, sulphur, silica, lime, baryta, etc. The domestic industry is further handicapped by the superiority of foreign ores and the small cost at which they can be laid down in this country. In Russia, Brazil and India, working costs are low and the ore may be shipped to the United States as ballast at a low rate and duty free.

PRODUCTION OF MANGANESE ORES IN THE UNITED STATES. (*a*)
(Tons of 2240 lb.)

Year.	Manganese Ores.				Manganiferous Iron Ores.				Man. Zinc Ores.	Total Production.	
	California.	Georgia.	Virginia.	Other States.	Arkansas.	Colorado.	Lake Superior.	Va. & N. Car.	New Jersey.	Long Tons.	Value.
1896..	318	2,538	1,588	*Nil.*	3,038	9,072	110,317	....	35,655	162,526	$339,083
1897..	450	962	2,408	190	4,430	18,600	80,260	....	50,000	(*b*)158,600	328,176
1898..	393	2,477	3,307	1,250	2,775	17,792	112,318	....	47,470	187,782	416,627
1899..	263	1,623	3,626	105	855	29,161	53,702	....	53,921	143,256	306,476
1900..	131	3,447	7,881	312	*Nil.*	43,393	75,360	*Nil.*	87,110	217,546	1,172,447
1901..	610	4,074	4,275	3,036	*Nil.*	62,385	512,084	20	52,311	638,795	1,644,117
1902..	846	3,500	3,041	90	*Nil.*	13,275	884,939	3,000	65,246	973,937	2,145,783
1903..	16	500	1,801	508	*Nil.*	14,856	566,835	2,802	73,264	660,582	1,670,349
1904..	60	*Nil.*	3,054	32	600	17,074	365,572	*Nil.*	68,189	454,581	789,132
1905..	1	150	3,947	(*e*)20	3,321	45,837	720,098	*Nil.*	90,289	863,663	1,681,472
1906..	1	.....	6,028	892	8,900	32,400	1,000,000	*Nil.*	93,461	1,141,681	(*e*)3,403,993
1907..	100	*Nil.*	4,604	900	4,133	99,711	(*e*)1,120,000	*Nil.*	95,423	1,324,871	*e*) 3,860,265

(*a*) Statistics of 1900 and subsequent years are by the U. S. Geological Survey. (*b*) Includes 1300 tons of manganiferous iron ore from Vermont. (*e*) Estimated.

MANGANESE MINING IN THE UNITED STATES.

Arkansas.—Manganese deposits are known to exist in several localities in the State, but mining is not carried on to any great extent. Toward the end of 1907 it was reported that capitalists had purchased a tract of 2000 acres of manganese-bearing land near Cushman, and that development work would be undertaken in 1908. Braunite (Mn_2O_3) is found in the deposits at Batesville, in the north central part of the State.

California.—Extensive deposits of manganese ore are known to exist near Livermore, Alameda county, while development to a less extent has been prosecuted in Riverside, Sonoma, San Luis Obispo, Santa Clara and Mendocino counties. In 1905 and 1906 there was practically no production from the State, but in 1907 interest was revived to a small degree. Early in the year shipments of rhodonite, a manganese silicate, taking a high polish and suitable for an ornamental stone, were made from Fresno county to New York City. Development work will be prosecuted on what is thought to be an important manganese discovery near Stony Ford, in Glenn county.

Colorado.—At Leadville several of the mines produce a manganiferous iron ore, carrying a small amount of silver, which is bought chiefly by the silver-lead smelters of the State for fluxing purposes, but the steel manufacturers buy a considerable quantity of it. The principal shippers of this class of ore are the Morocco Mining Company, the Morning Star and the Evening Star companies. The market in 1907 was exceptionally good; shipments now amount to about 4000 tons per month, for which the producers receive a net price of $3 per ton. Formerly much of this manganese ore was shipped to the Illinois Steel Company at Chicago, but the competition of the Cuban manganese producers destroyed that market. Shipments continue, however, to the Pueblo plant of the Colorado Fuel and Iron Company.

Georgia.—Although the manganese industry never attained any great importance in this State, there are several deposits which are worthy of attention. A favorable feature is the close proximity to the iron and steel centers of the South. During the summer of 1907 the Iron Mountain Mining Company was organized in Atlanta for the purpose of operating deposits of iron and manganese ores in Murray county. Washeries will be erected and a branch road built to the property.

Michigan and Wisconsin.—A large proportion of the hematite of the Lake Superior ranges carries manganese in quantities ranging from 1 to 8 per cent. The best grade of ore of this class comes from the Gogebic range, and is utilized in the production of spiegeleisen. On the Menominee and Mesabi ranges, with one or two exceptions, the percentage of manganese carried in the ores is low. The same may be said of the Marquette and Vermillion ranges.

Missouri.—The discovery of a large tract of manganese-bearing land in the vicinity of Salem, and within eight miles of rail transportation, was reported late in 1907. The ore is said to contain 49 per cent. manganese, 1.75 per cent. nickel and cobalt, and a trace of phosphorus.

New Jersey.—The Franklin mine of the New Jersey Zinc Company is the only producer of manganese ore in this State.

Tennessee.—The manganese deposits near Del Rio will be developed

by the Del Rio Mining Company, which was organized in October, 1907. Plans have been made for the erection of a plant capable of producing 50 tons of crude and ground manganese ore daily.

Virginia.—The most important deposits are found in the Shenandoah valley, in the northwestern part of the State, and in Campbell county, in the central part of the State. The Metallic Alloys Company, a newly organized corporation, purchased the property and plant of Kendall & Flick, at Elkton and Lyndhurst. Additional power and equipment were installed at Elkton and the capacity of the plant increased from 10 to 25 tons daily. The product of this mine is suitable for use in the manufacture of brick, tile, glass, pottery, etc. The Piedmont Manganese Company, of Lynchburg, produced a small amount of high grade ore during 1907. At the present time from 25 to 50 tons are being mined daily and preparations are being made to increase the output. This company is said to be operating on an exceptionally promising deposit. Development work has extended to a depth of 80 ft. and the ore is still rich and plentiful. Mining can be economically conducted and wood and water are abundant in the vicinity. The Norfolk & Western Railroad passes within three-quarters of a mile of the mine. The following is an analysis of the last shipment: Silica, 0.93 per cent.; iron and alumina, 2.63; lime, 0.81; magnesia, 0.22; phosphorus, 0.341; manganese, 55.65.

Wisconsin (By W. O. Hotchkiss).—Some fairly good sized bodies of manganese ore, carrying a high percentage of manganese, were developed by drilling in the Baraboo iron district.

CONSUMPTION OF MANGANESE ORE IN THE UNITED STATES.
(Tons of 2240 lb.)

Year.	Imports.		Consumption.		Production of Man. Silver Ores. (*b*)	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1897	119,961	$1,023,824	278,561	$1,352,000	149,562	$424,151
1898	114,885	831,967	302,667	1,248,594	99,651	295,412
1899	188.349	1,584,528	331,605	1,891 004	79,855	266,343
1900	256,252	2,042,361	473,798	3,214,808	188,509	897,068
1901	165,722	1,486,573	804,568	3,130,690	228,187	865,959
1902	235,576	1,931,282	1,209,513	4,077,065	174,132	908,098
1903	146,056	1,278,108	806,638	2,948,457	179,205	649,727
1904	108,519	901,592	563,100	1,690,724	105,278	348,132
1905	257,033	1,952,407	1,120,696	3,633,879	127,170	445,095
1906	221,260	1,696,043	1,362,941	5,100,036	163,760	573,160
1907	208,321	1,793,143	1,533,192	5,653,408	....	....

(*b*) Mined in Colorado and used as flux in silver-lead smelting; not included in the statistics of consumption.

Classification, Uses and Value of Manganese Ores.

Ores.—The chief manganese ores of commercial importance are the oxides, which may be classified according to the amount of oxygen they contain and the presence or absence of water. Polianite and pyrolusite,

the dioxides, are the most highly oxidized ores and are anhydrous; psilomelane and wad contain water. These minerals are black and in most cases yield a black streak. The hardness of polianite is from 6 to 6½, while that of pyrolusite is only 2 to 2½; the latter colors the fingers black and is often impure, containing iron, silica, lime, baryta and other impurities. Psilomelane is found in rounded masses, has a hardness of from 5 to 6 and contains varying amounts of water. The color of this mineral is black; the streak is brownish-black. Wad, or bog manganese, contains a still greater amount of water and is soft and pliable. Braunite, the anhydrous sesquioxide, contains about 8 per cent. silica and is sometimes described as a silicate. It has a hardness of from 6 to 6.5 and is brownish-black both in color and streak. Manganite is the hydrated form of this mineral. Hausmannite (Mn_3O_4) has still less oxygen, a hardness of from 5 to 5.5, and is brownish-black in color; the streak is chestnut brown.

Uses.—Over 90 per cent. of the manganese ore produced is utilized in the iron and steel industries, being smelted to form an alloy with iron, which is used in the manufacture of steel. The amount of manganese in the iron-manganese alloys used in this way varies widely, those containing up to 20 per cent. being known as spiegeleisen, while those containing any greater amount of manganese are termed ferro-manganese. In the manufacture of steel by the bessemer and open-hearth processes, the addition of these alloys serves to prevent oxidation of the iron, removes the small quantities of silicate and oxide of iron usually present and facilitates the combination of the carbon and iron. The amount of manganese consumed per ton of steel produced is approximately, for open-hearth steel 13.5, for mild bessemer steel 16.5, and for bessemer steel for rails 29 lb. per ton. Manganese-iron alloys are used also in the manufacture of special manganese tool-steel of high tensile strength and hardness, and of the chilled cast-iron of unusual toughness employed for the wheels of railway cars.

Some of the manganate and permanganate salts of manganese are employed, on account of their oxidizing powers, as disinfectants and as dryers in oil varnishes, while the natural ores, by reason of their high content of oxygen, are used to decolorize greenish glass by converting ferrous into ferric oxide. Used in larger amounts they impart a fine purple color to glass and pottery. In the United States inferior grades of ore are used for coloring brick. There are several other non-metallurgical applications of manganese, but they are of little importance.

Value.—The Carnegie Steel Company, which is the largest consumer of manganese ore in this country, paid during 1907 the prices quoted below for maganese-iron ores delivered at the Lucy furnaces, Pittsburg, Penn., the Edgar Thomson furnaces, Bessemer, Penn., or the South works of the Illinois Steel Company, South Chicago, Ill. The prices are per long ton

and per unit of metallic manganese: Above 49 per cent. $0.30; 46 to 49 per cent. $0.29; 43 to 46 per cent. $0.28; 40 to 43 per cent. $0.27. Iron is paid for at the rate of 6c. per unit. These quotations are based upon ore containing not more than 8 per cent. silica and not more than 0.25 per cent. phosphorus, and are subject to the following deductions: For each 1 per cent. silica in excess of 8 per cent., 15c. per ton is deducted, fractions in proportion; for each 0.02 per cent., or fraction thereof, of phosphorus in excess of 0.25 per cent., 2c. per unit of manganese is deducted. Ore containing less than 40 per cent. manganese or more than 12 per cent. silica or more than 0.27 per cent. phosphorus is subject to refusal or acceptance at the buyer's option. Settlements are based on analysis of sample dried at 212 deg. F., and the percentage of water found by this drying is deducted from the gross weight of the shipment. These terms were the same as prevailed during 1906.

High-grade ores, used chiefly as a source of oxygen in storage batteries and as a dryer in paints and oils, command higher figures than the ordinary ones used by the steel makers. Prices of high grade ores range from 1 to 20c. per lb., depending upon the ease with which the contained oxygen may be liberated, rather than upon the percentage of this element present. Thus of two samples carrying the same amount of oxygen, one may be much more valuable than the other. The only method of determining this point is by an actual working test of the ore.

Production and Imports of Iron-Manganese Alloys.

Statistics of production and imports of ferro-manganese and spiegeleisen, as compiled by the American Iron and Steel Association, are reported in the accompanying table.

UNITED STATES PRODUCTION AND IMPORTS OF IRON-MANGANESE ALLOYS.
(In tons of 2240 lb.)

	1904		1905		1906		1907	
	Production.	Imports.	Production.	Imports.	Production.	Imports.	Production.	Imports.
Ferro-manganese....	58,022	21,813	66,179	52,841	55,520	84,359	55,918	87,400
Spiegeleisen.........	162,370	4,623	227,797	55,457	244,980	103,267	283,430	48,995
Totals............	220,392	26,436	293,976	108,298	300,500	187,626	339,348	136,395

Manganese Mining in Foreign Countries.

The increasing demand for manganese and the recent diminution of the Russian output have broadened the market for manganese ores in other countries. Although deposits of manganese are to be found in nearly every part of the world, only in Russia, India, Brazil and the United States has the industry become of great importance. An in-

spection of the accompanying table will furnish an idea of the relative importance as producers of the different countries.

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*)
(In metric tons.)

Year.	Australia	Austria-Hungary.	Belgium.	Bosnia (*b*)	Brazil. (*d*)	Canada.	Chile. (*d*)	Colombia.	Cuba.	France.	Germany.	Greece.	India.
1897.	....	10,043	28,372	5,344	16,054	14	23,528	8,382	....	37,212	46,427	11,868	74,862
1898.	....	14,219	16,440	5,320	26,417	45	20,851	11,176	....	31,935	43,354	14,097	61,469
1899.	....	10,484	....	5,270	65,000	279	40,931	10,160	....	39,897	61,329	17,600	88,520
1900.	....	14,550	10,820	7,939	108,244	34	25,715	8,748	21,973	28,992	59,204	8,050	129,865
1901.	368	12,077	8,510	6,346	100,414	447	18,480	95	25,586	22,304	56,691	14,166	122,831
1902.	4,692	12 883	14,440	5,760	157,295	175	12,990	*Nil.*	40,048	12,536	49,812	14,960	160,311
1903.	1,415	11,489	6,100	4,537	161,926	135	17,110	(*c*)	21,070	1,583	47,994	9,340	174,563
1904.	843	15,460	485	1,114	208,260	123	2,324	(*c*)	33,152	11,254	52,886	8,549	152,708
1905.	1,540	23,732	*Nil.*	4,129	224,377	22	1,323	(*c*)	(*d*) 8,096	6,751	51,463	8,171	257,969
1906.	1,131	(*f*)13,402	120	7,651	201,500	84	(*c*)	(*c*)	(*d*)13,997	11,189	52,485	(*d*)9,200	503,686
1907.	1,134	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	74,683	(*d*)9,788	*d* 557,194

Year.	Italy.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	Spain.	Sweden.	United Kingdom	United States. (*e*)
1897........	1,634	15,448	182	1,652	403	263,115	100,566	2,749	609	161,138
1898........	3,002	11,497	220	907	68	329,276	102,228	2,358	235	190,787
1899........	4,356	11,336	137	2,049	747	659,302	104,974	2,622	422	145,548
1900........	6,014	15,831	166	1,971	77	802,236	112,897	2,651	1,384	221,714
1901........	2,181	16,270	208	904	221	522,395	60,325	2,271	1,673	649,016
1902........	2,477	10,844	*Nil.*	(*c*)	4,674	536.519	46,069	2,850	1,299	989,519
1903........	1,930	5,616	71	30	1,341	414,334	26,194	2,244	831	671,151
1904........	2,836	4,324	199	(*c*)	843	430,090	18,732	2,297	8,880	461,854
1905........	5,384	14,017	55	(*c*)	1,541	508,635	26,020	1,992	14,582	877,482
1906........	20,500	54,339	16	22	1,131	1,015,686	62,822	2,680	23,126	1,159,948
1907........	(*c*)	10,410	(*c*)	(*c*)	1,134	(*c*)	(*d*)67,996	(*c*)	16,356	1,324,871

(*a*) From official statistics. (*b*) Includes Herzegovina. (*c*) Statistics not available. (*d*) Export returns. (*e*) Includes output of manganiferous iron ore. (*f*) Austria alone.

Brazil.—Manganese ores are widely distributed in Brazil, occurring in the States of Minas Geraes, Bahia, Matto Grosso, Parana and Santa Catharina. The deposits extensively worked are in the State of Minas Geraes, small quantities only having been shipped from the Bahia mines. The Minas Geraes manganese ores occur in the mining zone of that State which is traversed by the Central Railway system, entering the district at Lafayette 465 km. from Rio de Janeiro. The deposits are found in the Burnier and Lafayette districts, occurring in the former in the metamorphic rocks and associated with limestones and iron ores. In the Lafayette district the ore occurs in fissure veins and has been formed by the decomposition and leaching of a rock containing manganiferous garnets.

The exports of manganese probably exceed in value $1,000,000 annually, it being difficult to trace the exact share the State has in the exports of Brazil as a whole. The exports in 1906 were 124,646 tons. The cost of extracting the ore, even with a minimum of labor and trouble, is great, while transportation charges are high and service correspondingly

poor. Under present conditions the possibility of mining and shipping manganese ore at a profit rests altogether upon whether or not exchange goes materially above 15d. (30c.).

The State of Bahia is rich in deposits of manganese ore of excellent quality, but owing to cost of production and inadequate transportation facilities only one such deposit is being worked in this State at the present time. This deposit is situated in the Nazareth district, about 48 km. to the south of the bay of Bahia. It consists of two mines known respectively as Pedras Pretas and Sapé. Both mines are owned by the local Companhia de Manganese de Bahia, which has a nominal capital of 400 contos of reis ($125,000), of which amount one-half is paid up. Of the two mines, one only—Pedras Pretas—has been worked hitherto, and that only on a small scale. During 1906 that mine was worked only during the last six months. During that period the shipment of manganese ore was 4800 tons, all of which went to the United Kingdom. The Pedras Pretas mine is estimated to contain 100,000 tons of ore and the Sapé mine 250,000 tons of ore. Analysis of the ores from the two mines yielded the following percentages: Manganese, 47; silica, 7.2; phosphorus, 0.038; moisture, 1.65.

India.—The rapid advance in the price of manganese ore during 1907 and the increasing foreign demand further stimulated the industry in this country. Development has progressed to such an extent that India is likely to become the leading producer of the world in a few years' time. At present India supplies more manganese ore to the iron furnaces of Great Britain than any other country. The manganese exported from Indian to Continental ports is steadily increasing, and the quantities shipped to Belgian and German ports exceed those sent to England. The great increase in the output is shown in the accompanying table.

PRODUCTION OF MANGANESE ORE IN INDIA.

District.	1903		1904		1905		1906	
	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.	Long Tons.	Metric Tons.
Bombay	....	....	*Nil.*	*Nil.*	....	....	7,517	7,638
Central India. Uhabua State	6,800	6,909	11,564	11,749	30,251	30,736	50,074	50,877
Central Provinces Bálaghát district	7,898	8,024	10,323	10,489	} 159,950	162,517	320,759	325,907
Bhandára district	....	....	8,558	8,695				
Nagpur district	93,656	95,159	66,153	67,214				
Madras. Vizagapatam district	63,452	64,470	53,699	54,260	63,695	64,717	117,380	119,264
Total	171,806	174,562	150,297	152,707	253,896	257,970	495,730	503,686

The mining of manganese ore in India is a comparatively new industry, having been commenced about 15 years ago. Operations were first

carried on in Vizianagram close to the Bay of Bengal, in the north of the Madras Presidency, but the greater portion of the output is now obtained from an extensive series of deposits in the Central Provinces and Central India, especially in the Nagpur, Bhandara, Balaghat, Chindwara, and Jabalpur districts, as well as the Gwalior, Khairagarh and Kalahandi States.

In most of these localities the ore occurs as lenticular masses and bands in quartzites, schists, and gneisses, and is supposed to be formed by the decomposition of manganiferous garnets. The orebodies, which are sometimes several miles in length, consist of a mixture of braunite and psilomelane. The ore exported contains over 50 per cent. of metallic manganese. The cost of transport is high and renders the working of any but the highest grade ore almost impossible; the charges to Bombay (about 500 miles) being 9s. 6d. per ton, and to Calcutta (700 miles) 13s. per ton.

In the summer of 1905, promising deposits were uncovered in Sandur and Mysore, and development up to the present has revealed immense reserves of ore. On account of their proximity to the sea these deposits may take an important part in the world's production, and Marmagoa harbor is likely to surpass other Indian ports in future manganese exports. But the present capacity of that harbor is extremely limited and unless this defect is promptly remedied the south India manganese industry will receive a serious check.

The Mysore deposits, as well as those of Vizianagram and Nagpur, were described in Vol. XV of THE MINERAL INDUSTRY.

The Indian ores have several important advantages over Russian ores, their most important competitor, which give them preference in smelting establishments. The Indian ores are noted for their compact structure, hardness, and friability. The comparative absence of dust puts them at a considerable advantage over Russian ores, which are soft. Consequently, Russian ores when delivered at the smelters are partially reduced to a powdered state, resulting in loss in transit and further loss during metallurgical treatment.

At present there are no facilities in India for smelting ores on the spot, as the deposits are far removed from any sort of fuel. Consequently, a considerable amount of ore is wasted every year as it is too low grade to stand transportation. However, the possibility of transporting the ore to the neighborhood of coalfields or of importing fuel to the manganese deposits is appreciated and if this possibility is realized, a much larger amount of ore can be utilized and transportation charges on the finished product will be greatly reduced.

Russia.—Nearly one-half of the world's output of manganese ore is obtained from Russia; the chief producing areas are the Caucasus, South Russia, and to a less degree the Urals. In these districts there are ample

reserves of ore, sufficient to enable Russia to maintain for a long time to come an important position as a producer of high-grade ore. The development of the industry is hampered by the lack of transport facilities, which prevents the opening of new deposits, and also by the high rail charges to the port of shipment. The methods of mining employed are primitive, and as there are no professional miners, occasional labor only is employed. There is also a lack of organization among the producers.

The most important deposits of the Caucasus are situated in the vicinity of Tschiatura. The mode of occurrence and method of mining are typical of the other Russian deposits. The manganese extends over an area of 30,000 acres, in a bed about 6 to 7 ft. thick, lying almost horizontally and consisting of pyrolusite and other oxides of manganese, together with a certain amount of sandy calcareous matter. The ore is not submitted to any mechanical cleaning process, but is hand-picked and classified as "very rich," "rich" and "medium," and by this process yields about 33 per cent. suitable for shipment. The ore has to be transported 1 to 6 km. in mine trucks, 40 km. by narrow-gage railway, and then reloaded into broad-gage cars and conveyed 131 km. to Poti, where it is shipped.

The principal deposits of South Russia are situated in the Nikopol district in the government of Ekaterinoslav. The ore consists of pyrolusite in lumps, and as exported averages 46 per cent. manganese, 12 per cent. silica, 0.25 per cent. phosphorus, and 1 per cent. iron. The crude ore is screened and separated into two classes, according to size, (*a*) above 0.5 in. in diameter, (*b*) below 0.5 in. in diameter. The adhering clay or foreign matter in class *a* is removed with hammers, and the residue, which amounts to about 12 per cent. of the crude ore and contains 48 per cent. manganese, is known as "large ore." Class *b* is screened and yields two classes, (*c*) above 0.16 in. in diameter, (*d*) below 0.16 in. in diameter. Class *c* is hand-picked; it averages about 12 per cent. of the crude ore and contains 41-42 per cent. manganese. Class *d*, amounting to 75 per cent. of the crude ore, carrying 50 per cent. of the total manganese, is waste. Processes for dressing the ore other than that described above are little used, as the principal consumers, the blast-furnace owners, prefer a high-grade lump ore to a small-sized, dressed product.[1]

Deposits of manganese were recently found on the Trans-Siberian Railway, in the neighborhood of Samtredi, but not much is yet known about them. Also another manganese deposit has been discovered in the eastern Caucasus, in the district of Yelisvetpol, near the village of Michailovska; but as yet there are no data concerning the amount and the quality of the ore, although according to all reports the deposit seems to be important.[2]

---

[1] *Chem. Trade Journ.*, Jan. 11, 1908.
[2] *Eng. and Min. Journ.*, Dec. 21, 1907.

MICA.

Notwithstanding the increased activity in mica mining, brought about by the many new uses for both sheet and scrap mica, the production in the United States during 1907 was by far too small to supply the domestic demand. As in former years, large quantities were imported, chiefly from Canada and India. The accompanying table shows the production and imports of mica for the last 10 years. These figures are quite unsatisfactory for the reason that the mica mining industry is largely in the hands of numerous small producers, some of whom furnish unreliable reports of production, while others will make no report whatever.

STATISTICS OF MICA IN THE UNITED STATES.
(In pounds and tons of 2000 lb.)

Year.	Production. (*a*)			Imports.			
	Sheet. (*b*)	Scrap.		Unmanufactured.		Cut or Trimmed.	
	Pounds.	Tons.	Value.	Pounds.	Value.	Pounds.	Value.
1897	118,852	2,882	$28,820	722,939	$161,334	226,771	$41,068
1898	110,918	3,529	39,837	877,930	115,930	78,567	34,152
1899	97,586	6,917	50,596	1,709,839	233,446	67,293	42,538
1900	127,241	5,417	42,889	1,892,000	290,872	64,391	28,688
1901	360,600	2,171	19,719	1,598,722	299,065	78,843	35,989
1902	373,266	1,400	35,006	2,149,557	419,362	102,299	46,970
1903	619,600	1,659	25,040	1,355,375	288,783	67,680	29,186
1904	668,358	1,096	10,854	1,085,343	241,051	61,986	22,663
1905	851,800	856	15,255	1,506,382	352,475	88,188	51,281
1906	1,423,100	1,489	22,742	2,984,719	983,981	82,019	58,627
1907	1,060,182	3,025	......	2,227,460	838,098	112,230	77,161

(*a*) Statistics for 1901 to 1907 inclusive are those of the Geological Survey. (*b*) The value of sheet mica being so widely variable, and so little indicative of commercial results, and all previous statistics being of doubtful accuracy, they have been omitted from this table.

Mineralogically, micas are aluminous silicates. The more common varieties are biotite (ferro-magnesian), phlogopite (magnesian mica), muscovite (potash mica) and lepidolite (lithia mica). The last is important only as a source of lithia for the chemical trade. Biotite, because of its dark color, iron content, and occurrence in small sheets, has never been extensively used. The two commercially important micas are muscovite, the white or water mica of the trade, and phlogopite, loosely termed amber mica. Muscovite is usually of a silver-gray or light-yellow color and is transparent and often almost colorless; exceptionally, however, as in some of the Indian localities, dark red muscovite is found. Phlogopite is of varying shades of brown.

Prices and Market Conditions.—The prices paid for mica in 1907 varied greatly, depending on the quantity and quality of the product. Sales of sheet mica were made in the South and West at prices varying from 15c. to 40c. per ℔., while scrap brought from $8 to $15 per ton. It is next to impossible to generalize respecting the value of sheet mica, inasmuch as it depends upon (*a*) the quality of the mineral, especially its transparency, color, freedom from flaws and streaks, and for electrical purposes, also its freedom from iron; and (*b*) the size of the sheets into which it can be cut. The actual market value of any new product can be determined only by submitting samples to dealers in the mineral. Well-known dealers in New York are Eugene Munsell & Co., of 68 Church street, and A. O. Schoonmaker, 221 Fulton street. An approximate idea of the value of good sheet mica in 1907 may be obtained from the report of J. Obalski, which shows the production of thumb-trimmed mica given in the accompanying table together with its value at the mines, while we have added the columns of square inches and average value per pound.

VALUE OF MICA IN QUEBEC.

Size, Inches.	Square Inches.	Pounds Produced.	Total Value.	Average per ℔. (*a*)
1x2	2	204,276	$30,633	$0.15
1x3	3	139,240	34,891	0.25
2x3	6	86,003	44,460	0.50
2x4	8	71,852	49,235	0.70
3x5	15	24,248	20,090	0.80
4x6	24	12,597	13,083	1.04
5x8	40	4,074	5,347	1.30
		542,290	$197,739	

(*a*) Approximate.

This table shows that up to sheets of 8 sq.in. the value increases a little more than in direct proportion to the size, but for the larger sizes it diminishes proportionately. Obalski's statistics are also interesting in showing the proportion of commercial sizes obtainable; upward of 92 per cent. is of 8 sq.in. or less. The average value of the whole production was a little less than 37c. per ℔. Besides the sheet mica there were produced 7957 ℔. ($2109) of split mica and 91.5 tons ($13,660) of crude mica, giving a total value of $213,508, the total cost for labor being $100,000.

Baker & Startin, of London, report that the demand for mica was well maintained throughout 1907, the London warehouse deliveries having exhibited a very satisfactory increase. Stocks in hand at the close of the year were normal, and dealers had only sufficient in hand to meet their immediate requirements. Prices were well upheld, with the exception of a decline at the close of the year, chiefly in Calcutta mica, on account of a falling off in the American demand. The steady advance

of electricity in all its branches insures a continued increase in the demand for mica, and as at present no fresh sources of supply seem available, there is but little probability of any permanent decline in values.

The prices for Calcutta mica steadily advanced, owing to the rather smaller shipments, and also the keen demand prevalent during the spring and summer from the United States. The year closed, however, with a temporary decline in prices of about 20 per cent., owing to the financial setback.

The prices of Madras mica ruled very steady throughout 1907, in spite of slightly increased shipments. Supplies of all clear grades were very limited, and met with keen demand.

Splittings showed a decided decline in value, owing to the high prices ruling in 1906 having caused some over-production. The Tiger mark maintains its superiority in regularity of quality over other marks.

South America and Canada contributed but small quantities to the London market, although there are indications of some improvement in South America in 1908, because of the greater stability of sterling exchange.

The following figures for cases averaging 112 lb. each are compiled from the returns of the London warehouses: 1906: arrivals, 19,075; deliveries, 15,680; stock on hand, Dec. 31, 6798. 1907: arrivals, 19,925; deliveries, 18,908; stock on hand, Dec. 31, 7820.

Mica Mining in the United States.

Alabama (By Eugene A. Smith).—In a belt of mica schists extending from Chilton county, through Coosa, Clay, Randolph and Cleburne counties, there are numerous veins of coarse-grained granite, or pegmatite, in which the constituent minerals, quartz, feldspar and mica, are segregated in large masses. The feldspar is generally weathered into kaolin; the mica is present in the form of large rough masses, of boulders, from which it may be split in sheets of varying size. In all this belt there are ancient mines, or pits, in which trees up to 18 in. in diameter are now growing.

More or less prospecting work has been done along this whole stretch, and some little merchantable mica has been produced at several points, especially in Chilton and Clay counties. The greatest amount of work has probably been done in Cleburne county, near the lines of Clay and Randolph counties, where the Great Southern Mica Company is now operating several mines, some of which may be across the lines in the two other counties named. The mica is brought to the company's plant at Heflin and there prepared for the market. The product consists of ground mica of four different sizes, and cut mica for both stove and electrical

uses. After finishing some repairs, now in progress both at the mines and at Heflin, the output of this company will be materially increased. Toward the end of the year, the Interstate Mica Company, of Opelika, was organized to operate properties in Alabama and Georgia.

Colorado.—Several companies are operating in a small way in the granite area north and east of Cañon City, and in other localities in the Front range where pegmatite dikes are found. The United States Mica Company, of Chicago, erected a mill at Micanite, in Fremont county, and produced sheet and scrap mica during 1907. The Federal Mining Company, of Duluth, Minn., developed a property in the Bare Hills district, Fremont county. The Cañon City Mica Mills Company, of Cañon City, ground mica during two months of 1907, and is now operating steadily. The Western Elaterite Roofing Company is developing a mica mine back of Morrison, near Denver, and is using the product in the finishing of a special roofing material which has an extensive sale.

Michigan.—A promising deposit of muscovite mica was discovered near Bessemer, on the Gogebic iron range, 10 years ago. Some books cleaving into sheets 3 x 4 in. were secured near the surface, and the mica was of good quality. No serious effort has, however, been made to develop the find.

Nevada.—The mica mine of the late Daniel Bonelli, near Rioville, Lincoln county, was sold to the American Mica Company, of Syracuse, N. Y.

North Carolina.—Active mica mining has been carried on in North Carolina for the last 38 years, though with varying degrees of energy and success. It is claimed that the earliest mica mining in the State was done in Jackson county in 1867, by a Mr. Person, of Philadelphia. After a number of years of depression, due to low prices, the production is again increasing. The principal activity during 1907 was in the western part of the State. The Franklin Kaolin and Mica Company made preparations for the erection of a works at Franklin, Macon county. L. E. Timmons, of Marion, secured an option on mica privileges near Pilot mountain, Stokes county. In Mitchell county, the greatest activity was displayed in the neighborhood of Plumtree and Spruce Pine.

(By Joseph H. Pratt.) There still continues to be a large demand for North Carolina mica, and during 1907 there was a slight increase in the production over that of the previous year. The production for 1907 amounted to 645,221 lb. of sheet mica valued at $209,956. Besides this, there were 1371 tons of scrap mica produced, valued at $15,250. As can be judged from the ratio of the price to the pounds produced, a large proportion of the North Carolina mica was of comparatively large sheets, some of which were valued at $3 or more per pound. This production was obtained from nine counties in the western part of the State.

South Dakota.—Mica mining in Custer county is becoming an important

industry. The largest operating company is the Westinghouse Electric Company, which is working, among other mines, the New York and the White Spar, where 35 men are employed. The New York mine is equipped with electrical power. One of the busiest camps in the district is the Saginaw, where the company is driving from the 400-ft. level and at the same time erecting a mill of 100 tons capacity. It has one vein 12 ft. wide and two others, one on each side, each about 7 ft. wide. The company has recently purchased the Philis and Solon groups of claims, which add 160 acres to its holdings of 21 claims in the district. On the Catawassa group, situated about nine miles northwest from Custer and near the Saginaw, Dan McGonigal and J. Hazlett are opening up a promising prospect. The American Mica Company acquired the St. Louis mica mine, near Keystone, and started operations. This mine is situated seven miles east of Custer and was formerly known as the Russell mine.

Virginia.—The Henry Mica Company was incorporated late in 1907 to mine and grind mica near Martinsville. The Roanoke Mica Company, of Roanoke, was also organized about this time. At the Cooper mines, in Franklin county, only development work was done during 1907. It is reported that a large amount of mica of a good quality is developed and that operations will be resumed in the Spring of 1908.

Wisconsin.—A new variety of mica, called irvingite, was discovered in the pegmatite veins in the vicinty of Wausau. The crystals vary in size from the fraction of an inch to over an inch in diameter, and have well developed basal cleavage and prominent prismatic partings. The mineral is extremely tough and elastic and fuses easily. Analysis shows the mica to contain a considerable amount of lithia and chlorine, and a relatively large amount of silica and soda.[1]

Mica in Foreign Countries.

Canada.—This country is one of the chief sources of supply of sheet mica, ranking next to the United States as a producer. Quebec and Ontario furnish the bulk of the supply. The value of the production from 1900 to 1906 was in the neighborhood of $160,000 yearly; in 1906, however, there was a large increase, the value of the product exported being $581,919. In 1907, owing to the curtailment in mining, the value of the production was only $333,022.

British Columbia.—Mica of the muscovite variety has been discovered in commercial quantities in two localities in British Columbia, viz., at Sil-why-a-kin mountain in the Cariboo district, and on the north fork of the Thompson river at Tete Juan Cache. At the former place the mineral occurs in a dike of pegmatite. The mica is found in uniform

[1] *Am. Journ. of Science*, June, 1907.

crystals of a pure white variety. The largest blocks or crystals favor the foot wall and increase in size and purity with depth. At Tete Juan Cache, the mica also occurs in pegmatite dikes. The mica blocks, generally wedge-shaped, are set in every conceivable fashion, and therefore a great deal of care is necessary in mining the crystals, to prevent them from being shattered.

Ontario (By Fritz Cirkel).—Mica mining received a severe set-back from the financial trouble in the United States. One large electrical concern of Pittsburg, which has a mica-cutting establishment at Ottawa, discharged almost its entire working force, consisting in the busy season of from 500 to 600 persons. Another big electrical company is working with a few men only, while the other mica-cutting establishments—Blackburn, Wallingford, Comet, Munsell and others—either reduced their forces or shut down altogether. Blackburn Brothers, the most prominent mica producers in the Perkins Mills district, 15 miles northeast of Ottawa, who used to employ between 70 and 80 people at their mine and the same number at their cobbing and cleaning establishments at Ottawa, closed down mine and shop. The only mines working at present are the Wallingford, and the O'Brien-Plaunt mine, both near Perkins Mills. It is reported that the Wallingford has discovered a strong lode, yielding from 1½ to 2 tons of run-of-mine mica per week.

The O'Brien-Plaunt mine is the old Post phosphate mine, and is about two miles from Perkins Mills. About 20 men are at work, and two pits are under development, the principal one having a depth of 110 ft. Both pits are in the regular pyroxene formation and promise well for the future. Large quantities of mica were raised during 1907 from the Lacey mine at Sydenham, Ont., and also from properties near Perth, the Hanlan, the Martha and the Richardson mines; but most of these mines were shut down on account of the financial depression. At present not more than 220 persons, mostly mica cleaners and splitters, are engaged in the industry, whereas in a busy season about 1700 are employed.

Quebec.—The production of mica in 1907 was 550,247 lb., valued at $199,848. This product is further classified on a preceding page of this article. In addition there was shipped 91½ tons of crude mica having undergone a first classification, valued at $13,660, making a total value of $213,508. The mica industry in the Province employed 275 workmen, of which 150 worked on the mines and the others in the classification. The work was done during periods of four to twelve months, and a sum of $100,600 was paid in wages.

China.—There is a mica deposit in Shantung which has been investigated by German capitalists, but no mining has as yet been done. This mica is reported to be rather cloudy, but the sheets obtainable are of unusually large size and do very well for stove windows and sim-

ilar uses. The supply is said to be large, and would be a paying proposition if it were not so far removed from all means of transportation. Should the proposed railway extension be built into the district this deposit will be worked.

France.—According to Consul-General Robert P. Skinner, the imports of mica into Marseilles in 1906 amounted to 18,974 lb., of which 8382 came from India, 7255 from England, 3086 from Egypt, and 251 from Madagascar. While the trade is considerable at Marseilles, the actual transactions are arranged commercially in Paris. The prices of mica vary from 50 centimes to 40 francs (10c. to $7.72) per kg.

India.—Mica mining is confined chiefly to Bengal and Madras, although interest in the industry is growing in the Ajmere district of Rajputana. The total weight of mica mined in 1906 was 2463 metric tons, valued at $1,297,720, as against 1174 tons, valued at $710,040, in 1905. The production almost doubled during the last fiscal year. In 1905-6 the amount was 1298 metric tons, while during 1906-7 it was 2562 metric tons. Much of the small size of mica that was formerly rejected is now in demand. It is also to be noted that the deeper the mines the better the quality, the surface stains on the mica disappearing as the mining goes deeper. The accompanying table shows the production of mica in India for a series of years.

PRODUCTION OF MICA IN INDIA.
(In metric tons.)

1898........	527	1901........	1,505	1904........	828
1899........	479	1902........	806	1905........	1,174
1900........	1,025	1903........	1,002	1906........	2,463

Preparation of Mica.

The mining and preparation of mica have been elaborately described in a monograph by Fritz Cirkel, published by the Department of Mines, Ottawa, Canada. Some later notes are to be found in the following articles.

Preparation of Mica in Canada.

By Fritz Cirkel.

The mica mined principally in Canada is the so-called amber or phlogopite variety; whether this mica comes from Ontario or Quebec, it possesses great flexibility, infusibility, and softness. Sheets of large size are now less in demand, owing to the introduction of the manufactured plate mica. This mica is built up of the thin split material, pressed in powerful hydraulic presses. As a rule the Canadian mica factories receive the mica from the mines in a rough thumb-trimmed condition; at the mica

shops this mica is culled, cleaned, graded, knife-trimmed, thin-split, and finally worked into plates. For the production of say 1000 lb. of thin-split and mica plate, the following force is required: 300 girls for cleaning, grading, and thin-splitting; 200 girls for the plate building department; 50 girls for hand and machine knife trimming; 20 men; total 570 persons.

In addition, from 35 to 40 machine knives, and two powerful hydraulic presses for the manufacture of plates are required. It is claimed that the manufactured mica plates are stronger than the natural mica sheets; it is argued that the lines of molecular weakness, which are the cause of the pressure and percussion figures in the natural sheets, are eliminated through the flexibility of the single mica films, and for this reason the strength is increased. Unless this be proved by a series of experiments under equal conditions, this statement cannot be accepted, because the natural mica sheets, if properly selected, show distinctly fine and perfect lamination without any foreign substance, and it is hardly possible that this can be made artificially. There is no mineral or artificial composition which equals mica in certain physical qualities, and the more mica in a sheet is replaced by foreign material—in the case of manufactured plates by shellac, or some other cementing material—the more its original qualities must be reduced.

However, it must be said that the manufacture of mica plates has put the Canadian mica industry on an entirely different footing; many mines which could not be worked with a profit 10 years ago owing to the impossibility of disposing of the small-size mica, are now in a favorable position in the market which demands sizes from 1x1-in. up.

Canadian mica has been introduced into the markets of Great Britain with some success; in the European market, however, it has a strong competitor in Indian mica. Trial shipments made to France and Germany have not satisfied the dealers, partly on account of the mode of preparation and partly on account of terms of payment for the consignments. At present the bulk of the Canadian amber mica is sold to the United States.

Preparation of Mica in the United States.

By R. F. Fitts.

There are two distinct classes of mica dealt with in factories, viz., fishbone or scrap mica, and sheet mica. Fishbone mica derives its name from the form of a rib running through the piece, having the appearance of the bone of a fish. This develops during the formation of the crystal. It is also called "A" mica, being thicker on one edge than the other. Sheet mica is found in blocks called books, and there seems to be no limit to the thinness to which a film of this mica may be split.

The mica enters the factory over a two-mesh screen which removes all dirt and pieces too small to be handled. Screenings are sold for "poul-

try grit" and roofing material. Labor on the mica passing over this screen is performed by boys, girls or women, and consists of separating the sheet from the scrap and removing the adhering rock from both. The scrap is run down a chute to a bin ready for the grinder or pulverizer.

Sheet Mica.—Sheet mica is separated into different classes according to its different uses. The tools used in this department consist of a thin bladed knife (a thin blade of hard wood) for splitting the sheets without scratching them, a common riveting hammer to break off the rock, and a rasp fastened to the bench on which the edge of the book is rubbed; this spreads the films so that the wood riving knife may be used.

Mica washers and discs are made by a power punching press fitted with a compound die, cutting outside and center hole at one operation. The washers are thrown out by a spring or rubber pad compressed at the time of punching. Discs are punched through the female die into a suitable receptacle, fastened to the bottom of the press. Washers and discs vary in size from $\frac{3}{8}$ to 2 in. in diameter, and the center holes in the washers from $\frac{1}{4}$ to 1 in. These are used in electric light sockets, spark-plugs, and for insulators on motors and switchboards. Spark-plugs for gasoline engines are made by pressing about 2 in. of small washers tightly together and smoothing down the edges by means of a lathe. Many washers are "built up" with a composition of shellac to any desired thickness; these make the most serviceable washer, as they are not inclined to split. The best pieces of sheet mica are used for guards on rheostats and fuse boxes, and for stove and oven windows.

Mica Plate.—Thin pieces and irregular films are rived down to about $\frac{1}{200}$ in. in thickness, then placed on a steam table and painted over with a preparation of shellac; other layers are added and painted until a plate is formed of about the desired thickness, usually from $\frac{1}{16}$ to $\frac{1}{2}$ in. This plate is then submitted to hydraulic pressure varying from one to two hundred tons. The press plates are heated by steam to keep the plate warm at all times until finished. After being dry-baked in an oven, the plate is run through sand-paper rolls, or millers. The miller rolls are adjustable and grind the plates to a uniform thickness. Plates 36 in. square are made of any thickness desired. Concerns using this material send a pattern which is generally of a commutator segment. The plates are cut to this pattern by band-saws or, if small, with dies on punching presses. The material is weighed before cutting and charged to the consumer at so much per pound with labor added for cutting. Commutator rings are another form of "built up" mica. They are made in steam heated molds, pressure being applied by means of a threaded rod and burr. The sides of the rings are beveled by the tapering of the mold.

Flexible Sheets.—Mica cloth, mica paper and flexible sheets are made

by using a different adhesive, and are very pliable when warmed. Very thin sheets of mica are laid on cloth or oiled paper and treated the same as "built up" plates, only left much thinner. Rheostat, stove and fuse-block mica is cut by hand or power shears, and small sizes by a die on a power press.

Ground Mica.—The pulverizing department handles the bulk of the material of many mica mills. There are three practical ways of pulverizing mica, viz., (1) disintegration by beating, (2) abrasion with mill-stones or metal buhrs, and (3) baking at intense heat, followed by pulverizing by a blast of steam. There are several styles and makes of pulverizers of the disintegrator class, but the principle of all is the same. They consist of a metal housing with a corrugated cast-steel lining, the beaters, or hammers, being hung or fastened to the shaft which revolves at a rapid rate of speed, generally from 2000 to 3000 r.p.m. The mica enters the machine either from the side or front, and is broken up and reduced to a powder which falls through an adjustable screen at the bottom; or is drawn out by an exhaust blower running at a speed producing enough current to lift only such flakes as are of the desired mesh, or finer. These are deposited in a tank collector, which releases the air and retains the mica. Care must be taken in feeding these machines as the pieces are very irregular in size and expand greatly after entering the machine, causing it to become choked and the belt to slip.

Mill-stones are not in use at the present time, inasmuch as they require water to wash out the material, making it difficult to dry. Baking and steam blast pulverizing, being a new process, has not been tried extensively. The product treated by this method has a silvery hue instead of a white or cream color, as in the other processes, and is very much softer.

Sizing.—The mica leaves the pulverizer in a very mixed state, from eight mesh to the finest powder, and the separation of this into more uniform sizes or "grades" is the next step. The grades take their name from the size of screen through which the mica passes, namely, 8-10-24-40-60-80-120-160- and 200-mesh. To obtain these different grades, numerous screening and bolting processes are used. The most practicable of these are the vibrating screens. Frames containing the different meshes are placed one above the other, with belt apron conveyers between, and set at an angle of from 15 to 30 deg. The product is distributed over the top of the first screen by means of a spiral conveyer. That which passes over the first or coarsest screen returns to the pulverizer to be further reduced, the throughs falling to the apron conveyer, by which they are carried to the top of the next screen. The second screen takes out all finer grades and allows the first or coarsest grade to pass from the bottom into a spout or hopper. The finer grades continue on until the finest screen is reached, each screen taking out the next coarser

grade. The screens are vibrated by means of an eccentric, with rod connections and spring bumpers.

A shaking box containing all the screens is another method of separation now in use. The screens are made on frames and laid in the box with blank frames between. The box is suspended horizontally by four spring rods at each corner. A short projecting shaft fits into a socket on a horizontal eccentric wheel mounted on a pedestal fastened to the floor. This wheel is driven by a belt and gives the box about a 3-in. swing. Mica travels over the screen with the motion of the machine, going down one side and back the other. About 4 oz. of locust seeds are used on each screen. These not only keep the meshes from becoming clogged, but give the mica more weight and make it travel further. The mica passes through a coarse screen at the end and the seeds continue around again. The end of the box is arranged with blank and open spaces, allowing the grade separated to pass into a spout and the finer to fall on the next screen. Air and gravity separations have been tried without success.

Pulverized mica is packed in sacks or barrels. The packing is done by hand, as the finer grades require smaller sacks than the coarse, and no standard size may be depended upon to hold different kinds of mica.

Uses of Ground Mica.—Pulverized mica is used for many purposes. For imparting to wall paper the gold and silver luster is one of the oldest and greatest uses. Of late years it has been used on patent roofing for a double purpose; to prevent the surface from sticking when in the roll and as a protective coating. The coarsest sizes are used for this purpose. Mica being a non-conductor of electricity, is now used with hard rubber for telephone receivers and the like. Used in this way, it makes a better product and saves the manufacturer the price of the rubber for every pound used. As a lubricant, pulverized mica cannot be surpassed. Being frictionless and even, the finest grades having great capillary attraction, it holds the oil or grease to the bearing; the small flakes adhere to both parts of the bearing and receive most of the wear. Mixed with a heavy oil or grease it will preserve the rubber packing of a boiler manhole and the same piece may be used over and over again.

Experiments with mica for fireproof paints are being made, and some very good paints are produced at present, but there is room for improvement in the best of these. Mica flakes sprinkled on the face of the tile before firing add a very bright appearance and make a fancy finish. There are many minor uses for this material, but the manufacturer looks to the uses above mentioned for his chief market.

MINERAL WOOL.

The mineral wool industry of the United States in 1907 was practically the same as in 1906. No new producers entered the field and no changes or innovations in manufacture were recorded. In 1907 the production was 9008 short tons valued at $81,769 as compared with 5375 tons worth $55,550 in 1906. The accompanying table gives statistics for 10 years except for 1903 and 1904 when no statistics were collected.

PRODUCTION OF MINERAL WOOL IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Amount.	Value.	Per Ton.	Year.	Amount.	Value.	Per Ton.
1898	6,560	$70,314	$10.72	1903	(a)	....	....
1899	7,448	85,899	11.53	1904	(a)	....	....
1900	6,002	60,320	10.05	1905	6,164	$69.560	$11.28
1901	6,272	68,992	11.00	1906	5,375	55,550	10.33
1902	10,843	105,814	9.67	1907	9,008	81,769	9.08

(a) No statistics collected.

There were but six companies making mineral wool in 1907. These were: United States Mineral Wool Company, 140 Cedar street, New York; Columbia Mineral Wool Company, 112 Clark street, Chicago, Ill.; Pennsylvania Mineral Wool Company, Norristown, Penn.; Union Fibre Company, Yorktown, Ind.; Banner Rock Wool Company and American Insulating Material Manufacturing Company, both at Alexandria, Ind. The first three make their product from furnace slag and the others from sandstone or other silicious rock. In the West the Union Fibre Company is the largest producer with the Columbia company second. In the East the United States company leads in production.

About 70 per cent. of the wool produced from furnace slag is used to insulate the floors and walls of buildings, its non-conducting properties making it valuable for this purpose. The remainder is used for covering water pipes, and for use in refrigerators. A possible new field is in the manufacture of "fireless cookers." The principal use for rock wool is in the form of compressed blocks used in cold storage plants. About 65 to 75 per cent. of the production is thus utilized. The remainder is used in building construction. The use of mineral wool for filters has been abandoned.

MOLYBDENUM.

By Reginald Meeks.

Molybdenum has much the same effect as tungsten, when alloyed with steel, but, it is claimed, the amount required to produce the same result is only one-half to one-third that of tungsten. However, the unreliability of the supply makes molybdenum of but little economic importance. Molybdenum steel is used for rifle barrels, for large guns, for propeller shafts, for wire and for special "high speed" steels. Molybdenum, in the form of ammonium molybdate, is used to determine phosphorus in iron, several tons per annum being used for this purpose in the United States alone. In Europe ammonium molybdate is employed as a fireproofing material and it is claimed that it is a powerful germicide. In the manufacture of pottery, molybdenum salts give a fine blue color.

Price.—Molybdenum ore, guaranteed 90 to 95 per cent. MoO_3, is quoted at $400@450 per ton. The price is variable and usually is a matter of negotiation between buyer and seller. The Primos Chemical Company, of Primos, Delaware county, Penn., and De Golia & Atkins of San Francisco, Cal., are the principal dealers. Fried. Krupp, Essen, Germany, is a large user of molybdenum.

Molybdenum in the United States.

Arizona (By Wm. P. Blake).—Molybdenite is found in limited quantity in many widely separated localities in the Territory. Its close association with decomposing galenite leads to the belief that it is present in that mineral, probably diffused in thin foliae along the crystalline planes, but invisible by reason of the similarity of color and luster of the two species. But when oxidized to molybdic acid, and in combination with oxide of lead, it assumes the yellow color and brilliant tabular crystallization of wulfenite and becomes strikingly evident in the oxidized ores. In this form it has been collected commercially and shipments of a few carloads were made a few years ago from the accumulated tailings of the Mammoth gold mine at Shultz, in Pinal county. In portions of this mine the mineral was very abundant, sometimes in massive aggregations of crystals weighing many pounds. It was not always carefully separated from the ore by hand-sorting and consequently was crushed with the ore in the stamp mill at Mam-

moth, on the San Pedro river, and became distributed through the tailings. From these accumulated tailings the wulfenite was secured by simple sluicing or other methods of concentration. The yield was said to be from 1 to 2 per cent. There was probably a small amount of vanadinite present as this mineral, and also a vanadate of zinc and lead (descloizite) occurs in the ore. The production of these concentrates has ceased. Wulfenite also occurs in considerable quantity, associated with vanadinite, at the old Yuma mine, about 20 miles northwest of Tucson; at the lead mines of Castle Dome; and in the silver districts on the Colorado river near Yuma.

Molybdenite occurs usually associated with iron and copper sulphides, generally in films or thin coatings, and is not available commercially. However, in the quartz vein on McClary's claim in the Santa Rita mountains, it is found in heavier, thicker masses, quite clean and pure. This locality can be relied upon for a commercial supply of ore suited to concentration by the oil or grease process which makes a clean separation of quartz and molybdenite. Very little exploratory work has been done on this vein and there have been no shipments. Molybdic ocher is abundant in the croppings of the McClary vein.

California.—The ore found in this State[1] is molybdenite and was discovered in 1906 in a granite quarry about 4.5 miles northeast of Corona, Riverside county. The granite is cut by pegmatite dikes 0.5 to 2 in. wide and molybdenite in flakes up to 0.5 in. across accompanies the dikes in small quantities.

Idaho.—A discovery of molybdenite was reported on the properties of the Sanca Consolidated Company at Knob Hill, Kootenai county.

Maine.—Molybdenite occurs widely distributed; the principal deposits are in Washington and Hancock counties and several companies have been organized to exploit them.

Oregon.—Late in 1907 deposits of molybdenite were discovered near Galice, Josephine county. On the property of J. E. Cross considerable development was done but no ore was produced during the year. The vein was cut while driving a tunnel to intercept a vein of copper ore. It has been found difficult to concentrate the ore and experiments are being carried on to determine the proper process.

Utah.—The deposits, occurring at Alta, consist of wulfenite and have been known for years. Heretofore the ore has been sold for its lead content only, but now its value as a source of molybdenum is recognized.

Molybdenum in Foreign Countries.

Australia.—Molybdenite occurs in Victoria, New South Wales and Queensland, the last State furnishing the bulk of the supply. In New

[1] Frank L. Hess, Bulletin No. 340, U. S. Geological Survey.

South Wales and Queensland the deposits are found at or near the junction of the granite and quartz in which they are imbedded. The New South Wales deposits are in the northern part of the State near Tenterfield, Bolivia, Deepwater and east of Glen Innes. Most of the production is from the Sach mine at Kingsgate, near Glen Innes, where the ore occurs in "pipes" or long, bent, cylindrical masses of quartz containing molybdenite and bismuth. The "pipe" varies in width from 8 to 20 ft. and its vertical direction is tortuous. Specimens weighing nearly 700 lb. have been obtained from this mine. After hand-picking the ore is crushed through rolls and is then jigged. The oversize is recrushed, hand-picked again and then treated on Wilfley tables.

In northern Queensland molybdenite is found in the Hodgkinson goldfield associated with wolframite. The gangue is clean white quartz, often clear and glassy, occupying the joints and contraction fractures of a gray biotite granite. Metallic bismuth is also found and recovered. The wolframite and molybdenite are readily separated from the quartz by hand dressing. The ore is dressed in much the same way as at Kingsgate; there is considerable waste of flaky material.

South Africa.—A discovery of molybdenite was reported by F. W. Glennie in April, 1907, to the Discoverer's Rights Commission at Pietersburg, Transvaal.[1] The deposit consists of molybdenum and tin in combination with arsenic and sulphur. It is situated about 27 miles northwest of Potgieters Rust in the Waterberg district.

---

[1] *South African Mines, Commerce and Industries*, May 4, 1907.

MONAZITE.

The chief occurrences of monazite in the United States are in North and South Carolina, and up to the present time this may be said to be the only locality in which the industry has become of commercial importance. Monazite also occurs in Georgia and sparingly in many other localities in the United States; the sands of the Pacific slope, especially those in Oregon and Idaho, which have been investigated by the U. S. Geological Survey, show the presence of some monazite and considerable quantities of zircon. The production of monazite during 1907 showed a decided decrease, due chiefly to the fall in the price of thorium nitrate, the depressed financial conditions toward the close of the year, the exhaustion of some of the richer deposits of Brazil, and the trouble which the miners in that country experienced over tax matters. The imports of thorium nitrate into the United States since 1903 have been as follows, the figures representing the year, the number of pounds, and the value in the order given: 1903, 72,990, $244,258; 1904, 71,595, $261,232; 1905, 38,274, $200,238; 1906, 57,892, $254,858; 1907, 88,653, $240,128.

Monazite occurs in paying quantities in Brazil, Norway, Australia, Liberia, and in Tringganu (an independent Malay State); it has also been found in Ceylon and in the Transvaal, but the value of these deposits

MONAZITE PRODUCTION IN THE UNITED STATES.
(In pounds.)

Year.	United States. (a)			North Carolina. (b)		
	Pounds.	Value.	Per Pound.	Pounds.	Value.	Per Pound.
1897	44,000	$ 1,980	$0.045	44,000	$ 1,980	$0.045
1898	250,776	13,542	0.054	250,776	13,542	0.054
1899	350,000	20,000	0.057	350,000	20,000	0.057
1900	908,000	48,805	0.054	908,000	48,805	0.054
1901	748,736	59,262	0.079	748,736	59,262	0.079
1902	802,000	64,160	0.080	802,000	64,160	0.080
1903	862,000	64,630	0.075	773,000	58,694	0.076
1904	745,999	85,038	0.114	685,999	79,438	0.116
1905	1,352,418	163,908	0.121	894,368	107,324	0.120
1906	846,175	152,312	0.180	697,275	125,510	0.180
1907	547,948	65,754	0.120	(c)547,948	65,754	0.120

(a) Statistics of the United States are those of the U. S. Geological Survey. (b) The figures for North Carolina, from 1897 to 1906, inclusive, are from "The Mineral Industry of North Carolina." (c) The figures for 1907 were collected jointly by the U. S. Geological Survey and the N. C. Geological and Economic Survey; they include the production of South Carolina.

has not been proven as yet. During 1907 important discoveries were reported on Kangaroo island, South Australia, and in the Ural mountains in Russia. The chief supply comes from Brazil, monazite being among the important items of export from the State of Bahia. According to the *Brazilian Review*, the exports of monazite from Brazil since 1902 have been as follows, in metric tons: 1902, 1205; 1903, 3299; 1904, 4860; 1905, 4437; 1906, 4351; 1907, 4438. The proportion of thoria in the Brazilian monazite sands varies from 5 to 5.7 per cent. in the interior of the Republic, and is as high as 7 per cent. on the coast. The monazite, being magnetic, may be easily separated from other minerals.

Notes on the methods of mining, treatment and industrial conditions in the monazite industry are to be found in previous volumes of The Mineral Industry.

Monazite in the Carolinas.

By Joseph Hyde Pratt.

One of the most important events in connection with the monazite industry during 1907 was the extension of the monazite area into Alexander county, North Carolina, where commercial deposits of this mineral were found and purchased by the National Light and Thorium Company. It is interesting to note the extension of the monazite area since this mineral was first mined in the Carolinas in 1893. At that time the monazite field was given something like an area of 2000 square miles in Burke, Catawba, Cleveland, Gaston, Lincoln, McDowell, Polk, and Rutherford counties in North Carolina, and extending into Spartanburg and Greenville counties, South Carolina. Now the area is known to cover approximately 3500 square miles including, besides the above-named counties, portions or all of Alexander, Caldwell and Iredell counties in North Carolina, and Anderson, Cherokee, Laurens, Oconee and Pickens counties in South Carolina. The discovery of the Alexander county monazite deposits was the direct result of prospecting by the National Light and Thorium Company, of Youngstown, Ohio.

The commercial deposits of monazite are the gravel beds in streams and bottom lands and, in certain places, the surface soils which adjoin the original gravel deposits. Mill and laboratory tests have been made upon the monazite-bearing rock, but thus far no deposit has been discovered that could be worked profitably. Occasionally certain portions of the saprolitic rock in close proximity to the gravel beds, especially that immediately underlying the gravel beds, have been scraped up and washed, giving a small margin of profit. The greater percentage of the monazite in these alluvial deposits is usually, as in all placer deposits, near bed rock, but monazite occurs throughout the top soil or overburden as well as the gravel, so that it usually pays to wash all of this material.

Monazite occurs for the most part in the pegmatized gneiss and schist of the country rock of this area, which is known as Carolina gneiss. Those portions which have been highly pegmatized are rich in secondary quartz and contain numerous small masses of feldspar, with some graphite, biotite and other accessory minerals. The monazite is nearly always well crystallized, although the crystals are extremely small. The percentage of this mineral in the rock varies considerably and will not average over 0.5 per cent. Attempts that have been made to work the rock have all failed on account of the low grade of the material, although a clean monazite product could be obtained by concentration.

During 1907 the total production[1] of monazite in North and South Carolina was 547,948 ℔. of the refined or concentrated sand, which would average approximately 90 per cent. of monazite. This is valued at 12c. per ℔., giving a total value of $65,754.

---

[1] The figures given regarding production were collected co-operatively by the U. S. Geological Survey and the North Carolina Geological and Economic Survey.

NATURAL GAS.

Natural gas is a highly important industrial fuel which is produced in 19 States, but is of great consequence only in Pennsylvania, West Virginia, Ohio, Kansas, Oklahoma, and Indiana. These States rank in importance in the order mentioned. According to the U. S. Geological Survey, the value of the production in 1906 was about $47,000,000 against about $42,000,000 in 1905. However, these figures must be general approximations only, inasmuch as natural gas is an extremely difficult substance to account for statistically. The Geological Survey stated last year that for the first time in the history of its statistics a successful effort was made in 1906 to collect reports of the quantity of gas produced. It reported that in 1906 the product amounted to 388,842,562,000 cu.ft., measured at the atmospheric pressure, or 9,396,964 short tons. On the basis of weight the average price per short ton of natural gas was $5, against an average of $1.11 per short ton for bituminous coal. The difference in the value between natural gas and bituminous coal is, of course, due to the superior fuel efficiency of natural gas, weight for weight, and the greater economy of labor in its use. There is nothing that has to be shoveled into the furnace and no ashes to be removed.

In the eastern States, natural gas is now largely employed as a domestic fuel and for special industrial purposes, wherefore its value is high, the averages for 1906 having been 13.4c. per thousand cubic feet in Pennsylvania, 11.5c. in West Virginia, 15.7c. in Ohio and 22.2c. in Indiana. In Kansas natural gas is still employed more largely as a commercial fuel, the average for the whole State in 1906 having been only 5.8c., while in the zinc smelting district the average value was only 1.8c. per thousand cubic feet, the price ranging from 1 to 3c. per thousand. In some parts of West Virginia gas is still obtainable at as low a price as 2c. per thousand cubic feet at the wells. Besides its use as a fuel, large quantities of natural gas are consumed, especially in West Virginia, for the manufacture of carbon black.

Natural Gas in Kansas and Oklahoma.

By Erasmus Haworth.

Kansas.—Every year it becomes more difficult to estimate the value

of gas produced in Kansas. At the end of 1907 pipe-line companies were paying as high as 3c. per 1000 cu.ft. for gas delivered to them directly from the wells; a number of private producers disposed of their gas in this way. Other producers had contracts at a less price. Kansas now has more than 125 towns and cities using natural gas; a great majority of these use meters and pay at the rate of 25c. per 1000 cu.ft., some even 30c. The Kansas Natural Gas Company is the largest retail dealer, but by no means the only one. The pipe line of this company goes as far north as St. Joseph, Mo., taking in St. Joseph, Mo., Atchison, Leavenworth, Lawrence, and Topeka, and also the smaller cities and towns between. A pipe line also reaches Kansas City, which has an aggregate population of nearly 400,000, about two-thirds of whom are now using natural gas. Another pipe line belonging to the Kansas Natural Gas Company is carried east from the gas fields by way of Parsons, Oswego, Columbus, Pittsburg, and into the entire zinc-mining district of southeastern Kansas and southwestern Missouri. A portion of this is sold at the rate of 25c. per 1000 cu.ft., but a much larger portion, that used for generating power, is disposed of at the rate of 10c. per 1000 cu.ft. Another pipe line is carried westward to Wichita and beyond, connecting with all the intermediate towns. If we reckon this gas and that consumed by the cement plants, smelters, brick yards, and minor manufacturing plants at 3c. per 1000 cu.ft., the total production from Kansas alone for 1907 will have a value of between $5,000,000 and $7,000,000. Should it be estimated at the actual retail price, the total value would be very much greater, but here again the complexity is so great that this has not been attempted. The pipe line companies make various rates to different factories, less than 10c. and upward, depending on the size of the factory and the particular kind of a bargain that may be made. Were the entire consumption paid for at specific rates, difficulties would not be so great.

Oklahoma.—But little gas was marketed from Oklahoma except for local consumption. One small pipe line was laid across the State line near Caney and a large amount of gas was conducted through it and delivered to the pipe lines of the Kansas Natural Gas Company. The Secretary of the Interior and the State of Oklahoma are making efforts to prevent any further transportation of gas out of the State. Early in the present session of the Oklahoma legislature a bill to that effect was passed and signed by the Governor, and is now a law. Steps have already been taken to place the pipe line now crossing the State line in the hands of a receiver, hoping thereby to close it. In Oklahoma, consumption was confined almost entirely to domestic uses in the various cities and towns, for but few factories are as yet established. At Bartlesville, two zinc smelters were operated and a third was completed at the end of the year. These give to that town an extra consumption. At

Dewey, four miles north, the portland cement plant began operation at the close of 1907.

The total value of gas in Oklahoma actually consumed in 1907 was approximately $1,500,000, the estimate being based on a rate of 3c. per 1000 cu.ft. This statement, however, of itself would convey an extremely imperfect idea of the possibilities of gas production in Oklahoma. Probably no other place in the world, now or at any other time, ever had so much gas developed ready for immediate consumption as has Oklahoma. Natural gas occurs everywhere throughout all the productive oilfields. Many wells range from 15,000,000 to 30,000,000 cu.ft. per day, while a few are reported to have a capacity close to 40,000,000 cu.ft. per day. The large amount of fuel now awaiting consumption is astonishing. And this, too, in face of the fact that all the development companies, with but few exceptions, have been trying to keep away from gas in their search for oil. What the result will be in the future when an intelligent search for gas is made, no one can state at the present time, but the value of the annual production certainly will reach several million dollars.

Developments.—In Kansas, considerable search was made for gas, the difficulty of piping gas from Oklahoma having served as a strong incentive. The most remarkable individual field discovered is about six miles southwest of Chanute. Here, on the high land between the Neosho and Verdigris rivers, a field was developed almost entirely during 1907; some wells of this district are so large as to compare favorably with the best of the wells drilled in Montgomery county two years ago. Wells ranging from 2,000,000 to 30,000,000 cu.ft. per day seem to be comparatively common.

Another commercially important field, developed during 1907, lies to the east of Fredonia. The wells are not very large, ranging from 3,000,000 to 5,000,000 cu.ft., but the field is so situated that the flow is consumed by various nearby manufacturing industries. In a similar manner, considerable development was made to the northeast of Chanute, or southeast of Humboldt, near the southern line of Allen county; this field extends eastward more than half-way to Savenburg. The wells here, likewise, are comparatively small, but in the aggregate produce a large quantity of gas. Still another field of equal importance is being developed near Hale, in the northeast part of Chautauqua county. Wells from 3,000,000 to 5,000,000 cu.ft. per day are comparatively common.

An interesting small gasfield was opened up near Elmdale, in the Cottonwood river valley, near Cottonwood falls. Here is a well marked anticlinal ridge. A member of the Kansas State Geological Survey suggested to certain citizens of Elmdale that it would be a good place to prospect for gas; accordingly a number of wells were drilled only a few hundred feet deep and a flow of shallow gas was obtained; the wells vary in flow from 500,000 to 1,000,000 cu.ft. per day. A similar condition exists to

the southwest in the vicinity of Augusta, where shallow but good gas is obtained in a number of wells; the anticlinal ridge is not so fully marked there as at Elmdale. The now somewhat famous gas at Dexter is obtained from a similar anticlinal ridge, but the quality of the Dexter gas is so different from that at Elmdale and Augusta as to cause one to think that it comes from an independent pool.

The gasfield at Arkansas City continues to be very interesting; it is similar to the Augusta-Elmdale field. The gas is first-class in quality and is much greater in quantity than has as yet been developed at either of the other two places. A flow of gas has been developed sufficiently to supply Arkansas City and still leave a surplus. What future developments will bring forth, of course, is largely conjecture, but it looks as though important developments may be expected in this district near and between Elmdale and Arkansas City.

In Oklahoma, gas was found in many new wells along the ridge east of Bartlesville and north practically to the State line; a few large wells, in what appears to be the same field, were developed across the line in Kansas. South, along the Hog Shooter, good wells are also found. Here the gas lies beneath the oil and frequently, when a well drilled for oil goes dry, it is carried deeper and a good gas well is obtained. The same is true regarding the region east of Collinsville, where wells, having a capacity of from 5,000,000 to 12,000,000 cu.ft. per day, are obtained. Also a number of good gas wells have been obtained in the vicinity of Tulsa and west of Red Fork, in the district surrounding the Glenn oil pool, and also near Muskogee; the boundaries of this last field have approached closer to Muskogee than was the case a year ago.

While no distinctly new field was opened up in Oklahoma during 1907, still gas was found in so many different places that the total possible production for the year was greatly increased.

Geology of the Field.—The oil and gas produced in the Mid-Continental field, except that from the gas wells in the Elmdale-Arkansas City district and the Muskogee oilfield on the southeast, come mainly from Lower Carboniferous strata. In Oklahoma, the westward development seems to result in deeper wells, so that the productive horizon remains about the same. In Kansas, however, there is a slight modification of this and many of the oil wells and gas wells in the west part, near Longton and Howard, are not deeper than wells in the eastern part. This means that the productive horizon there is higher up geologically than in the heart of the field. Gas in the Elmdale-Arkansas City district is obtained fully 1000 ft. higher geologically than the gas about Iola, Chanute and Independence. The mouths of these wells are in the Permian, but the productive sandstones are in the uppermost part of the Carboniferous. Some oil has been obtained at Muskogee from sandstones evidently

below the Mississippian. Few attempts have been made elsewhere to penetrate the formations below the Mississippian except in the vicinity of Chelsea, Miami, and a few other points, outside of the productive fields to the east. All of these wells, except those in the immediate vicinity of Muskogee, have been barren of oil and gas.

NICKEL AND COBALT.

By Reginald Meeks.

The production of nickel and cobalt in the United States in 1907 continued to be insignificant in so far as the domestic mines are concerned, but beginning with 1908 there is strong prospect of an important production by the North American Lead Company of Fredericktown, Mo. At that place, on the southern border of the disseminated lead district of southeastern Missouri, copper-, cobalt-, and nickel-bearing minerals, which in other parts of the district are found in small quantity, have been found to occur in an extraordinary amount. The copper occurs as chalcopyrite and the nickel and cobalt as linnæite, both of these minerals being associated with galena. In some parts of the deposit the galena predominates; in other parts the copper-nickel-cobalt minerals are largely in excess. The ore of the latter class as mined assays about 6 per cent. copper, and 3 per cent. nickel and cobalt. Obviously this is a high grade of ore, considering its total of metallic contents. I am reliably informed that the deposits are large, and consequently there is much reason to believe that these mines will be an important source of nickel and cobalt.

Ore of this character occurs not only in the mine of the North American Lead Company, but in adjacent mines, although few, if any, of the latter are so rich in copper, nickel and cobalt. However, they have obtained, for a long time past, more or less of these metals as a by-product and the works that have been installed by the North American Lead Company will furnish a convenient market for them. The works of the North American Lead Company comprise mechanical roasting furnaces, a blast furnace, blister copper furnaces, and an electrolytic refinery for both copper and nickel, beside a department for the production of cobalt oxide. The present capacity of these works is about 1,700,000 ℔. of copper per annum, 720,000 ℔. of nickel, and 180,000 ℔. of cobalt oxide. Electrolytic copper was produced to a considerable amount in 1907. The production of cobalt oxide and electrolytic nickel was begun in the early part of 1908.

Besides the North American Lead Company, electrolytic nickel is produced in the United States by the Balbach Smelting and Refining Company, of Newark, N. J., which obtains its crude metal from the Orford

Copper Company. The latter smelts copper-nickel matte from Sudbury, Ontario, by the clever process of "tops and bottoms," which has been described in earlier volumes of THE MINERAL INDUSTRY. The works of the Orford Copper Company are at Constable Hook, N. J. During 1907 these works were largely rebuilt, being modernized in many particulars, especially with respect to the economical handling of material, and at the same time were greatly increased in capacity. This company being the only producer of metallic nickel in the United States in 1907, we are unable to give statistics of the production, but on the basis of the importations of nickel in matte into the United States we estimate the production of the metal in this country at 17,500,000 ℔. in 1907 against 14,300,000 ℔. in 1906.

UNITED STATES IMPORTS AND EXPORTS OF NICKEL AND COBALT.
(In pounds, and tons of 2240 ℔.)

Year.	Imports.							Exports.	
	Nickel Ore and Matte.		Nickel Alloys. (a)		Nickel Mnfrs.	Cobalt Oxide.		Nickel. (b)	
	Long Tons.	Value.	Pounds.	Value.	Value.	Pounds.	Value.	Pounds.	Value.
1897..	12,420	$781,483	(c)	....	....	24,771	$34,773	4,255,558	$997,391
1898..	26,826	1,534,262	(c)	....	....	33,731	49,245	5,657,618	1,359,609
1899..	19,857	1,216,253	(c)	....	....	46,791	68,847	5,004,377	1,151,923
1900..	25,670	1,183,884	455,188	$139,786	....	54,073	88,651	5,869,906	1,382,727
1901..	52,111	1,637,166	635,697	209,956	$2,498	71,969	134,208	5,869,655	1,521,271
1902..	14,817	1,156,372	752,630	251,149	30,128	79,984	151,115	3,228,607	925,579
1903..	15,936	1,285,935	521,344	170,670	37,284	73,350	145,264	2,414,499	703,550
1904..	8,548	915,470	589,555	203,071	2,950	42,352	86,925	7,519,206	2,130,933
1905..	13,451	1,626,920	941,966	331,920	3,291	70,048	139,377	9,550,918	2,894,700
1906..	15,156	1,816,631	210,000	77,373	8,963	41,084	83,167	10,620,410	3,493,643
1907..	(d)16,888	2,153,873	180,025	80,994	9,159	42,794	73,028	8,772,578	2,845,663

(a) Includes nickel, nickel oxide, and alloy of any kind in which nickel is the material of chief value, in ingots, bars and sheets. (b) Comprises domestic nickel, nickel oxide and matte. (c) Not separately enumerated; included in "Nickel Ore and Matte." (d) Contained 18,418,305 ℔ nickel; not reported previous to 1907.

Markets.—The markets for nickel and cobalt were stationary throughout 1907. The price for nickel was fixed late in 1906 and the quotation remained unchanged, in 1907, at 45@50c. per ℔. for large lots according to the size of the order and terms of sale. Smaller lots fetched 50@65c. per ℔.

Cobalt is marketed exclusively in the form of oxide and fetched $2.50 per ℔. throughout 1907. In March, 1908, a violent rate war occurred between the two producers and the price was cut at frequent intervals until on Apr. 1 it was $1.45 per ℔., a reduction of $1.05 in a few weeks.

NICKEL AND COBALT IN THE UNITED STATES.

Alaska.—Prospecting for nickel ore at Port Valdez and near the mouth of Miners river, on the shores of Prince William Sound, failed to

disclose nickel in the pyrrhotite disseminated through the diorite country rock.

Arizona.—According to William P. Blake, Territorial Geologist, cobalt occurs in some of the gold-bearing lodes of the Quijotoa district in Pima county, but no development of importance has yet been made.

Missouri (By E. R. Buckley).—The North American Lead Company, with mines at Fredericktown, is mining copper sulphide ore which carries with it nickel and cobalt, both of which are recovered. Nickel and cobalt are refined at this plant, which is the first in the State to produce refined nickel. Heretofore ore carrying nickel and cobalt has been shipped, but the quantity has been small. The first lot of refined nickel ever made in Missouri was shipped by the North American Lead Company about March 1, 1908, and consisted of 10,000 ℔. of metal, 98 per cent. pure.

Nevada.—The copper-nickel-platinum ores of the Bunkerville district in northeastern Lincoln county, Nevada, occur in diabase eruptives in metamorphic schists, and nearly parallel lenses, the middle one of which only has been considerably exploited. The average assay of the ores exposed is 3.5 per cent. copper, 2.5 per cent. nickel and 0.25 to 0.3 oz. platinum. The ore is, as regards appearance and metal contents, very similar to that of Sudbury, Canada.

The Nevada Nickel and Copper Company has developed the Key West group of claims with upward of 5500 ft. of laterals from three levels. The Nevada Copper, Platinum and Nickel Company has opened up the Great Eastern properties with 1200 ft. of drifts and crosscuts.

The district is accessible by a good 45-mile wagon road from Moapa station on the San Pedro, Los Angeles & Salt Lake Railroad.

Nickel and Cobalt Mining in Foreign Countries.

Canada.—With the exception of the nickel contained in the ores shipped from the Cobalt district, the production of nickel in Canada is derived entirely from the well known nickel-copper deposits of the Sudbury district. The output has been increasing steadily for a number of years, although the actual amount of nickel contained in matte shipped in 1907 was somewhat less than in 1906. Two companies are carrying on active operations: The Mond Nickel Company, at Victoria Mines, and the Canadian Copper Company, at Copper Cliff. The ore is first roasted and then smelted to a matte containing from 77 to 80 per cent. of the combined metals, copper and nickel, which is shipped to the United States and Great Britian for refining. Of the exports of nickel in matte, 2,518,338 ℔. went to Great Britain and 16,857,997 ℔. to the United States according to the Canadian customs returns for 1907. The copper produced from the nickeliferous pyrrhotite of the Sudbury district amounted to 6996 short tons in 1907 against 5205 in 1906.

PRODUCTION, EXPORTS AND IMPORTS OF NICKEL IN CANADA. (a)

Year.	Production.		Exports.		Imports.
	Pounds. (b)	Value. (c)	Pounds. (d)	Value. (e)	
1900	7,080,227	$3,327,707	13,493,239	$1,040,498	$6,988
1901	9,189,047	4,594,523	9,537,558	958,365	12,029
1902	10,693,410	5,025,903	3,883,264	834,513	15,448
1903	12,505,510	5,002,204	9,032,554	878,159	26,177
1904	10,547,883	4,219,153	14,229,973	1,237,307	16,330
1905	18,876,315	7,550,526	11,970,557	1,185,056	19,076
1906	21,490,955	8,948,834	23,959,841	2,166,936	(f)15,808
1907	21,189,793	9,535,407	19,376,335	2,280,374	....

(a) Statistics for production and imports cover calendar years, and are taken from the Annual Reports of the Geological Survey of Canada. Figures for exports cover the fiscal years ending June 30, and are taken from the Statistical Year Book. (b) Pounds metallic nickel contained in copper and nickel matte exported. (c) On the basis of refined nickel at New York, from the *Engineering and Mining Journal* average annual quotations. (d) Pounds of nickel contained in ore, matte or speiss. (e) Spot value, to the producer, of the exported material; the variety of stages at which the material is shipped, as well as the different periods of time covered, lead to the apparent discrepancy in value when it is known that practically the entire production is exported. (f) Anodes only.

ONTARIO NICKEL STATISTICS.
(In tons of 2000 lb.)

Schedule.	1901	1902	1903	1904	1905	1906	1907
Ore raised	326,945	269,538	152,940	203,388	277,766	343,814	351,916
Ore smelted	270,380	233,388	220,937	102,844	251,421	340,059	359,076
Per cent. nickel	....	2.55	3.16	4.58	(b)	(b)	(b)
Per cent. copper	....	1.78	1.81	2.41	(b)	(b)	(b)
Ordinary matte	29,588	24,691	30,416	19,123	17,388	20,364	22,041
Bessemerized	15,546	13,332	14,419	6,926			
Nickel content	4,441	5,945	6,998	4,743	9,438	10,745	10,095
Copper content	4,197	4,066	4,005	2,163	4,386	5,265	6,996
Value of nickel (a)	$1,859,970	$2,210,961	$2,499,068	$1,516,747	$4,019,814	$4,629,011	$3,289,382
Value of copper (a)	589,080	616,763	583,646	297,126			
Wages paid	1,045,889	835,050	746,147	570,901	(b)	1,117,420	1,278,694
Men employed	2,284	1,445	1,277	1,063	(b)	1,417	1,660

Note.—The quantities reported in 1901, 1902 and 1903 under "bessemerized matte" include both bessemerized matte and high-grade matte, the former being the product of the Mond Nickel Company's works and the latter of the Ontario Smelting Works, which re-treat the low-grade matte produced by the Canadian Copper Company. (a) Value based on nickel and copper in matte and not on refined metals. (b) Not available.

The figures of nickel production do not include the content of the silver-cobalt ores from Cobalt district, complete statistics of which have not been obtained. The shippers of silver-cobalt ores receive practically no return for the nickel content, although this amounted in 1906 to about 3 per cent. of the ore shipped, according to data reported by the Ontario Bureau of Mines.

The Copper Cliff smelter of the Canadian Copper Company treated Cobalt ores on terms that were regarded as reasonable by the mine owners. Payment is made in two installments of 45 and 90 days, respectively, after sampling the ore, and is based on the official value at New York on the first day of settlement. The purchaser reserves the right to pay in silver bullion delivered at New York in lieu of cash. The price is fixed on a sliding scale according to the grade of ore, the highest figure being 94 per cent. of the silver content when the assay shows 4000 oz. or more to the ton, and the lowest, 80 per cent., when the silver is less than 300 oz. Payment is also made for cobalt at the rate of $30 per ton

of ore when it contains 12 per cent. of cobalt and over; $20 per ton for 8 per cent. and $10 for 6 per cent. of cobalt; but no payment is made for a smaller amount of cobalt nor when the nickel content is higher than that of cobalt.

The Canadian Copper Company is successfully operating its new smeltery at Copper Cliff. Electric power is generated at the High falls of the Spanish river, near Turbine station on the Canadian Pacific Railroad. The Mond Nickel Company is developing a new deposit in Garson township and is contemplating the development of water power on the Vermillion river to supply current to its mines and smeltery. Extensive deposits of nickel ore have long been known to exist in the Northern range and now that railway facilities have been provided by the Hutton branch of the Canadian Northern Railway, the company, which has acquired an extensive interest in this region, is likely to inaugurate early development.

(By Thomas W. Gibson.) The once current aphorism that there are no precious metals east of the Rocky mountains has been abundantly disproved by the Cobalt silver camp. Discovered in 1903, the development has been rapid, the output of silver rising from 206,875 oz. in 1904, to 10,023,311 oz. in 1907 derived from 14,788 tons of ore. The aggregate production of the camp up to the end of 1907 was 22,425 tons of ore, containing 18,083,308 oz. silver. Up to the end of 1906 the ore shipments averaged 1055 oz. silver per ton, a very high figure, but the average content fell during 1907 to about 677 oz. per ton. This reduction does not necessarily mean that the richness of the veins at the surface is lessening in depth, although this is probably to some extent the case, but rather that low-grade ores, instead of going to the dump as formerly, are now being sold and shipped. The installation of concentrating plants at the Nipissing, Buffalo, Coniagas, McKinley-Darragh and Cobalt Central mines will partly counteract this downward tendency and also obviate the necessity of paying freight charges to Denver or Perth Amboy on worthless rock matter.

The cobalt, nickel and arsenic which the ores of Cobalt carry, in addition to the silver, have so far proved of little advantage to the mine owners. The nickel is not an important constituent, but cobalt and arsenic are valuable, and when the refining stage is reached in Ontario these elements will form an important source of revenue. At present nickel and arsenic bring nothing and cobalt is paid for only when in excess of 6 per cent. None of the reduction plants so far projected or begun has yet started operations except that of the Canadian Copper Company at Copper Cliff which has treated since its erection a considerable proportion of the ore produced at Cobalt, particularly of the higher grades. At Deloro, the former seat of the arsenic industry in this Province, the Deloro Mining

and Reduction Company is at this moment starting its new works, where it intends treating the argentiferous ores of Cobalt as well as the auriferous mispickel of the Hastings district.

The chief producers in the Cobalt district are the Nipissing, Coniagas, O'Brien, La Rose, Buffalo, Kerr Lake, Tretheway, Right of Way, Silver Queen, Drummond, McKinley-Darragh-Savage, Temiskaming & Hudson Bay, Temiskaming, Foster, Nova Scotia, Townsite, Green-Meehan, Colonial, Cobalt Lake, University, Crown Reserve, and Standard. The Silver Leaf and Red Rock have also shipped some ore, and the Temiskaming, Cobalt Lake, and Imperial shipped a carload each of silver-free cobalt ore. The Temiskaming & Northern Ontario Railway commission is building a spur line from Cobalt to Kerr Lake by way of the Gillies limit. The Government mine on the latter is being opened up, and shipments of ore will no doubt shortly begin.

The declining price of silver during the latter part of 1907, and the complexity of the ores, led the ore buyers, principally the American Smelting and Refining Company and the Orford Copper Company, to make their terms of purchase more stringent, and to impose severe penalties for excess of insoluble matter and other deleterious elements. When concentration becomes more general in the camp these difficulties may be eliminated at least in part.

(By R. Meeks.) Early in 1908 the Montreal Reduction and Smelting Company of Canada, Ltd., had nearly completed its plant at Trout Lake, near North Bay, Ontario.

The Montreal river district lies 55 miles northwest of Latchford and Cobalt. The mineralized area lies mostly in James township and on a large number of claims native silver has been found. The surface showing is not as rich as at Cobalt, but it is believed that with depth the veins will widen and become fully as rich as in the older district. The district is practically undeveloped and so far most of the work has been done to comply with the assessment requirements. A few shafts have been sunk to a maximum depth of about 50 ft. and the showing has been encouraging.

The country rock is principally diabase and quartzite with here and there conglomerate. The diabase extends eastward about 1½ miles into Tudhope township at which point granite completely cuts off the diabase. A few veins have been found in the granite, but these are, so far, valueless. The silver-bearing ground comprises a large part of James township and a strip of from one to one and one-half miles partly surrounding it. Outside of this area no finds of importance have been made, except in the Silver Lake district to the westward. The veins are narrow on the surface and vary in width from a knife-edge up to about 5 or 6 in. The filling is usually calcite but some partly decomposed quartz is also encountered and on one claim, in Smythe, barite is found. The mineral portion is

smaltite, niccolite and native silver and the blooms of cobalt and nickel. In this respect they resemble the veins of Cobalt, but unlike that camp most of the discoveries have small amounts of chalcopyrite or bornite, or both.

The district is entirely too young and undeveloped to form a definite opinion of the mineral resources but the possibilities are good and with the completion of the railroad to the camp the development should be rapid.

Late in 1907 discoveries of veins, similar to those at Cobalt were made in the southern part of Lorraine township. The ore is said to be rich and in considerable quantities.

During the 11 months from May 1, 1906, to March 31, 1907, the Nipissing Mining Company, Ltd., which is the operating company of the Nipissing Mines Company, produced 1103 tons of first-class ore; 1296 tons of second-class ore; 33 tons of cobalt ore; total 2432 tons. The proceeds from this ore were $1,053,299 or $433 per ton. Operating expenses amounted to $195,436 or $80 per ton. The net earnings were $866,095 and the surplus from the previous year of $624,628 gave a total surplus of $1,490,723 on March 31, 1907. During the period from Feb. 1, 1907, to Dec. 31, 1907, the income from ore production, including ore on hand, in transit and in process of refining, was $1,298,636. It was announced that the company will erect a concentrating plant at a cost of $75,000 early in 1908.

The Cobalt Silver Queen, Ltd., shipped in the year ended March 31, 1908, twenty carloads of ore for which was received $112,552.89. In addition $61,860.90 was due from the smelters, making a total of $174,421.79. Expenses for mining, development, new construction and general operations were $90,285.38. The balance on March 31, 1908, was approximately $120,000. Diamond drill explorations proved silver at a total depth of 400 ft. from the surface and the shaft will be sunk to that point. Two new veins were discovered both of which are high-grade.

The Buffalo Mines, Ltd., in the six months from May 1 to Oct. 31, 1907, produced 660.11 tons of ore containing 318,999.88 oz. silver and 5,646.19 lb. cobalt which was sold for $214,064.38. In addition nine bars of bullion were sold for $2,707.90, making the total receipts from ore and bullion $216,772.28. Treatment and transportation charges were $29,094.66, making the net ore and bullion receipts $187,677.42. The operating expenses were $79,396.71, of which mining ($36,864) and installation and repairs ($28,532) were the largest items. The total net earnings for the period were $108,281, out of which $54,000 was paid in dividends. The total net surplus was stated to be $82,320.94. Late in the year a concentrating plant was completed which contains rolls, a Huntington mill, Wilfley tables, vanners, slime tanks and cyaniding tanks. The plant is able to treat 15 tons of slimes per day.

According to Albert R. Ledoux, in a paper read before the Canadian Mining Institute at Toronto in 1907, his sampling works handled since Jan., 1905, a total of 366 carload lots of ore and 52 other lots—less than carloads—including nuggets coming either separately consigned or as part of a carload. These nuggets are not pure silver, but run anywhere from 700 to 870 parts of silver per thousand. There are more or less gangue and other minerals associated with the silver, and the metallic silver itself, visibly free from gangue, runs about 950 fine.

Leaving out of consideration the nuggets and native silver, and including only the lots of regular ore, a review of 394 lots sampled, shows that the highest lot ran 7402 oz. of silver to the ton, the next in order being 6909, 6413, 6163, and 5948 oz. per ton.

PROPORTION OF SILVER IN 394 LOTS OF COBALT ORE.

Silver Content.	No. of Lots.	Approximate Per Cent. of Whole.	Silver Content.	No. of Lots.	Approximate Per Cent. of Whole.
Over 6000 oz.	4	1.00	700 to 800 oz.	12	3.00
5000 to 6000 oz.	3	0.75	600 to 700 oz.	21	5.25
4000 to 5000 oz.	12	3.00	500 to 600 oz.	10	2.50
3000 to 4000 oz.	17	4.25	400 to 500 oz.	13	3.25
2000 to 3000 oz.	39	10.00	300 to 400 oz.	20	5.00
1000 to 2000 oz.	72	18.25	200 to 300 oz.	44	11.25
900 to 1000 oz.	11	2.75	100 to 200 oz.	66	17.00
800 to 900 oz.	7	1.75	Less than 100 oz.	43	11.00

While the greater part of shipments of cobalt have come to New York, some have gone abroad and many have gone to Copper Cliff, Ontario.

Silver, of course, in point of value is the most important element. The highest percentage of cobalt found in any one shipment is 11.96 per cent., the average being 5.99 per cent. The highest assay for nickel in any car is 12.49 per cent., the average being 3.66 per cent. The highest percentage of arsenic is 59.32 per cent., the average, 27.12 per cent. Mixed with the nuggets are sulphides, arsenides, etc., and the gangue matter.

New Caledonia (By G. M. Colvocoresses).—Nickel, chrome and cobalt ores have long been the chief mineral exports of New Caledonia and the mines are distributed over many parts of the island except the northern half of the east coast. These ores are always found in the serpentine formation, the serpentine resulting from the decomposition of the peridotite, which covered about one-third of the surface of the island.

Although there is always some cobalt, say about 0.2 per cent., associated with the nickel ores and some nickel, say 0.8 per cent., associated with the cobalt ores, yet the workable deposits of these two metals are always found separately, often long distances apart. The cobalt ore is associated with manganese, occurring as cobaltiferous manganese wad, carrying from 2 to 9 per cent. CoO.

By far the most important New Caledonian industry is the production of nickel ore. This ore is a product of decomposition of the peridotite and in this case the result is the hydrous silicate of nickel and magnesia, known as garnierite or noumeaite. The ore has a hardness of from 2 to 3 and a specific gravity of from 2.2 to 2.8. Its color is naturally dark green, but excess of magnesia may render it pale green, and the presence of iron gives it a chocolate or brownish yellow tinge. As taken from the mine, the great bulk of the impure ore resembles hard yellow clay, or decomposing rock, and it always carries from 10 to 25 per cent. of uncombined water. The nickel is always found in surface deposits. These are situated on spurs on the sides of plateaus more generally on those forming outposts of the Chaine Centrale. The hight of these deposits varies, but may be said in general to be between 400 and 700 m. Usually a mine does not consist of one large deposit, but rather of a group of small deposits with barren ground between. Thus one plateau contained eight large producing quarries within a radius of 2000 m. At another mine there is a single large deposit estimated to contain 250,000 tons of ore, and on an adjacent spur, 1000 m. away, is another deposit on which about 60,000 tons have been developed. Another large mine has three distinct sets of quarries, one at either end and one in the center of the flat-topped spur on which it is situated, the distance between the extremities of the quarries being 1400 m. Other mines consist of six, or even more, small quarries, each one of which may yield 5000 to 10,000 tons. Practically all these deposits have one physical feature in common, viz., they overlook a river or stream valley, whose sides rise at a very sharp angle. It is a feature in New Caledonian mining that in all mines (nickel, chrome and cobalt) the ore is mined from quarries or adits (excepting a few cobalt shafts).

The nickel content of the ore is far from uniform, the same quarry often producing small lots of 15 per cent. ore, larger ones of 10 per cent. and the bulk of 7 per cent. The ore contracts call for a certain minimum grade and a higher price per kilogram of metal contained is paid on a sliding scale for higher grades of ore, so that it is necessary to calculate just what grade of ore each mine can furnish with the greatest economic advantage, and to mix the ore accordingly. For many years no ore could be profitably shipped which did not contain 7 per cent. nickel on the dry

SHIPMENTS OF NICKEL AND COBALT ORES FROM NEW CALEDONIA. (a)
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905	1906	1907
Nickel ore ...	103,908	100,319	133,676	129,653	77,360	98,655	125,289	130,688	101,708
Cobalt ore...	3,200	2,400	3,110	7,512	8,292	8,961	7,919	2,487	3,943

(a) Reported by *Le Bulletin du Commerce*, Noumea.

assay, but the crude ore invariably contains from 15 to 28 per cent. of uncombined moisture (average 20 per cent.), so that a shipment of ore spoken of as 7 per cent. stuff really contains only about 5.6 per cent. nickel, or 56 kg. of nickel to the metric ton. The ocean freight and insurance to Europe are $7 per ton on the average.

Queensland.—It is reported that a discovery of nickel has been made on the south-east side of the Meredith range, between Parson's Hood and Roseberry.

PETROLEUM.

By Harold C. George.

In total production and general prosperity, 1907 was the banner year for the petroleum industry in the United States. The previous year, 1906, showed a decrease in the production of about 8,000,000 bbl. below that of 1905. The total production of petroleum in the United States in 1907 was 164,347,930 bbl., against 131,771,505 bbl. in 1906, and 139,889,210 bbl. in 1905.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.
(In barrels of 42 gal.)

Field.		1901	1902	1903	1904	1905	1906	1907
California (a)		8,786,330	13,973,500	24,337,828	28,476,025	35,671,000	30,538,000	40,085,000
Colorado		460,520	396,901	483,925	(b) 501,763	(e) 550,000	600,000	400,000
Gulf	Texas	4,393,660	18,083,658	17,955,572	21,672,111	30,354,263	12,666,000	12,350,000
	Louisiana		548,617	917,771	6,611,419	9,672,015	7,100,000	4,620,000
Illinois		......	......	......	......	......	4,900,000	24,540,024
Lima	Indiana	5,757,086	7,535,561	9,177,122	10,744,849	22,102,108	25,680,000	8,030,000
	Ohio	16,176,293	15,877,730	14,893,853	13,350,060			
Mid-Continental (c)		179,150	359,123	1,157,110	5,617,527	12,000,000	21,929,905	47,556,906
Ken.-Tennessee		(f)	(f)	(f)	998,284	(e) 1,200,000	1,000,000	1,250,000
Pennsylvania (d)		33,618,180	32,018,787	29,897,815	(b)30,410,183	28,324,324	27,345,600	25,500,000
Wyoming		5,400	6,253	8,960	11,542	(e) 12,500	13,000	13,000
Others		2,585	957	3,000	2,572	(e) 3,000	4,000	3,000
Total		69,379,204	87,801,087	98,832,956	118,396,335	139,889,210	131,771,505	164,347,930

(a) Reported by the California Producers' Association, except the statistics for 1907, which are of our own collection. (b) Statistics of the U. S. Geological Survey. (c) Kansas and Oklahoma. (d) Pennsylvania, New York, West Virginia, Eastern Ohio, and, until 1904, Kentucky and Tennessee. (e) Estimated. (f) Included in Pennsylvania.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (In gallons.)
(1=1000 in quantities and values.) (a)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffin		Residuum.		Totals.	
1897	121,864	$5,044	13,704	$1,020	804,446	$46,876	52,659	$6,732	(b)12,247	$ 335	1,004,920	$60,007
1898	120,436	5,016	17,258	1,071	764,823	38,895	65,526	7,626	(b)30,436	815	998,479	53,423
1899	117,690	5,958	18,210	1,597	733,382	49,172	71,116	8,658	(b)21,609	658	962,007	66,043
1900	133,161	7,341	18,570	1,681	739,163	54,693	71,211	9,933	(b)19,750	845	986,855	74,493
1901	127,008	6,038	21,685	1,742	827,479	53,491	75,306	10,260	(b)27,596	1,255	1,079,059	72,786
1902	145,234	6,331	19,683	1,393	778,801	49,079	82,200	10,872	(b)38,316	922	1,064,234	68,597
1903	126,512	6,782	12,973	1,519	691,837	51 356	95,622	12,690	(b) 9,753	282	936,697	72,629
1904	111,176	6,351	24,989	2,322	761,358	58,384	89,738	12,389	34 904	1,174	1,022,165	80,620
1905	126,185	6,086	28,420	2,215	881,450	54,901	113,730	14,312	70,728	2,128	1,220,513	79,641
1906	148,045	7,731	27,545	2,488	878,284	54,858	151,269	18,690	64,645	1,971	1,269,788	85,738
1907	88,495	5,273	28,080	2,969	844,996	56,454	147,558	18,325	74,162	2,459	1,183,292	85,480

(a) In addition to the above, the following quantities of paraffin and paraffin wax were exported (1=1000): 1897, 136,069 ℔. ($5,284); 1898, 166,317 ℔. ($6,363); 1899, 181,861 ℔. ($7,650); 1900, 157,108 ℔. ($8,186); 1901, 151,695 ℔. ($7,960); 1902, 175,268 ℔. ($8,398); 1903, 204,120 ℔. ($9,596); 1904, 174,582 ℔.($8,273); 1905, 160,836 ℔. ($7,873); 1906, 173,504 ℔. ($8,463); 1907, 207,504 ℔. ($10,209). (b) Reported in barrels of 42 gallons.

EXPORTS OF MINERAL OILS FROM PORTS OF THE UNITED STATES, AND SHIPMENTS TO ALASKA AND HAWAII FROM PACIFIC PORTS IN 1907. (*a*)

(In gallons.)

	Crude.	Refined or manufactured.			
		Naphtha, etc.	Illuminating.	Lubricating.	Residuum.
Exports from Pacific ports	14,314,977	130,171	54,845,893	550,240	90,957
Exports from all other ports	88,494,616	28,080,439	844,996,003	147,558,283	74,162,347
Total exports from the United States	102,809,593	28,210,610	899,841,896	148,108,523	74,253,304
Shipments to Alaska from Pacific ports.	9,104,300	640,881	515,044	100,145	....
Shipments to Hawaii from Pacific ports	39,916,400	484,435	1,427,577	262,468	....

(*a*) Figures supplied by Lewis H. Eddy, San Francisco, Cal.

The most important features of the petroleum industry in the United States in 1907 are summarized as follows: (1) The extension in area and the great increase in production in the Mid-Continental field, which increased from 22 million barrels in 1906, to 47½ million barrels in 1907. (2) The great increase in the production of oil in Illinois, from five million barrels in 1906, to 25 million barrels in 1907. (3) The growth of the consumption of the fuel oil in California, which in 1907 exceeded the total production of the State by several million barrels. (4) The decrease in the supply of oil from the Gulf States. (5) The great decrease in the production of oil from the Lima field. (6) The continued decrease in the production of high-grade petroleum in the Appalachian fields.

PETROLEUM OUTPUT OF THE WORLD. (*a*)

(In metric tons.)

	1902	1903	1904	1905	1906	1907
United States	11,906,000	13,402,000	16,055,000	18,969,000	17,862,000	22,287,985
Russia	10,445,536	9,759,214	10,058,968	7,505,637	8,167,934	8,435,708
Sumatra, Java, Borneo	732,000	662,767	955,957	1,062,224	1 186,907	(*c*) 1,200,000
Galicia	576,000	675,518	826,077	794,862	739,885	1,176,000
Rumania	310,000	384,303	497,000	614,870	887,000	1,129,097
India	209,000	352,848	475,869	581,519	564.470	490,944
Other countries	270,000	250,000	250,000	350,000	(*b*) 350,000	(*b*) 350,000
Total	24,448,536	25,486,650	29,118,871	29,878,112	29,758,196	35,069,734

(*a*) In the above table the statistics for the United States are computed from the production reported in barrels as given in the first table of this article. As a gallon of the crude petroleum found in the United States varies in weight from 6.41 to 7.83 lb., the oil in a barrel varies from 269.22 to 328.86 lb. The arithmetical mean of these figures is 299.04 lb., which figure has been used as a factor in converting the output stated in barrels into metric tons. This is not strictly correct, because in the period from 1900 to 1907 the proportion of petroleum production of various gravities has altered materially, especially because of the largely increased production in California, Texas and Louisiana. However, the method adopted is as near an approximation as can be made at this time. (*b*) In order to arrive at an approximate total, the production has been estimated, taking the same figures as for the previous year. (*c*) Estimated.

Review of the Oil Fields of the United States.

Appalachian.—The production of the Appalachian high-grade oilfields declined to a low point, and it is with the greatest difficulty that it is main-

tained at about 70,000 bbl. a day. Desirable territory is becoming very scarce and only wells of small production are to be found by drilling. There was an earnest search for new pools constantly going on, but every month showed a large percentage of dry holes and a constantly decreasing production. West Virginia, the region which formerly furnished many large gushers, now seldom produces a large well, and those that are found are short lived. The obliteration of the color line in classifying high-grade petroleum, and the advance in prices over that of 1906, undoubtedly stimulated operations; and thus maintained the production nearly up to that of the two previous years. Conditions were very similar to those of 1906. No new pools of importance were discovered and the new wells drilled were widely scattered. The accompanying table gives a summary of the operations in the Appalachian fields in 1907.

OPERATIONS IN THE APPALACHIAN OILFIELD IN 1907.

Field.	Number of wells drilled.	Daily production.	Daily production per well drilled.	Per cent. Dry Holes.
		Bbl.	Bbl.	
New York	575	1,114	1.9	16.0
Pennsylvania	3,611	12,176	3.3	21.0
West Virginia	1,320	21,300	16.1	38.0
Southeast Ohio	1,335	6,793	5.9	39.5
Kentucky and Tennessee	212	2,006	9.4	32.0
Total	7,053	43,389	6.1	27.0

Pennsylvania ranks first in the number of wells completed, while West Virginia ranks first in production. The total production of the Appalachian field in 1907 was 25,500,000 bbl., as compared with 27,345,600 in 1906 and 28,324,324 in 1905.

Both West Virginia and southeastern Ohio showed a large percentage of dry holes, which helped greatly to reduce the average daily production per well drilled. The average daily production of the new wells in the Appalachian field was estimated on the daily average production of each for the first month after being drilled. The daily production of the new wells from the time they were drilled until Jan. 1, 1908, is doubtless much less, probably about 35,000 bbl., or half of the total daily production, including that from both new and old wells.

The record of the petroleum developments in Tennessee and Kentucky shows an expansion of the industry in both States. Their total production of petroleum in 1907 was 1.250.000 bbl., as compared with 1,000,000 bbl. in 1906. The prices paid were much higher than in 1906, averaging about 85c. per bbl., as compared with 77c. per bbl. in 1906. The average price paid for the better grade oil was about $1.20 per bbl. and for the poorer grade $0.75 per barrel.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average
1899.....	$1.17	$1.15	$1.13	$1.13	$1.13	$1.13¼	$1.22½	$1.27½	$1.44½	$1.50⅞	$1.57⅜	$1.65⅛	$1.29⅜
1900.....	1.66⅔	1.68	1.68	1.55	1.39⅜	1.25½	1.25⅝	1.25½	1.23	1.10⅜	1.06¾	1.08¾	1.35¼
1901.....	1.69½	1.25	1.29	1.20½	1.07⅜	1.05	1.13⅜	1.25	1.25⅜	1.30	1.30	1.21	1.21
1902.....	1.15	1.15	1.15	1.17½	1.20	1.20	1.22	1.22	1.22	1.28	1.38	1.49	1.24
1903.....	1.52	1.50	1.50	1.51½	1.51½	1.50	1.53	1.56	1.56	1.69½	1.79½	1.88½	1.59
1904.....	1.85	1.82½	1.73	1.65	1.59½	1.57	1.53½	1.50	1.54½	1.56	1.58	1.55	1.62⅜
1905.....	1.43½	1.39	1.37½	1.32½	1.28	1.27	1.27	1.27	1.39	1.58½	1.59½	1.58	1.31⅜
1906.....	1.58	1.58	1.58	1.60	1.64	1.64	1.63½	1.58¾	1.58	1.58	1.58	1.58	1.5969
1907.....	1.58	1.63	1.68	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.7425

California.—The production of petroleum in California increased greatly in 1907, amounting to a little more than 40,000,000 bbl. A conservative estimate of the consumption of California crude in 1907 is 41,000,000 bbl., distributed as follows: Railways (steam), 15,000,000 bbl.; Point Richmond refinery (Standard), 3,000,000; Union Oil refinery (Oleum), 1,750,000; All others, 23 in number, 4,000,000; 360 steamers, large and small, that have been granted permits to burn oil fuel, 2,750,000; Japan contract, 2,000,000; Chile contract, 1,000,000; Panama pipe line, eastern trade, together with the Government requirements for canal construction, 3,000,000; all other requirements, California, Oregon, Washington, British Columbia and Alaska, 8,500,000.

G. A. Scott, secretary of the California Oil Producers' Association, recently made an interesting report on conditions in the California oil industry. Among other things he said: "Reliable information has been obtained from all fields in the State, with perhaps the exception of the Fullerton field, the operators of which district hesitate to give out information. I find that the present actual production of oil throughout the State is somewhere from 105,000 to 110,000 bbl. per day; in other words, at the rate of 38,000,000 to 40,000,000 bbl. per year, which is a larger production than has ever before been reported. The following is my estimate of the daily production from the different fields: Kern county fields, 37,000 bbl.; Coalinga fields, 27,000; Santa Maria fields, 27,000; Los Angeles county fields, 8000; Whittier field, 2000; Fullerton field, 7000; other fields, 1500.

"For the first time in several years every well in the State, with hardly an exception, is producing its maximum quantity of oil, and there is no longer the statement made that there is an immense possible supply of oil which is held in check and not allowed to come upon the market; in fact the most strenuous efforts are being put forth everywhere to bring every possible production on the market to overcome the shortage which every producer now sees to exist. The amount of oil held in storage in this State is practically nothing, with the exception of the oil held by the Standard Oil Company in the Kern river fields; outside of that it is doubtful if

there is 15 days supply of oil held in the tanks of the producers and consumers. The oil held by the Standard Oil Company in the Kern river fields is estimated to amount to between ten and eleven million barrels.

"The consumption of oil in the State is from 20,000 to 25,000 bbl. per day in excess of the production, and the shortage is increasing daily. A great many consumers were discouraged during 1907 by a failure to obtain oil deliveries promptly, owing to the lack of transportation facilities. If the actual demands for oil were filled there would be a shortage amounting to 30,000 bbl. per day. There is also a great deal of new business in sight for 1908, which will consume large quantities of oil. The question has been repeatedly asked oil men lately, whether there is enough oil in the State to satisfy the immense demand now in sight. Judging from the fact that the Kern river field, the oldest large field in the State, is holding up its production so near to its maximum, in spite of the reports that have been circulated in regard to deterioration from water and other causes, and the fact that many of the fields of the State show the same staying qualities, it is evident to me that from 40 to 50 million barrels of oil can be produced annually in California for many years to come.

"No large or important areas of oil-bearing lands were developed during 1907, and unless some immense fields are brought in, no danger of over production need be feared, even if the price of oil should go much higher. It seems inevitable that the price of oil to consumers must go higher at all Coast points of delivery, bringing such a price as will be attractive to the producer, at the same time giving the marketing and transportation companies a good profit, and still enabling the consumers to save millions of dollars annually by using oil instead of other fuel."

The following is from Bulletin No. 6 of the California Petroleum Miners' Association, prepared by the secretary, Dr. C. T. Deane: "Among the many oil fields of California there are three that have proved great producers, viz., the Kern river, the Coalinga and the Santa Maria. Coalinga promises to surpass even Kern in its acreage and production of oil. Its proved ground is being continually extended by the addition of new territory. The fields north of Coalinga have not as yet proved of permanent value. The southern fields of the State, compared with the three districts above mentioned, have been disappointing as far as a large permanent output is concerned."

The following data are taken from a recent report of the U. S. Geological Survey on California oils: "As the natural oils range in gravity from 10 to over 50 deg. B., the refinery products are varied. The oils of higher grade produce gasoline, benzine and kerosene, and No. 1 and No. 2 distillates, stove distillate and a residuum of fuel and road oil; the heavier oils yield No. 1 and No. 2 distillate, stove distillate, lubricating, fuel, and road oils with a residuum of asphalt. The No. 1 and No. 2 distillates are used

mostly in gas engines. The stove distillate is used where the fuel is consumed without steam pressure; the regular fuel oils are usually forced into the fire boxes by means of special pressure burners."

Until recently many things conspired to keep down the price of crude oil in California, but the present situation is more encouraging. Producers have generally made the mistake in the past of overestimating the production of their wells. This in turn has made the purchasers assume that the production would be so great that the producers would be willing to sell their oil at a very low price.

There is no doubt that California's supply of reserve oil is fast disappearing, and this is simply due to the fact that the State is using more oil than it is producing. The increase in demand is caused mostly by the export trade. The Panama pipe line constructed by the Union Oil Company, of California, as an outlet for California crude, has been making no regular shipments. Now that the contract price of California oil has advanced to $1 per bbl., as compared with 35c. to 50c. per bbl. in 1905 and 1906, the Union Oil Company will probably be contented to take its share of the local business instead of transporting oil 3000 miles to the Isthmus by steamer. The scheme to supply the Atlantic Coast and Europe with California oil through the Panama pipe line looked good when the market

PRODUCTION OF CRUDE OIL IN CALIFORNIA. (a)
(In barrels)

District.	1902	1903	1904	1905	1906	1907
Coalinga	500,750	2,138,058	5,114,000	8,869 000	8,500,000	10,850,000
Sta. Maria and Lompoc	116,500	208,890	670,500	5,300,000	5,400,000	10,000,000
Kern River	8,872,115	16,342,100	17,500,000	14,000,000	11,000,000	11,000,000
Los Angeles	1,047 300	793,765	1,200,000	3,000,000	1,700,000	2,000,000
Sunset	144,200	353,100	400,000	400,000	500,000	600,000
Midway	50,000	29,200	910	5,000	5,000	500,000
McKittrick	639,500	1,353,500	1,875,925	720,000	500,000	1,000,000
Newhall and Ventura	626,540	683,500	663,100	500,000	400,000	500,000
Fullerton and Brea Canyon	1,195,015	1,427,700	147,500	1,750,000	1,700,000	2,200,000
Whittier and Puente	687,030	878,015	748,000	960,000	750,000	1,000,000
Summerland	94,550	131,000	120,000	75,000	56,000	....
Sargents	....	....	35,090	20,000	15,000	....
Halfmoon Bay	....	....	1,000	2,000	2,000	....
Arroyo Grande	....	....	....	5,000	10,000	....
All others	....	....	....	....	....	350,000
Total	13,973,500	24,337,828	28,476,025	35,671,000	30,538,000	40,000,000

(a) Reported by the Petroleum Miners' Association.

PRODUCTION OF CRUDE OIL IN CALIFORNIA IN 1907. (a)

County.	Barrels.	Value. (b)	County.	Barrels.	Value. (b)
Fresno	8,990,000	$3,558,541	Santa Barbara	8,366,000	$3,311,291
Kern	15,600,000	6,216,666	Santa Clara and Santa Cruz	20,000	7,916
Los Angeles	4,324,000	1,711,459	Ventura	365,000	144,479
Orange	2,420,000	957,916	Total	40,085,000	$15,908,268

(a) Statistics collected by *The Mineral Industry*.
(b) Average price during first five months, 25c. per bbl.; average price during last seven months 50c. per bbl.

was low, but the chances are now that very little California oil will find its way to the Atlantic Coast by way of Panama as long as the local market is on a satisfactory basis. The great increase in the production of Oklahoma has also been a factor in the situation, making the sale of California oil difficult on the eastern seaboard.

(Communicated by Lewis H. Eddy.) Following is a description of the present oil fleet of the Associated Oil Company operating on the Pacific coast, recently augmented by the construction of the "W. S. Porter" and the conversion of the "Falls of Clyde" into a tank vessel. This fleet is now one of the largest on the Pacific coast, its increase being requisite to meet the transportation demands of the company occasioned by the increase in the demand for California petroleum.

SHIPS OF THE ASSOCIATED OIL COMPANY.

Name.	Class.	Gross tons.	Length feet.	Breadth feet.	Depth feet.	H.P.	Capacity barrels.
W. S. Porter	Steel Stmr.	4,904	385.0	49.7	30.2	3,000	53,000
Rosecrans	Stmr.	2,976	335.0	38.2	28.5	1,250	23,000
Monterey	Iron Schr.	1,854	260.0	39.5	26.0		19,000
Falls of Clyde	Steel Ship	1,809	266.1	40.0	25.1		19,000
Roderick Dhu	Iron Schr.	1,534	257.1	40.2	24.3		17,000
Marion Chilcott	Iron Ship	1,737	256.4	38.2	24.7		16,000
Santiago	Steel Schr.	979	207.6	33.1	21.5		11,000
Navigator	Steel Tug	414	134.3	26.0	16.0	1,000	1,550
Milton	Wood Tug	23	44.6	14.2	4.9	100	
Barge No. 1	Wooden		145.0	35.0	7.0		2,400
Barge No. 2	Wooden		120.0	28.0	7.0		1,600
Barge No. 3	Wooden	150	91.0	26.0	7.0		1,500

Colorado and Utah.—The production of crude petroleum in Colorado in 1907 was 400,000 bbl. This includes both the Florence and Boulder fields. The United Oil Company, of Denver, Colo., with refinery at Florence, buys all of the Colorado crude petroleum. The principal productive area in Colorado is in the vicinity of Florence. There are other regions in the State where the existence of oil is known, but where development has not been attempted because of remoteness from railway. The Boulder field is the only exception in this respect. In that field the United Oil Company drilled a number of wells during 1907, and reported an increased production of petroleum. In the Florence field the production in 1907 was about equal to that of 1906, an average of about 1000 bbl. per day.

The operations in the Florence field during 1907 were largely in the nature of prospect work. There was a considerable increase in the well drilling forces, new territory was exploited and a number of new wells sunk to the oil plane, but for the most part these new wells took the place

of the exhausted ones, so that the progress made in that direction only compensated for the delinquency of old wells. Deep drilling has been the policy of the company for the last two years, and the fact that it now pumps oil from a depth of 2500 to 3000 ft. is one of the most encouraging conditions. The most important work of the company was the enlargement and improvement of its manufacturing plant, which was completed by the close of the year, the new plant being equipped with every modern facility for the manufacture of all kinds of high-grade lubricating oils and paraffin wax. These improvements were made at a cost of about $200,000. The company now has facilities for handling all the petroleum that is produced in Colorado.

In the region east of Salt Lake, in Utah, prospecting was carried on to about the same extent as in 1906. Some Nevada and California oil men did considerable prospecting for petroleum in the Sinbad oil belt, near Price, Emery county. Some prospecting was also done during 1907 near Mt. Pleasant, San Pete county, where local capital was invested in boring for oil several years ago.

Illinois (By H. Foster Bain).—The year 1907 was a very prosperous one in the petroleum fields of Illinois. The area was extended rapidly to the southeast, many gaps were filled in, new and lower sands were tapped, additional pipe lines were laid, a new refinery was built and the output was phenomenal.

At the close of 1906 the number of producing wells was estimated at 4185, and 532 dry holes were known to have been drilled. The total number of producing wells Jan. 1, 1908, may be estimated at 9175, with 1260 dry holes. At this rate 88 per cent. of the holes put down have proved productive despite the fact that the outlines of the field are at many points yet to be determined. The new production in 1907 may be estimated at 139,163 bbl. daily. The detailed figures for the year are given in the accompanying table, being derived from the careful monthly records of the *Oil City Derrick*.

WELLS DRILLED IN ILLINOIS, 1907.

Month.	Completed.	Production, bbl.	Average initial production, bbl.	Dry Holes.
January	253	9,433	44	41
February	356	9,842	$32\frac{2}{3}$	55
March	351	10,392	$35\frac{5}{7}$	60
April	387	11,083	32	40
May	493	13,329	31	64
June	639	18,807	$33\frac{1}{3}$	75
July	521	17,375	$38\frac{3}{4}$	72
August	461	11,240	$27\frac{1}{5}$	45
September	400	10,967	$32\frac{1}{3}$	62
October	363	8,157	$25\frac{1}{2}$	82
November	430	9,780	$28\frac{1}{4}$	80
December	334	8,758	$31\frac{1}{17}$	52

The first oil was shipped from this field in June, 1905, and the shipments for that year, all of which went out in tank cars, amounted to 156,502 bbl. In 1906 a pipe line was extended into the territory, and the shipments were as follows: January, 55,680 bbl.; February, 65,209; March, 19,352; April, 102,862; May, 267,746; June, 410,654; July, 610,401; August, 778,463; September, 722,168; October, 463,819; November, 350,985; December, 538,130; Total, 4,385,470.

There are now collecting mains extending from north to south throughout the field and four 8-in. lines (or an equivalent) from Martinsville, the central pumping plant, eastward across Indiana. A new line is nearly ready for service running westward to a large refinery built this year near Alton, Ill., by the Standard Oil Company.

The pipe-line runs for 1907, given through the courtesy of the Ohio Oil Company, were as follows: January, 752,670 bbl.; February, 918,620; March, 1,494,598; April, 1,823,024; May, 2,094,194; June, 1,830,633; July, 2,376,281; August, 2,398,895; September, 2,560,592; October, 2,818,952; November, 2,464,980; December, 2,201,265; Total, 23,733,790 bbl. To these figures must be added something for the fuel oil shipped by cars from Duncanville, the oil used by the local refinery at Robinson, and the tank car shipments of the Pure, Sun, Cornplanter and other independent companies amounting to something over 800,000 bbl. The recorded production for 1907 was 24,540,024 bbl. and probably the actual production ran a little above this. The oil, in the main graded 32 deg. or better, and sold at the standard price of 68c. per bbl. Only a limited amount was lower and sold at 60c. Of the year's production 12,610,618 bbl. are stored in the field by the Ohio Oil Company and a large amount is in the producers' tanks.

The situation as relates to natural gas did not change materially in 1907. Gas is found somewhat generally with the oil in Clark and Crawford counties. The wells, while showing good pressure and fair initial capacity, have usually proved short lived and so far the gas has been of local value only. In 1906 gas to the value of $186,000 was utilized according to the U. S. Geological Survey. Probably not more than $250,000 worth was sold in 1907. The southern part of the field in Lawrence county has yielded practically no gas. Late in December two wells were brought in here which yielded respectively 3,000,000 and 6,000,000 cu.ft. per day from a depth of 1500 ft. Considerable confidence in their probable life is felt and it is possible that an important gasfield is about to be brought in. A gas main is being laid to Vincennes, Ind., the largest nearby town.

This enormous development was accomplished in a thoroughly businesslike and quiet manner. Leases are selling at very good prices and a bonus of $150 to $200 an acre with a royalty of one-eighth is not uncommonly demanded in the productive district. At the same time there was little

speculation by those not familiar with the oil business and its risks. Practically none of the usual stock-peddling companies was organized, and there is a strong sentiment against them.

Experienced men have found this field an unusually profitable one despite the high bonus asked and certain other drawbacks. One conservative operator estimates that three out of four will make money. It is by no means unusual for a well to flow enough oil to pay for itself by the time it is connected up, and initial productions of 1000 bbl. are not uncommon. So far the wells have stood up well under pumping. The most northerly, or Westfield pool, is the only one which is even approximately drilled in. It was here that oil was first found and the shallow depth, 350 to 400 ft., has made its exploitation rapid. In October a careful estimate showed that the wells of this pool were yielding an average of about 6 bbl. daily and many of them had been pumped more than two years. The Crawford county wells were at the same time estimated to be yielding 20 bbl., while those of Lawrence county were yielding 40 bbl.

The oil occurs in a number of isolated pools which, however, are being brought closer together by drilling. It is not improbable that they will eventually be found to overlap. To the north they are higher stratigraphically and also shallower in depth. The Westfield pool is in the upper coal measures. Most of the oil in Crawford county seems to come from the lower coal measures, well down toward the base. In Lawrence county there are two sands, the main production being from the Buchanow sand at 1300 ft. This probably represents the Mansfield sandstone of the Indiana geologists, an approximate equivalent of the Pottsville strata of the east. The Kirkwood sand at 1600 ft. may also be Pottsville, though this is as yet uncertain. Farther south in the Princeton, Ind., field, a still lower horizon in the Chester group is productive.

In general the work of 1907 resulted in extending the field to the south and in connecting up intervening territory. Wildcatting was active in other parts of the State, but so far without much result. Some gas was found near Medora and one or two oil wells were brought in at Sparta, but as yet too little has been done to test thoroughly any considerable portion of the field.

WELLS COMPLETED IN ILLINOIS IN 1907.

County.	Completed.	New Production.	Dry Holes.	County.	Completed.	New Production.	Dry Holes.
Crawford	2,840	83,263	376	Cumberland	152	3,612	13
Clarke	1,176	20,385	201	Coles	56	314	11
Lawrence	690	30,543	70	Edgar	25	118	14

Drilling was also carried on, but less actively, in Jasper, Wabash, Edwards, Williams, Fayette, Clinton, Richland, Clay, Douglas, Madison, Franklin, Washington, Randolph, McClean, Wayne, White, St. Clair, McDonough, Montgomery, Hamilton and Jefferson counties. According to the *Oil City Derrick*, this work resulted in the completion of 48 wells, of which five were producers to the amount of 28 barrels.

Topographic surveys were made over much of the eastern Illinois field, and the maps are now being drawn, preparatory to field use by the geologist.

Ohio and Indiana.—The Lima oil field, or more properly the oil fields of northwestern Ohio and Indiana, experienced a great decrease in production in 1907. Large numbers of wells were abandoned, and the new wells did not produce as did the new wells of former years. In 1907 there were 930 wells completed in northwest Ohio with a total daily production of 8100 bbl. or 8.7 bbl. per well drilled; 15 per cent. of the wells drilled were dry holes. Indiana showed 682 wells completed with a total daily production of 5673 bbl., or 8.3 bbl. per well; 20 per cent. of the wells drilled were dry holes. The total production of the Lima field in 1907 was 8,030,000 bbl. as compared with 25,680,000 bbl. in 1906. The average price paid for North Lima was 93½c. per bbl. and for South Lima and Indiana 88½c. per bbl.

Kansas and Oklahoma (By Erasmus Haworth).—During 1907 there were extensive developments in the Mid-Continental field. The oil development was confined almost entirely to the Indian Territory, now Oklahoma, but a large supply of gas was developed in Kansas. Likewise the total oil output in the Mid-Continental field was made very largely by Oklahoma. It is impracticable to quote figures for each State, as the statistics obtainable are gathered in such a way that no discrimination is made.

The Prairie Oil and Gas Company (Standard) was the principal consumer of oil, although other interests, such as independent pipe-line companies, independent refineries and dealers in fuel oil consumed a comparatively large amount. The accompanying table gives a summary of the monthly reports of the Prairie Oil and Gas Company, showing a total of 35,756,366.76 bbl. handled by this company. Much the greater part of this was used for refining purposes, so that only 10,762,333.34 bbl. were stored. This brings the total storage of the Prairie Oil and Gas Company at the close of 1907 up to the large sum of 33,703,367.74 bbl. To obtain the total production of the Mid-Continental field, we must add to these figures the consumption by the independent refineries, the consumption as fuel oil, and the runs by the independent pipe-line company.

Nine independent refineries were in operation during the year. The one at Paola was bought by the Standard Oil Company during the year; a few other smaller ones were operated irregularly and these, when in full operation, had a capacity of only a few hundred barrels per day. The largest refineries are at Humboldt and Chanute. A conservative estimate of the total consumption of the independent refineries plus the total crude oil used for fuel is 2,309,500 bbl.

The Gulf Pipe Line Company completed a line from the Glenn pool to

the Gulf of Mexico and began pumping oil about Oct. 1. This company began storing oil in January; before the pipe line was built it shipped largely by rail, and even yet ships small quantities by rail from wells not yet connected with the pipe line. A fair estimate of the business done by this company during the year is 5,381,794 bbl., the output during the last half of December only being estimated. The Texas company also is shipping large quantities of oil from the Mid-Continental field. It began shipping by rail in March; by October the company had completed its pipe line as far south as Dallas and hopes that this will reach tidewater by the end of the year. The Texas company handled a total of 3,359,265 bbl.

CRUDE OIL BOUGHT BY PRAIRIE OIL AND GAS COMPANY DURING 1907.

Month.	Total Runs, Bbl.	Daily Average, Bbl.	Deliveries, Bbl.	Stored, Bbl.
January	2,337,164.90	75,392.42	1,646,090.76	691,074.14
February	2,292,116.70	81,861.31	1,759,151.91	532,964.79
March	2,795,969.12	90,192.55	1,965,586.57	830,382.55
April	3,098,915.89	103,297.20	2,166,236.16	932,679.73
May	3,023,776.68	97,541.18	2,342,348.78	681,427.90
June	3,021,285.46	100,709.52	2,162,624.65	858,660.81
July	3,053,284.85	98,493.06	2,325,471.93	727,812.92
August	3,213,547.64	103,662.83	2,281,436.97	932,110.67
September	3,089,627.93	102,987.60	2,200,032.11	889,595.82
October	3,486,804.13	112,477.55	2,267,631.40	1,219,172.73
November	3,231,769.51	107,725.65	1,942,309.80	1,289,459.71
December	3,112,103.95	100,390.45	1,872,689.40	1,176,991.57
Total	35,756,366.76	....	24,931,610.44	10,762,333.34

The Haywood Company, of Texas, also is shipping extensively by rail, supplying fuel oil to various parts of Texas. This company produced about 750,000 bbl. The total production of the Mid-Continental field is summarized as follows: Prairie Oil and Gas Company, 35,756,366.76 bbl.; independent refineries and fuel oil, 2,309,500.00; The Gulf Pipe Line Company, 5,381,794.00; the Texas Company, 3,359,245.00; the Haywood Company, 750,000.00; total, 47,556,905.76. The most remarkable developments during 1907 were in the Glenn pool, situated about 10 miles south of Tulsa, Oklahoma. Here a number of wells with an initial capacity close to 2000 bbl. per day were developed and also many others producing more than 1000 bbl. per day. The Glenn, therefore, is the most remarkable pool yet developed in the Mid-Continental field. How long it will continue to be so productive, of course, no one can tell, but it is certainly one of the greatest oil fields ever developed in America.

About the middle of the year considerable excitement was caused by finding oil near the new town of Morris, 30 miles south from the Glenn pool toward Ardmore; a few good wells were obtained but not much

drilling had been done before a number of dry wells were obtained, a fact which checked development.

A nice field has been developed along Hog Shooter creek, from 6 to 15 miles southeast of Bartlesville. Late in 1906 a well was drilled and plugged; then a report was given out that the well was abandoned. But the company that drilled the well soon began to acquire other leases; the suspicion of others was aroused and a miniature boom resulted. In a short time a number of different companies were drilling on different leases and a nice group of oil and gas wells was developed; the larger of the oil wells have a capacity of about 500 bbl. oil per day and the gas wells from 5,000,000 to 15,000,000 cu.ft. of gas.

During the first half of December what appears to be a possible repetition of the Glenn pool was developed in the northeast part of Osage reservation, only a few miles west of Dewey, the largest well being on lot 32. Previously, the Dewey-Copan field was limited on the west by a number of dry wells. These checked developments in a westward direction, but someone finally grew bold enough to go a few miles farther west, with the results described. It is reported that one well has been obtained equal to, if not greater than the biggest well of the Glenn pool.

The districts productive at the close of 1906 remained equally productive during 1907. The shallow field in the Alluwee-Chelsea district sustained its production, although but few new wells were brought in, especially after the middle of the year. This was also true with reference to the Dewey field, the Bartlesville field, and practically all the others. A good healthy activity prevailed, but developments have been confined principally to pools already opened up.

A number of different rulings were given out by the Secretary of the Interior; some of these stimulated development, but the greater number had the opposite effect. During October a series of rulings were made regarding oil royalties, gas royalties, and the transfer of leases, which were so objectionable to operators that development work was practically stopped. The royalties in many instances were increased and in some other ways restrictions were enforced which were very objectionable to the operators.

Texas and Louisiana.—The prominent feature of 1907 was the unusually high price obtained for crude petroleum compared with the selling prices in other States of petroleum of a much higher specific gravity. Field operations produced nothing beyond the normal until November when the first gusher in the Anse Le Butte district in Louisiana was drilled in, too late in the year, however, to have any great influence on 1907 production. The 1907 production, as might be expected under prevailing conditions, shows a decline when compared with that of 1906, although the decrease was occasioned entirely by the reduced output of the Jennings

district of the coastal field. The old Texas districts in this field as a rule maintained, and in some cases increased their output, mainly by development in proved territory; this activity was due to the high prices paid for crude oil. The Spindletop field showed the remarkable increase of more than 500,000 bbl. over its 1906 output.

The production of petroleum in Texas in 1906 was 12,724,000 bbl. of which 11,600,000 bbl. came from the Gulf coastal field. The Louisiana output in 1906 was 7,110,000 bbl. and the total output of the Gulf coastal field, 19,834,000 bbl. The 1907 production of Texas was approximately 12,350,000 bbl., of which the North Texas fields produced about 850,000 bbl. The Louisiana production was 4,620,000 bbl., making a total for the Gulf coastal field of 16,970,000 bbl. There was a reduction of 2,480,000 bbl. in Louisiana, a decrease of 316,000 bbl. in Texas, or a decrease of 2,796,000 bbl. in the coastal field, as compared with 1906. The producers, however, received about $14,000,000 for their petroleum, a sum $4,000,000 greater than the value of the 1906 output.

PRODUCTION OF GULF COAST FIELDS. (*a*)

(In barrels.)

District.	1905	1906	1907	District.	1905	1906	1907
Spindletop	1,652,780	1,077,492	1,613,513	Piedras Pintas	....	....	8,354
Sour Lake	3,362,153	2,156,010	2,354,997	Corsicana	311,544	332,622	248,138
Saratoga	3,125,028	2,182,057	2,198,585	Powell	132,866	673,221	600,454
Batson	3,774,841	2,289,507	2,166,554	Henrietta	75,592	111,072	46,937
Humble	15,594,310	3,571,445	2,930,842	South Bosgue	300	1,300	8,000
Dayton	60,294	92,850	108,036	Jennings	8,891,416	9,025,174	4,450,725
Matagorda	46,471	8,000	4,500	Welsh	10,000	23,996	36,000
Hoskins Mound	....	72,591	12,000	Anse La Butte	9,000	23,708	76,938
San Antonio (Mission field)	....	....	5,000	Caddo	....	4,650	50,000

The year developed no new big field in Texas, and while the old districts were actively prospected by deepening old wells and drilling new ones in proved territory, fewer wildcats were drilled, mainly because operators preferred to take a chance in Oklahoma. The percentage of completed producers of fair capacity in the coastal field was high. According to the *Oil Investors' Journal*, the number of wells completed was 890 of which 669 are classed as oil producers and 11 as gassers. This record compares favorably with the 1906 data, which showed 508 producers out of 728 completed wells. The Jennings district was the largest producer in the coastal field with an output of approximately 4,450,725 bbl.

The average price paid producers in 1906 was 46c. per bbl., the Texas coastal producers averaging 50c., while the Jennings producers received only 39c. Taking the posted pipe-line credit balance prices as a basis of the prices paid for crude in 1907, it shows that quotations on Jan. 1, 1907,

ranged from 55½c. for Batson heavy to 76c. for Sour Lake light. Under a strong market prices steadily advanced until on Aug. 1st, the posted prices ranged from 85c. for Batson heavy to 95c. for Spindletop and Humble, and some oil was sold on contracts for more than $1 per bbl. The market was weaker in September with prices unchanged, and when the monthly report showed a surplus, prices promptly slumped several cents and the decline continued in October and November, until on Dec. 1, prices ranged from 67c. for Dayton to 80c. for Spindletop, and 75c. for Jennings. Further reductions were made, so that at the end of the year prices were on a level with those of November, 1906, with a tendency toward decline. The average price for 1907 received by coastal field producers was between 75c. and 80c. per bbl. The causes contributing to end the era of abnormally high prices, as compared with other fields, for crude petroleum of the gravity produced in Texas and Louisiana are: (1) The refinery consumption of Mid-Continent crude displacing about 20,000 bbl. daily of Texas crude; (2) The restricted consumption for fuel caused by the high price and to a small extent the use for fuel of the heavier grades of Mid-Continent crude shipped to Texas users by tank car; (3) The general opinion that the Anse Le Butte district will be a large producer.

The petroleum stocks on Dec. 31, 1906, amounted to 8,150,000 bbl. and in every month up to and including August the consumption exceeded the output until on Sept. 1 the oil stocks were estimated at 3,450,000 bbl. In September for the first time in two years the output of the wells exceeded the consumption and October and November added a surplus to the stored oil. At the end of the year the stocks amounted to about 4,000,000 bbl., a decrease of 4,150,000 bbl. during 1907.

In order to obtain an adequate supply of crude suitable for refining at a lower price both the Texas Company and the Gulf Refining Company, operating refineries at Port Arthur, constructed pipe lines to the Glenn pool of the Mid-Continent field. The Gulf pipe line runs north from Sour Lake to Tulsa, a distance of about 415 miles, and was completed in six months. It is 8 in. in diameter and has an estimated capacity of about 25,000 bbl. daily. From Sour Lake to Port Arthur the company has two 6-in. lines. The completed pipe line has been in operation only since September, but rail shipments of Mid-Continent crude were made as early as February, 1907, and increased in succeeding months.

The Texas Company pipe line runs from the Glenn field via Dallas to Humble where it connects with a pipe line to Port Arthur. While not in operation yet (January, 1908) for its entire length the portion from Dallas north has been in use since July and the crude piped to Dallas has been stored or shipped south by tank cars. Practically none of the Texas refineries now utilize coastal crude, and since April, 1907, the Standard

Oil Company has made no shipments of crude by water to eastern ports.

During 1907 the State of Texas vigorously prosecuted the Standard Oil Company or its alleged subsidiary or affiliated corporations. The Waters Pierce Oil Company was convicted of violating the anti-trust law of Texas, fined $1,549,500 and placed in the hands of a receiver appointed by the State courts. The Security Oil Company, Union Tank Line and other corporations were sued in November and an injunction was issued forbidding the removal of any of their property from the jurisdiction of the court. This order restrained the shipment, by water from Sabine, of refined oil and other products made at the large Beaumont refinery of the Security Oil Company. It also prevented the return of empty tank cars to the Glenn field and the Union Tank Line naturally refused to allow any more loaded cars to cross the Texas border. No modification of the injunction could be secured and having taken up its gathering lines in Texas the Security company was forced to close its plant.

In view of these actions which were directed almost entirely against the refining and distributing interests, it should be noted that there are, and have been for several years, large independent refineries at Port Arthur and other places which have been operated steadily and apparently successfully in competition with the Standard Oil Company. With regard to the market for crude it is well to quote the findings of Commissioner of Corporations Herbert Knox Smith, who in his report states "that in the coastal oil field the price of crude seems to be determined by genuine competition and regulated solely by the law of supply and demand." That this finding is just and accurate is undoubted, so that there is at least one large oil field in which the Standard Oil Company does not dictate crude oil prices.

While the officials grappled with anti-trust violators the State legislature introduced various enactments directed against producers and dealers in petroleum. The most drastic of these, "The Gross Receipts Tax Bill," threatened the very existence of the oil industry by the imposition of a heavy cumulative tax on every step necessary in the production and marketing of petroleum. This bill was vigorously fought and was passed only after its exactions were greatly modified. It remains, however, one of the most conspicuous burdens imposed on any legitimate industry in the United States.

In Louisiana, the Anse Le Butte district is at present the only one that calls for special attention. It is near Breaux Bridge in St. Martin's parish. As long ago as 1900, Capt. A. F. Lucas, of Spindletop fame, drilled some shallow wells almost on the site of the gusher, showing indications of oil and gas. Other persons, especially Heywood Bros. and Robert Martin,

obtained extensive interests in the district and numerous wells have been drilled, some of which are small producers; the reported output in 1906 was 24,000 bbl.

The Lake Oil Company began operations in 1905, sinking seven wells. The No. 7 well of this company was brought in on Nov. 14, 1907, at a depth of 1850 ft., and while not properly cleaned out, its capacity was between 3000 and 4000 bbl. daily. The crude is a heavy fuel oil of 22.5 deg. B., very similar to that of Jennings, and the oil sand strata are said to be very thick. Other wells are going down and several strong corporations are in control so that the district will be promptly and efficiently developed. The district is favorably situated for oil shipments by rail; it already has a pipe line to Breaux Bridge, and the pipe line of the Evangeline Oil Company, between Jennings and the Atchafalaya river, passes less than half a mile distant from the new gusher.

The output of the Jennings field in 1907 varied from 276,000 bbl. in May to 495,000 bbl. in September. The proved territory was extended slightly to the southeast. The comparatively few wells are the largest average producers in Texas and Louisiana. The Caddo district produced some gas wells of immense pressure and volume, but the production of oil failed to fulfil expectations, although the output increased from 4650 bbl. in 1906 to about 50,000 in 1907. A portion of the gas is piped to Shreveport and other pipe lines are contemplated. The Welsh district was another disappointment; its output remains small. Many operators, however, hold the opinion that it is only a question of time when both Welsh and Caddo will be large producers. Wildcat wells were sunk in several Louisiana parishes, but nothing of any commerical importance resulted.

In Texas, wildcat operations, while not as numerous as in 1906, were conducted in many counties too numerous to specify, especially as they did not open up any extensive new field. In Duval county some small rail shipments were made from Piedras Pintas and in the San Antonio Mission field, 10 miles south of San Antonio, several small wells were successfully brought in. While operations were active in the old gusher fields, nearly all the new wells were in proved territory. Humble led in field work, followed in order by Batson, Sour Lake and Saratoga. The finding of a new sand at a depth of 1170 ft. in Humble field enabled it again to lead the Texas districts in production, although its output was 600,000 bbl. less than in 1906.

The outlook for an increased production in Texas remains practically as it was in December, 1906. There are certainly large undiscovered oil deposits in the coastal field and eventually they will be found, but when no one can foretell. Regarding prices for crude oil it is reasonable to believe that they will decline, when the enormous stocks and low

prices prevailing in the Mid-Continent region are taken into consideration.

PRODUCTION OF LOUISIANA AND TEXAS. (a)
(In barrels of 42 gal.)

TEXAS.				LOUISIANA.		
Year.	Production.	Value.	Avg. value per bbl.	Production.	Value.	Avg. value per bbl.
1896	1,450	$1,050	$0.720	.....	.....	.....
1897	65,975	37,662	.570	.....	.....	.....
1898	546,070	277,135	.508	.....	.....	.....
1899	669,013	473,443	.708	.....	.....	.....
1900	836,039	871,996	1.043	.....	.....	.....
1901	4,393,658	1,247,150	.284	.....	.....	.....
1902	18,083,658	3,998,097	.221	548,617	$188,985	$0.344
1903	17,955,572	7,517,479	.418	917,771	416,228	.453
1904	22,241,413	8,156,220	.367	(b)2,958,958	1,073,594	.363
1905	28,136,189	7,552,262	.268	8,910,416	1,601,325	.180
1906	12,567,897	6,565,578	.522	9,077,528	3,557,838	.392
1907	12,305,910	10,755,943	.874	4,613,663	4,060,023	.88
Totals	117,802,844	$47,454,015		27,026,953	$10,897,993	

(a) From *The Oil Investors' Journal.* (b) In addition 3,670,000 bbl. were produced but not sold until 1905 and 1906.

Petroleum in Foreign Countries.

Africa.—A syndicate was recently formed to prospect for petroleum in Cape Colony in the Aliwal North district.

Australia.—The discovery of petroleum in the Boonah district of Queensland is reported in the *Queenslander*. It asserts that crude petroleum, a heavy black oil, was found in a well 100 ft. deep. On a farm five miles from Boonah, bores are said to show oil at a depth of 130 ft. There are said to be other indications of oil in several parts of the district, notably near Harrisville. While payable oil is yet to be found, indications are said to be that the field will be productive.

Austria-Hungary.—A crisis has arisen in the petroleum industry in Galicia, brought about by over-production. The Austrian Credit Anstalt has indirectly granted, through the Petrolia company, advances on crude oil forwarded to the latter's reservoirs, and this has been a stimulus to the development of the oilfields. The apprehension that foreigners might obtain a further footing in the country has also assisted in promoting the production of crude oil. As a result there was in 1907 a large surplus of output over the demand, a decline of 62 per cent. in the price of crude oil, a filling of the storage reservoirs, and an unfavorable outlook for the immediate future. The Credit Anstalt has submitted an exhaustive report on the subject to the Austrian Minister of Finance.

PRODUCTION OF BORYSLAW-TUSTANOWICE FIELD.
(In metric tons.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1898	13,000	1900	55,000	1902	226,000	1904	546,000	1906	562,000
1899	18,000	1901	132,000	1903	373,000	1905	546,500	1907	(a)1,011,590

(a) Includes the Tustanowice field in 1907.

The other Galician fields, without exception, showed a decrease from their 1906 production.

PRODUCTION OF OTHER GALICIAN FIELDS.
(In metric tons.)

Field.	1904 Tons.	1905 Tons.	1906 Tons.	1907 Tons.
Potok	22,864	22,479	16,325	13,850
Rogi	47,531	24,234	11,452	9,033
Rowne	2,454	1,609	1,535	1,981
Tarnawa-Wielopole	10,707	32,956	24,870	17,390
Krosno	48,228	43,559	34,267	29,960
Other West Galician fields	34,411	35,670	30,883	25,290
Schodnica	72,627	60,201	47,151	39,650
Urycz	27,420	20,346	17,930	13,510
Mraznica	4,915	3,646	1,610	1,490
Other East Galician fields	9,909	10,600	12,220	12,230

Burma.—At present oil is brought down from the oilfields in iron barges on the Irawaddy river to the storage tanks at Rangoon. This method of transportation is expensive and several companies are preparing to lay pipe-lines from the oilfields to Rangoon, a distance of 300 miles. This project should create considerable business for American manufacturers of oil well supplies, as American engineers are largely in charge of the oil operations. The Burma Oil Company has a large refinery a short distance below Rangoon. It is enlarging its facilities and improving the quality of its oil, hoping to win a monopoly of the Indian trade. It commenced shipping in its own tank steamers in 1900 and has raised a fleet of vessels of a capacity of 500,000 gal. each. It has established ocean installations for bulk oil at Rangoon, Chittagong, Calcutta, Madras, Bombay, Marmagoa and Karachi, and has besides for the distribution of its oil a network of minor installations all over India. The exports of kerosene oil continue to increase rapidly, amounting to a total of 52,926,407 gal. of a value of $8,568,900 in 1906, as against 16,542,432 gal. valued at $2,654,038 in 1902. A high grade of waterwhite oil is now refined locally and is claimed to equal that from America. At present Burma exports a little fuel oil, although this is used in the refineries and tank steamers of the Burma Oil Company and on several of those of the Irawaddy Flotilla Company.

Canada.—The production of petroleum in Canada has been steadily increasing. The bounty of 52½c. per bbl. paid by the Government upon home-produced petroleum has done much to stimulate prospecting operations. In 1905 Canada's petroleum production amounted to 634,000 bbl., as compared with 552,500 bbl. in 1904. The year 1906 showed a production of about 750,000 bbl., while the production in 1907 was 788,872 bbl. Most of the petroleum production of Canada is derived from Ontario, but both oil and gas are known to occur in other Provinces.

New Brunswick has a small producing field at Memramcook, where several wells are regularly pumped, although the output is small. In Albert county there are large deposits of oil bearing shales, from which oil could be extracted by distillation, and these deposits under prudent and careful management may yet become a feature in the Canadian oil industry.

In Gaspe, Province of Quebec, prospecting for oil has been carried on for the last 10 years on a large scale, but the results so far have been disappointing. More than 50 holes have been bored, one of which reached a depth of 3700 ft.

Ontario is responsible for practically the total Canadian oil production. The Petrolia district has been yielding since 1862 and the Oil Spring pool is about contemporaneous. Since then other oil areas have been brought in, among which are the Bothwell, Moore, Leamington, Dutton, Thamesville, Wheatly and Tilbury; most of these are in the counties of Lambton, Kent and Essex. During the last two years the chief feature of the oil industry in western Ontario has been the discovery of the new field in East Tilbury township, where a pool which promises well has been struck in the vicinity of Bothwell. On Manitoulin island boring operations were actively carried on during 1906 by five different companies. This field, however, has not yet contributed to the Canadian production. In western Canada a great deal of energy is being displayed in prospecting for petroleum. Bore holes are being put down at Maniton, in southern Manitoba, and at Neepawa, on the Minnedosa branch of the Canadian Pacific railway. Deep boring operations are also carried on in Saskatchewan and in Alberta. There are at present 12 or 15 deep-drilling rigs prospecting various areas between the international boundary on the south and the lower part of Athabaska river on the north.

In southwestern Alberta oil has been struck in two wells at depth slightly exceeding 1000 ft. It is reported that these wells could be pumped and made to yield, but the lack of means of transportation is a drawback at present. At Calgary and at Medicine Hat drilling rigs are in operation; at the latter place provision has been made to reach a depth of 3000 ft. In the northern part of Alberta the search for petroleum is being carried on in the vicinity of Fort McMurray, on the Athabaska river, about 300 miles north of Edmonton.

In British Columbia two companies were working in southeast Kootenay in 1906. The depths reached in this district are not yet sufficient to warrant definite conclusions. Some work was also being carried on in the Cariboo district, where promising indications have been reported.

Dutch East Indies.—According to *Jarrbock van het Mijnwesen*, there were 26 petroleum concessions in force in Java and Madoera during the year ending with June, 1906. The companies having these concessions

produced 1600 million liters of crude petroleum. The refinery at Semarang yielded 41,085,847 liters of illuminating oil, 1,000,000 liters of gasoline, and 428 kg. of paraffin. The Wonokromo refinery treated 50,229,000 liters of crude oil. The refinery at Balik Pappan was recently enlarged.

Italy.—A brief review of the petroleum industry in Italy was given by *Chem. Zeit.* (July 27, 1907). There are three producing districts, viz., the Emilia in the provinces of Parma and Piacenza, the Abruzzi district, and in Sicily. The Emilia district is the only one of the three in which exploration has been carried out in a modern and rational manner. The first well was bored in 1860. A refinery was established in 1880 in Borgo St. Dounino. Later two other refineries were built in Fiorenzuola and in Mailano. The greatest part of the Italian petroleum comes from Montechino and Velleia, in the province of Piacenza. A French company was organized in 1903 with a capital of 1,200,000 fr. and this concern was the first to apply modern and systematic methods to the exploration of Montechino. As a result of its efforts the yearly output increased from 800 in 1903 to 2000 tons in 1905. This company in 1905 had 28 wells in operation. The region about Velleia is exploited by the Sociètè Française du Petrole, which in 1905 had an output of 1870 metric tons. This company operates a refinery at Montechino. In 1906 the Società Petrolia d'Italia was formed in Genoa. It purchased both of the above companies, and set itself energetically to the task of exploring all the petroleum deposits of the country. In 1906 the output of naphtha amounted to 20,000 metric tons.

Japan.—The sale by the Standard Oil Company of its oil property in Japan indicates that it does not consider the oil deposits sufficiently large to make development work on a large scale a profitable undertaking. According to the last official statistics the production of petroleum in 1905 amounted to 1,332,000 bbl. In the same year the imports amounted to 1,397,600 bbl., more than half of which came from the United States. The large contracts recently made by Japan with the California oil companies show that the supply of petroleum in Japan is not keeping pace with the increasing demand.

Korea.—According to a recent report of the United States Consul-General at Yokohama, a charter has been granted to several Japanese residents in Seoul to work a petroleum deposit in North Phyong-an Province. This is said to be the only petroleum yet discovered in Korea.

Mexico.—Petroliferous lands extend from the hacienda of San Jose de las Ruinas, in central Tamaulipas, to the district of Valles, in San Luis Potosi (where the Cebano oil deposits are being worked), through the counties of Uzuluama, Tuxpan and Papantla, in Vera Cruz. Further to the south another region is found which embraces the Vera Cruz counties of Acayucan and Minatitlan, and extends southward through the States

of Tabasco, Campeche and Chiapas. Petroleum has also been found in small quantities in the Federal District, in Jalisco, and in Oaxaca. Bituminous asphalt also is common in parts of the States of Vera Cruz and San Luis Potosi. The Ebano oil is being used as a fuel for locomotives by the Mexican Central railway. The oil fields in northern Vera Cruz are not yet operated commercially, but are being actively developed. In southern Vera Cruz, especially at Minatitlan, oil is being stored preparatory to the completion of large refining works.

According to a recent consular report, the Mexican Central railway is now taking 4000 bbl. of fuel oil daily from the Mexican Petroleum Company, at $1.10 per bbl. All new locomotives purchased by the Mexican Central are equipped for burning oil, and those in use are being constantly remodeled to burn oil.

The regulations which Mexico imposes with reference to prospecting for oil and gas on Government lands are substantially as follows: The Federal Executive is authorized to grant permission for exploration, which permission may be issued to private persons or duly organized companies and shall be valid for only one year from date of publication in the *Diario Oficial* and such permission cannot be extended. During the term of said permission no one except the grantee named shall have the right to explore within the zone described. Permissions for such explorations incur a tax of five centavos per hectare for which stamps must be attached to the document. Those who may discover petroleum or natural gas shall immediately give notice to the Department of Fomento in order that the secretary thereof may issue a patent by virtue of which the patentee may exploit the wells or deposits discovered; for the issuance of such patents the following conditions must be complied with: The Secretary of Fomento shall designate an expert to examine the discovery. The wells discovered must have a production of at least 2000 liters of petroleum daily, or 20,000 liters of natural gas of good quality and suitability for fuel. Compliance with this law must be guaranteed by a deposit of the bonds of the Public Debt, the amount of which is fixed by law. Patents for exploitation shall be effective for 10 years from date of publication in the *Diario Oficial*. The holders of patents under this law have the following exemptions for the exploitation thereof: To exploit free from all taxes the natural product and the refined or manufactured materials produced; to import free from duty, once only, the machinery for refining petroleum or natural gas or manufacturing all products which have crude petroleum for their base, the piping necessary for this industry, together with the accessories for said piping, pumps, wood and iron tanks, gas meters and materials for buildings for such exploitation.

Each individual or company who receives a patent shall pay into the

Federal treasury at once, and annually thereafter, $2400 for the payment of an inspector of oils and prospecting methods, appointed by the Government. In addition to this tax, 10 per cent. of all dividends must be paid to the Government. Added to the privileges and requirements set forth above are others which are almost equally important, but those given are sufficient to furnish a comprehensive idea of Government regulation of the petroleum industry in Mexico.

(By C. A. Bohn.) Recent developments point to a warm fight between the various oil interests in Mexico. The business has long been in the control of the Waters-Pierce Oil Company, a branch of the Standard Oil Company. Of late years, however, an enormous amount of development work has been carried on in San Luis Potosi, Tamaulipas and Vera Cruz, and late in 1907 it was stated that a combination had been effected between the principal operators in those States. Recent developments have been on the Isthmus of Tehuantepec, along the line of the Tehuantepec National railroad, by S. Pearson & Son, contractors and builders of that railroad. The wells are many and strong and the Tehuantepec National railroad is using the oil therefrom as its fuel. At Minatitlan, S. Pearson & Son have been engaged in the erection of the largest refinery in the Republic. From Minatitlan, S. Pearson & Son have a railroad connecting with the Tehuantepec National at Kilometer 30 and thence to the wharves at the port of Coatzacoalcos. The company therefore has ready access to Central American and South American trade, as well as the Gulf ports of the United States, having its own tank steamers. It is not known whether a three-cornered fight may be expected or whether the San Luis Potosi companies will join forces with S. Pearson & Son. Mexico can derive nothing but good from the contest and the resulting reduction of cost of oil. The Pearsons will undoubtedly give a great deal of their attention to export trade, for the Minatitlan refinery is to have a capacity of 5000 bbl. a day while the demand of the Mexican market is at present only about 1000 bbl. a day.

Peru.—Details concerning the exploitation of Peruvian petroleum were given in the *Petroleum Review* (Oct. 26, 1907). The oil region is situated on the narrow strip of land between the Andes mountains and the Pacific ocean. The wells vary in depth from 180 to 1760 ft. The oil is remarkably heavy. F. C. Piaggio & Co. have been in the business for 35 years and are the largest concern in Peru producing petroleum; two other large producers are the London Pacific and the Peruvian Corporation. The Piaggio company is reported to have 100 wells producing 150 bbl. daily; the oil compares favorably with the California product. It has an asphalt base and about 30 per cent. is refined into more volatile oils. The refinery at Zorritos is equipped with stills, storage facilities, cooper shop and canning plant.

Portugal.—Hopes are entertained that the recent discovery of petroleum in the Province of Angola may prove to be of importance. Prospecting has been carried on in the district of Daude, and special Government concessions have been granted.

Rumania.—The production of crude petroleum in 1906, according to the *Moniteur du Petrole Roumaine,* was 887,091 metric tons, against 614,870 in 1905.

PRODUCTION OF PETROLEUM IN RUMANIA.

Year.	Metric Tons.	Year.	Metric Tons.	Year.	Metric Tons.	Year.	Metric Tons.
1897	110,000	1900	250,000	1903	384,302	1906	887,094
1898	180,000	1901	270,000	1904	497,000	1907	(a)1,129,097
1899	250,000	1902	310,000	1905	614,870		

(a) Reported by the *Petroleum Review.*

The advance in the refineries was proportionate to the production of crude oil, as shown by the following figures (the leading figures being for metric tons produced in 1906, while those in parentheses are for 1905): Benzine, 114,428 (78,128); illuminating, 221,683 (153,499); oils, 53,588 (17,255); residues, 333,714 (237,677). The home consumption was—benzine, 566 (615); illuminating, 35,243 (31,558); oils, 5,350 (4,921); residues, 237,477 (162,243).

Russia.—The production of the Baku field in 1907 is estimated at 474,900,000 poods, as compared with 446,100,000 poods in 1906, 400,000,000 poods in 1905 and 615,300,000 poods in 1904. An Associated Press despatch of Dec. 6, 1907, stated that a new gusher has been opened at Surakhanti, 10 miles from Baku. This indicates an important extension of the Baku oil industry. According to geologists this discovery opens up a possibility of oil strata in a region many times larger than the present one. The new well is 1589 ft. deep and is said to yield 10,000 bbl. per day. The oil is heavier than ordinary Surakhany crude, its specific gravity being about 0.820; it yields more than 60 per cent. of illuminating oil, but little benzine or solar oil, and about 20 per cent. residuals.

Wilbur T. Gracey, United States consul at Tsingtau, China, reports that large petroleum wells have been discovered on the island of Sakhalin, and also that a lake of petroleum has been found in the same district. The lake and wells have been discovered by Russian engineers, and have been reported to the Russian mining department. It is reported that the department will place foreigners on the same footing as Russians in the development of the discoveries. The wells and lake are situated on the northeast coast of the island of Sakhalin, near Nabil Bay, in the Russian section of the island. The place is said to be easily accessible to steamers.

According to a British Consular report, pumping was to have begun at the Cluinion oil field in Turkestan last September, but labor troubles interfered with the work. Reservoirs to hold 5000 tons had been built at Vannooskaya station, and the Orenburg-Tashkent railway had given an order for 1,000,000 poods at 25 kopecks per pood. Another field is said to have been discovered at Gulkhan. At Chongclek, on Kertch strait, the existence of petroleum has long been known, and renewed efforts to create an industry there are now being made. On the river Ukhta, a tributary of the Izlema, which runs into the Pechora, there was a small naphtha refinery in the early part of the 18th century and the product was sent to Moscow. An attempt was made to start work again 40 years ago, and quite recently borings resulted in a fountain at a depth of 45 fathoms. The oil is said to be better in quality and more abundant than that of the Caucasus. The region is difficult of access, and the field can not be worked unless the Kotlas railway is extended to the works.

(By I. I. Rogovin.) The output of petroleum in Russia in 1907 showed only a slight increase as compared with 1906, and there was little recovery from the setbacks received by the industry in the latter year. It may be said decidedly that no branch of Russian industry has suffered more from labor strikes of recent years than the Baku oil industry. The year 1905 was a critical one and led to a decrease of output, increase of price, loss of market, etc. The accompanying table shows the output of oil during the last seven years.

OUTPUT OF OIL AT BAKU AND GROSNY FROM 1901 TO 1907.
(In millions of poods.)

	1901	1902	1903	1904	1905	1906	1907
Baku	671	636	600	615	410	448	475
Grosny	35	34	33	40	43	38	40
Total	706	670	633	655	453	486	515

The labor strikes, especially those of sailors of merchant ships, had an important bearing on industrial conditions at Baku. In order to realize their influence on the oil industry, it may be mentioned that according to reliable statistics, the time lost during the first nine months of 1907 amounted to 30.678 days per well. Of this, 20.627 was for productive wells, and 10.051 for new wells. Comparing these figures to the yield of oil, the Statistical Bureau states that the actual output during nine months of 1907 was 5.9 per cent. below the possible production. In connection with the sale of oil residues and petroleum at home and abroad, it is an important fact that the whole amount of oil produced in 1907 was not worked into petroleum, a considerable quantity of crude oil having been mixed with the residues and exported for heating purposes. The

accompanying table shows the amounts of residues and crude oil shipped by the Volga river from the Baku region to the distilleries of central Russia.

SHIPMENTS FROM BAKU TO CENTRAL RUSSIA.
(In millions of poods.)

	1903	1904	1905	1906	1907
Residues	301	300	259	205	250
Crude Oil	26	18	29	34	36

The comparatively great quantity of oil residues exported, however, did not supply the demand, as almost all stores of oil in the interior of Russia were exhausted. Therefore the prices for these products were very high. At the beginning of 1907, before the opening of navigation, the price of residues at Baku was 26 to 27 kopeks per pood; with the opening of the navigation it rose and in August attained 31 to 32 kopeks per pood, falling to 27 kopeks toward the end of the navigation season.

The petroleum trade showed a considerable increase in the home market. The supply in 1907 was 62.6 million poods, against 39.2 million poods in 1906, thus showing an increase of 22.6 million poods, or 60 per cent. The normal consumption of petroleum in Russia amounts to about 55 million poods; about 8 million poods are supplied by the factories of central Russia and about 45 come from the Caucasus.

EXPORTS OF RUSSIAN PETROLEUM.
(In millions of poods.)

Year.	From Baku	From Novorossieps	Total
1904	64.4	21.8	86.2
1905	27.7	9.9	37.6
1906	19.1	2.4	21.5
1907	23.0	2.5	25.5

The decrease of exports to China, Indo-China, India and the Far East is due to the development of the oil industry on the Sunde Islands; exports to Austria, Italy and the Danube countries, decreased on account of the development of the industry in Rumania and Galicia. In Great Britain and France the chief supply comes from America, as is shown by the following figures of oil imported into England: In 1904, from Russia 49.6 per cent., from America 47.1 per cent., from other countries 3.3 per cent.; in 1907, from Russia 19.9 per cent., from America 73.2 per cent., from other countries 6.9 per cent. Even in Germany, Russia's nearest neighbor, as well as in Austria, American oil has the preference. Only in Turkey, Greece, Arabia and Egypt did the export of Russian oil show an increase in 1907; however, the total amount of oil exported to these four countries does not exceed 12 million poods. Such a marked decrease in the exports of Russian oil does not promise well for the industry in Russia.

PHOSPHATE ROCK.

The production of phosphate rock in the United States in 1907 showed an increase of approximately 8 per cent. as compared with that of 1906. This increase was effected despite the shortage of cars early in the year, the inclement weather in Tennessee and South Carolina during the winter of 1906-07 (which curtailed mining operations somewhat), the unfavorable labor conditions following close upon the above difficulties and the financial depression which set in late in the year. Florida continued in the lead as a producing State, Tennessee and South Carolina following in the order named.

IMPORTS OF FERTILIZERS INTO THE UNITED STATES.

(In tons of 2240 lb.)

	1904		1905		1906		1907	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Guano	35,876	$478,388	25,651	$365,823	22,947	$320,565	29,141	$365,257
Crude phosphates	130,214	745,744	56,421	275,889	23,281	147,547	25,876	163,944
All other fertilizers	....	2,856,141	....	4,048,403	....	4,231,723	....	4,994,346
Of which— Kieserite and Kainite	218,957	1,050,082	....	....	....	....	....	....

STATISTICS OF PHOSPHATES IN THE UNITED STATES. (a)

(In tons of 2240 lb.)

Year.	Production	Imports.	Exports. (b)	Consumption.	Year.	Production	Imports.	Exports.	Consumption.
1898	1,257,645	71,388	570,948	758,085	1903	1,581,576	153,972	785,259	950,289
1899	1,663,476	118,613	867,790	914,299	1904	1,874,428	166,090	842,484	1,198,034
1900	1,527,711	144,006	619,995	1,051,722	1905	1,933,286	82,072	934,940	1,080,418
1901	1,483,723	180,714	729,539	934,898	1906	2,052,742	46,228	904,214	1,194,756
1902	1,600,813	145,793	802,086	944,250	1907	2,251,459	55,017	1,018,212	1,288,264

(a) Production statistics of 1901 and subsequent years, except 1905-1907, are those of the U. S. Geological Survey and are based on marketed products. (b) Neglecting the insignificant re-exports of foreign product.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Phosphate.	1904.		1905. (b)		1906. (b)		1907. (c)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Florida hard rock..	531,087	$2,672,184	579,228	$3,011,986	561,370	$3,312,083	589,217	$3,714,767
Florida land pebble	460,834	1,102 993	401,997	803,994	603 382	1,810,146	721,028	2,523,598
Florida river pebble	81,030	199,127	90,065	216,156	41,742	116,878	36,729	139,570
Total Florida....	1,072,951	$3,974,304	1,071,290	$4,032,136	1,206,494	$5,239,107	1,346,974	$6,377,935
S. Car. land rock...	258,806	$830,117	221,712	$731,650	270,000	$990,000	228,354	$890,581
S. Car. river rock...	12,000	31,200	30,284	87,824	45,000	144,000	37,303	126,830
Total S. Carolina ..	270,806	$861,317	251,996	$819,474	315,000	$1,134,000	265,657	$1,017,411
Tennessee.........	530,571	$2,037,804	610,000	$2,074,000	520,381	$2,029,486	626,683	$3,008,078
Other States........	100	200	......	....	10,867	61 942	12,145	47,098
Total United States	1,874,428	$6,873,625	1,933,286	$6,925,610	2,052,742	$8,464,535	2,251,459	$10,450,522

(a) Statistics for 1904, are those of the U. S. Geological Survey. (b) Statistics of 1905 and 1906 are those of J. M. Lang & Co., Savannah, Ga., with respect to quantity, and are based upon shipments. (c) As compiled by *The Mineral Industry*. the tonnage figures of Florida being supplied by J. M. Lang. & Co.

PRODUCTION OF PHOSPHATE ROCK IN THE WORLD.
(In metric tons.)

	1901	1902	1903	1904	1905	1906	1907(d)
Algeria.............	269,878	260,859	301,112	344,969	334,784	333,531	315,000
Dutch W. Indies.....	(c)	10,530	15,511	22,764	22,940	26,138	....
Belgium.............	(a)222,520	(b)135 850	(b)184,120	(b)202,480	193,305	152,140	180,000
Canada..............	937	776	1,205	832	1,180	....	(c)100,000
Christmas Island....	42,125	61,179	70,096	71,757	97,052	90,561	290,000
France..............	535,676	543,900	475,783	423,521	476,720	469,408	375,000
Norway..............	738	2,295	1,795	1,456	2,522	3,482	....
Russia..............	21,276	13,709	14,635	20,282	(c)	(c)	(c)
Spain...............	4,220	1,150	1,124	3,305	1,370	1,300	(c)
Sweden..............	(c)	3,895	3,219	2,929	(c)	....	....
Tunis...............	178,018	263,482	332,888	455,789	559,645	796,000	1.040,300
United States.......	1,507,548	1,514,159	1,606,881	1,904,419	2,135,449	2,085,586	2,251,459

(a) Cubic meters. (b) Metric tons of phosphate of lime; in addition there were 315,200 cu.m. of phosphatic chalk in 1902, 350,250 cu.m. in 1903, 311,640 cu.m. in 1904, 80,380 cu.m. in 1905 and 119,450 cu.m. in 1906. (c) Statistics not available. (d) Estimated by *l'Engrais*, Liege, Belgium, except for United States. (e) Includes all other countries.

Market and Prices.—The exceptionally heavy demand early in 1907 for phosphate rock sustained the high level of prices prevailing during the latter part of 1906. The financial disturbances late in 1907 were not to any great extent influential in weakening the market. The accompany-

PRICES OF PHOSPHATE ROCK IN 1907.

	January.	December.
Phosphate rock, ground, f.o.b. Charleston, 2000 lb..............................	$6.75 @ 7.00	$6.75 @ 7.00
South Carolina phosphate rock, undried, per 2400 lb., f.o.b. Ashley River.........	6.00 @ 6.50	5.75 @ 6.00
South Carolina phosphate rock, hot air dried, f.o.b. Ashley River................	7.00 @ 7.50	7.00 @ 7.25
Florida land pebble phosphate rock, f.o.b. Port Tampa, Fla......................	5.75 @ 6.00	7.50 @ 7.75
Florida high-grade phosphate hard rock, f.o.b. Florida or Georgia ports...........	10.25 @10.50	10.25 @ 10.50
Tennessee phosphate rock, f.o.b. Mt. Pleasant, domestic, per ton, 78 @ 80 per cent..	6.00 @ 6.25	6.50 @ 6.75
75 per cent. guaranteed....................	5.50 @ 6.00	6.00 @ 6.25
68 @ 72 per cent...........................	4.00 @ 4.50	4.00 @ 4.25

ing table shows the prices ruling for phosphate rock in January and at the end of the year.

Exports and Imports.—The exports of crude phosphate rock from the United States during 1907 were 1,018,212 long tons, valued at $8,387,176, as compared with 904,214 tons, valued at $7,373,945, in 1906. In addition to this there were exported 45,000 tons of guano, dried blood, bones, etc., classified as "all others," valued at $1,641,316, as compared with 31,999 tons, valued at $1,088,004, in 1906.

The imports during 1907 were as follows, the figures in parentheses being for 1906: Guano, 29,141 tons, valued at $365,257 (22,947 tons, $320,565); crude phosphate 25,876 tons, valued at $163,944 (23,281 tons, $147,547); all others, valued at $4,994,346 ($4,231,723).

Review by States.

Arkansas[1].—The developed phosphate deposits of Arkansas are on Lafferty creek, on the western edge of Independence county. The only point at which the beds are now worked is about three-quarters of a mile east of White river, and the same distance from the White river branch of the Missouri Pacific Railway. The undeveloped portion of these deposits reaches from the town of Hickory Valley, 10 miles northeast of Batesville, westward as far as the town of St. Joe, Searcy county, a distance of more than 80 miles.

The developed deposits are confined to Secs. 14 and 15, T. 14 N., R. 8 W., situated on Lafferty creek, near the junction of East Lafferty and West Lafferty creeks, about four miles southwest of the town of Cushman, in Independence county. In June, 1900, the Arkansas Phosphate Company was organized for the purpose of developing these deposits. After several months of prospecting, a mining and milling plant was erected; this plant was destroyed by fire after only a few months of active operation. A much larger plant is now nearing completion at Little Rock. The new plant will have an annual capacity of 40,000 tons, with a shipping capacity of 15 to 18 cars per day. Up to October, 1906, the company had mined a total of about 10,000 tons of crude phosphate. The name of the company was recently changed to the Arkansas Fertilizer Company.

The deposits first developed are in a small ravine which enters East Lafferty creek from the east. The bed here worked was about 22 in. thick. The discovery of heavier deposits on Lafferty creek, about a mile southwest of the above described workings, led to their abandonment and the opening of mines at the new place. Here the phosphate occurs in two beds, the upper from 4½ to 6 ft. in thickness, and the lower about 4 ft. thick; only the upper bed is worked, the lower being of too low grade. Quarrying methods are employed in taking out the rock.

[1] Abstracts from "Developed Phosphate Deposits of Northern Arkansas," in *American Fertilizer*, December, 1907.

Florida.—The year 1907 was a prosperous one for the phosphate industry in this State. The production of phosphate rock of all kinds showed a substantial increase over 1906. Approximately 714,702 tons of land pebble, 591,719 tons of hard rock, and 36,729 tons of river pebble were mined. According to the *American Fertilizer*, 122,511 tons of land pebble were shipped to Continental ports, 82,540 tons to Mediterranean ports, 38,613 tons to Baltic ports, and 53,254 tons to United Kingdom ports; the remainder was distributed to domestic ports. Of the output of hard rock, 76,294 tons were shipped to United Kingdom ports, 126,449 tons to Baltic ports, 338,439 tons to Continental ports, and 45,054 tons to Mediterranean ports; this left only a little more than 5000 tons for shipment to domestic ports. The pebble rock phosphate was consumed entirely in this country. During 1907 the land pebble region in south Florida was again the scene of great activity. Several extensive mining plants were built and beds of pebble previously discovered were further developed.

In an address before the Florida Bankers' Association, at St. Petersburg, Fla., Feb. 21, 1908, C. G. Memminger, of Lakeland, stated that Florida produced during 1907 a total of 1,343,115 tons of phosphate of all classes, with a value of about $6,715,575 at the mines. Florida phosphates are classed under three heads: Hard rock, land pebble and river pebble. The hard-rock deposits are included within an area approximately 100 miles long and from 8 to 10 miles wide, beginning at Fort White and extending in a southerly direction to Bay City. Florida hard rock is sold under a guarantee of a minimum of 77 per cent. bone phosphate of lime and a maximum of 3 per cent. oxides of iron and alumina and 3 per cent. moisture. The entire output of hard-rock phosphate is exported, the amount shipped during 1907 being 591,719 tons.

Land-pebble phosphate occurs in Polk, DeSoto and Hillsborough counties. This variety represents the medium-grade phosphate which is well adapted chemically to the manufacture of fertilizers. Approximately 60 per cent. of this product is consumed in the United States, the remainder being exported. Land pebble is sold under a guarantee of 68 per cent. bone phosphate of lime and a maximum of 4 per cent. oxides of iron and alumina and 3 per cent. moisture. The production of this class of phosphate in 1907 was 714,702 tons.

River-pebble phosphate occurs in DeSoto county, on Peace river, and is commonly known as Peace river pebble. There has been a gradual decrease in the output of this class of phosphate, only 36,729 tons having been produced in 1907. River pebble is sold under guarantee of 60 per cent. bone phosphate of lime, 3 per cent. oxides of iron and alumina, and 3 per cent. moisture.

Florida phosphate mining, especially in hard rock, has passed through

many vicissitudes, but the industry today is on a sound, conservative basis and in comparatively few hands. In the hard-rock district four companies produced the bulk of the rock; in the pebble district eight companies were operating on Jan. 1; in the Peace river district only one company operates.

South Carolina.—This State produced 265,657 long tons of phosphate rock, valued at $1,017,411 in 1907, as compared with 315,000 tons, valued at $1,134,000, in 1906. As in previous years, land rock made up the bulk of the total, river rock constituting but a small percentage of the output.

The following companies mined land rock during 1907: Charleston Mining and Manufacturing Company, Bolton Mines Company, Runnymede Phosphate Company, and the Bulow Mining Company; all of these have their offices at Charleston. The first named company is the largest operator. The Stono Mining Company, of Charleston, and the Central Phosphate Company, of Beaufort, are river-rock miners.

The State Phosphate Commisson of South Carolina met at Columbia on April 21, 1908, at which time the regular inspection was made of the land and river dredges on the coast. During 1907 a total of 37,303 tons of rock was mined, showing an increase of 6121 tons, while the shipments amounted to 36,022 tons, which was an increase of 1544. Royalties amounted to $9,005.50.

The deposits[1] in the South Carolina field belong to the class of amorphous nodular phosphates, and were the first rock-phosphate deposits in America to be worked. Operations began there with the organization of the Charleston Mining and Manufacturing Company, in 1867, and have continued ever since. Nodular phosphates occur in practically all limestone regions, especially the geologically younger ones. Deposits are found at intervals all along the Atlantic Coast, from North Carolina southward; but the only beds rich enough to work are those of South Carolina, which lie along the coast, practically parallel to the shore line for a distance of about 70 miles, between the Wando river on the north and Broad river on the south, at a distance of from 10 to 30 miles from the ocean. The rock does not lie in continuous beds, but occurs in basins of from a few acres to several square miles in extent.

The phosphate occurs as nodules lying, more or less closely packed, in a matrix of varying hardness and composition. The nodules vary in size from that of a pea to pieces weighing up to 2000 lb. The specific gravity is about 2.4 and the hardness 3.5 to 4. The beds are generally level, and do not follow the contour of the surface. Mr. Chazal, following

---

[1] Abstracts from a paper on "Phosphate Deposits in the Southern States," by Lucius P. Brown, Nashville, Tenn., in *Proceedings of the Engineering Association of the South*, Vol. XV, 1904.

Dr. C. U. Shepard, Jr., gives the following as a characteristic section of a bed of land rock: (1) Soil and subsoil—few inches to a foot. (2) Light-colored silicious clay, containing much fine sand, with small mica scales, and little calcareous matter—one foot or more. (3) (Wanting in the more superficial beds.) A blue, argillaceous marl, probably altered marsh mud, containing fragments of recent shells—about 2 ft. (4) Coarse sand—1 to 3 in. (5) The phosphate nodules, lying in either a loose silicious or a tenacious bluish or buff argillaceous marl, frequently accompanied by abundant fossil bones and teeth. The lower nodules are often softer than the upper, and at some localities show a gradual transition, by loss of cohesion and decrease of phosphatic contact, into: (6) A marl highly phosphatic in its upper portion, but at a depth of a few inches containing only 10 to 20 per cent. of phosphate. (7) Argillaceous or arenaceous marl, containing 7 to 10 per cent. of phosphate.

The South Carolina phosphates are derived from the concentration of material already existing in the Eocene limestones, locally called marls. The phosphate layer runs from a few inches to 2½ or 3 ft. in thickness. The workable beds average 8 or 9 in. and yield from 300 to 1200 tons of clean phosphate per acre, according to the closeness with which the nodules are packed in the matrix; the average yield is from 700 to 800 tons.

A distinction is made between land and river rock, but the two are the same in origin, the river rock being simply land rock washed out from the banks and deposited in depressions in the beds of the streams. This rock is now practically exhausted.

Of the land rock there still remains a great deal. This is prospected by rodding and pitting. In rodding, the rod (usually a selected piece of ¾- or ⅞-in. octagon steel about 15 ft. long) is put down every 50 ft. through or to the rock layer. This determines the presence and something as to the amount of the rock, and pits are then sunk at such intervals as seem proper to the engineer in charge of prospecting; these determine the quality of the rock and confirm the indications of the rod. The boundaries of a phosphate field having been determined, a main line of railroad, starting at the river front or on the bank of some convenient stream, is established across the field, and from it laterals are run off at right angles at any convenient distance, say 500 ft. Between and parallel to these laterals, ditches are dug to 4 or 5 ft. below the phosphate stratum; these discharge into a main ditch, at the lowest point of which is placed a pump for draining the field. From the lateral ditches the miners start their work, at right angles to the lateral railways, commencing at one end of the field and digging trenches 15 ft. wide and 500 ft. long. The overburden and rock are thrown to different sides of the trench, and the latter is carried in wheelbarrows to the railway cars. These are then run down to the mill, which usually ocnsists of a crusher and log washer; the rock is dried on

kilns of cord wood. Occasionally cylinder dryers are also used. After drying, the rock is loaded into lighters, or boats, for shipment.

The river rock is found at depths from tide level to 10 or 15 ft., in the same sort of stratum as, though usually thicker than, the land rock, and occasionally under layers of sand and clay. A limited amount was obtained by hand labor at low tide, and by divers, but by far the greater quantity was mined by heavy dredges especially made for the work. The rock went from the dippers to the washers, thence to lighters, to be carried to the docks for drying.

Tennessee (By H. D. Ruhm).—The year 1907 was a banner year in the phosphate field of Tennessee so far as development of new mines, installation of modern machinery at the old mines, and high price of rock were concerned. About 300,000 tons of the output in 1907 was shipped from the Mt. Pleasant field proper and the remainder from the Centerville field, the Franklin field and the Pulaski field, with scattering shipments from many small stations along the main line of the Louisville & Nashville Railroad from Pulaski to Gallatin. Although it seemed positive that the high-water mark had been reached as to prices in November, 1906, when 75 per cent. rock sold at $5 per ton, the price climbed steadily to $6 during the winter and spring of 1907, and then instead of going down during the summer months as is usually the case, it went higher until $6.25, $6.50 and even as high as $6.75 were obtained for 75 per cent. domestic and as high as $8 for 78 per cent. for export.

Notwithstanding the high prices, labor was so greatly in demand all over the country, that it was impossible for the miners to supply their shipping requirements, until the acute car shortage put in its appearance in the latter part of August and early September. This caused considerable stock to accumulate, but kept prices well up to the mark until the financial catastrophe of November caused buyers to cancel contracts whenever possible and to stop shipment temporarily when they were unable to cancel.

During the last weeks of the year we were presented with the anomalous condition of an ample car supply, plenty of labor, fairly good weather and plenty of rock to ship, but very few places to ship it to. There was no diminution in price, except on the part of a few small miners selling at lower prices to those having contracts to ship on, and as most buyers stopped shipment on account of inability to get money to pay for rock and even to pay freight, it does not seem logical to expect any decrease in price. In fact, as all the miners practically stopped digging rock, and confined their efforts exclusively to preparing and shipping out the rock already mined, thus rapidly depleting stocks on hand, instead of increasing them as would ordinarily be the case this time of the year, it seems probable that whenever financial conditions justify a resumption of business and

rock is once more in demand, prices will ascend even above the present high-water mark.

In 1906 several predictions were made as to probable causes that might bring about a drop in prices and I made use of the following language: "The independent factories are likely to get together and purchase their own supply of rock," and, "The demand may temporarily lull in the face of a greatly increased production; a temporary financial flurry may cause a decrease in the demand," etc.

The first of these predictions came true when 27 independent factories were brought together through the efforts of T. C. Meadows, of Buffalo, N. Y. The Independent Phosphate Company was organized and purchased from J. H. Carpenter and his associates their brown-rock holdings at Solita, Estes Bend and Satterfield, and their blue-rock holdings at Leatherwood, paying $1,000,000, the price including the construction of railroad lines to Estes Bend and Leatherwood. The company started operations at Solita and Satterfield at once, but was badly handicapped in many particulars, so that up to Sept. 1 only a small part of the requirements had been produced, and the companies composing the combination had in many cases to pay some rather extravagant prices for rock. However, since Sept. 1, with a change of management and other changes, they produced nearly up to their requirements.

This company adopted the policy pursued by the Federal Chemical Company for the last few years, and installed mechanical driers with coal as fuel, instead of the time-honored custom of burning rock on cord wood.

Following this lead several other new drying plants were constructed. At the end of the year the following were in operation, ready to operate, or in process of erection: Federal Chemical Company, four cylinders at Century, two cylinders at Mt. Pleasant; Independent Phosphate Company, two cylinders at Satterfield, two at Solita mines (Mt. Pleasant) and two under construction at Estes Bend; Mt. Pleasant Dryer Company, one cylinder; Richland Phosphate Company, one cylinder; Blue Grass Phosphate Company, four cylinders; Jackson Phosphate Company, two cylinders; Middle Tennessee Phosphate Company, one cylinder; Charleston (S. C.) Mining Company, two cylinders, all at Mt. Pleasant; Granbery Jackson, one cylinder at Satterfield; Paragon Phosphate Company, one cylinder at Twomly; Ward Mining Company, one cylinder at Swan Creek; Armour & Co., one cylinder at Swan Creek; and the American Phosphate Company, one cylinder at Wales, near Pulaski.

Thus the year 1907 closed with 27 drying cylinders with an output capacity of 2700 tons per day of 24 hours. The year 1906 closed with only four such cylinders in operation.

Other Fields.—A new industry has been opened up in the West through

the discovery of important phosphate deposits over a considerable area in southeastern Idaho, southwestern Wyoming and northeastern Utah. The future of the industry depends largely on the granting of such rates by the railroads as will enable the manufactured product or raw material to be sold at a profit in Australia, Honolulu, Japan, and the Middle States, the whole market on the Pacific Coast not being at present extensive enough to warrant general development. The only production of any moment from these fields during 1907 was made by the San Francisco Chemical Company, from its property at Montpelier, Bear Lake county, Idaho. This company, the home office of which is in San Francisco, shipped several thousand tons of 70 per cent. bone phosphate during the year. Some trouble was experienced in obtaining satisfactory freight rates.

A 250-ft. incline shaft has been sunk in the black shaly limestone strata of the phosphate formation. This formation is exposed in Montpelier cañon, three miles east of the town for a distance of one mile south and two miles north of Montpelier creek. The strata composing the mountain just south of the cañon occupy an inverted position with the geologically upper rich bed of phosphate lying at the bottom, the mountain representing an immense reverse fold. The horizon is without much doubt high in the Upper Carboniferous, probably not Permian, though not far below it. The phosphate generally can be classed as a black oolitic phosphate, in appearance like the black phosphate of Tennessee, and averaging 70 per cent. bone phosphate. This same bed has been uncovered and seems to hold its character and relative position at Hot Springs on Bear lake, 15 miles south, at Thomas Fork, 15 miles east, and at Cokeville, 20 miles southeast of the Montpelier deposit. Phosphate deposits have been uncovered at several other points in this district.

Among other districts where deposits similar to those described above have been uncovered may be mentioned the vicinity of Woodruff creek, in Rich county, Utah; to the south of Logan, Utah; at Corydon station and north of Fort Douglas, just east of Salt Lake City, Utah; in the Sublette range of mountains of Wyoming, just east of the Idaho line; on the Thomas fork of Bear river; and one mile east of Cokeville, Wyo.

Phosphate Mining in Foreign Countries.

Algeria.—The production of phosphate rock in 1907 showed a slight increase, being estimated at 315,000 metric tons, as compared with 302,262 tons in 1906. The phosphate deposits, which are of immense extent, are generally found at the junction between the Chalk and Tertiary systems, but those in the Chalk are nowhere worth working. Rock phosphate is also found in the Jurassic system in association with calamine, but the

occurrences are isolated and without economic importance. It is also found in the younger Tertiary series, principally the Miocene, and the first mining operations were begun in these deposits, but were abandoned owing to the expense of working and the restricted area of the deposits, notwithstanding their nearness to the coast. The principal mines are at Tebessa, where the phosphate rock is found in the Lower Eocene series just above the Danian section of the Chalk.

France.—The most important phosphate deposits are situated in the departments of Somme, Aisne, Pas de Calais, Oise, and Nord. By far the larger portion of France's total output of phosphate comes from these districts, the deposits of phosphatic chalk being almost exclusively the source of supply. The center of the phosphate mining industry in the Province of Picardy is the township of Roisel, situated on the frontier line between the departments of Somme and Aisne. At Templeux-la-Fosse large pockets of phosphatic sand, some of which extended 20 m. deep into the chalk, have been mined. The phosphatic chalk which still remains will be mined by means of shafts and drifts. The whole deposit of the phosphatic chalk forms a syncline 150 m. broad and about 3 km. long; the greatest width of the deposit is 9 m., the lower section containing about 45 per cent. and the upper about 35 per cent. of tricalcic phosphate. Only the lower section is being mined, there being several companies working on the deposit.

L'Engrais estimates the total production of phosphate rock in France during 1907 at 375,000 metric tons, as compared with 425,000 tons in 1906.

New Zealand.—A recent discovery is the existence of phosphate in the calcareous pockets near Clarendon, in the Province of Otago. The phosphates of New Zealand, which have not as yet been found in commercial quantities, except at Milburn and Clarendon, generally range from a yellowish white to a light gray in color. The phosphate is generally amorphous, but that recently found at Clarendon is crystalline apatite. Mining operations are being carried on by the Ewing Phosphate Company, at Clarendon, and by the Milburn Lime and Cement Company, at Milburn. Phosphates have also been discovered near the Waiau river, Southland.

Russia.—The phosphatic minerals on the banks of the Kertch straits have recently attracted attention. Some of the specimens examined ranged from blue to gray, while others were quite transparent and almost colorless, slightly tinged with blue, and with a specific gravity 2.66 at 20 deg. C. Samples from the Yanisch Tagil mines, about 20 miles south of the town of Kertch, yielded on analysis from 25 to 28 per cent. phosphorus pentoxide, 38 to 48 per cent. of iron oxide, 25 to 29 per cent. water, and small amounts of the oxides of manganese, magnesia and calcium.

Spain.—The high price of phosphate rock prevailing early in 1907

caused attention to be directed to the possibility of home production. Important phosphate deposits are known to exist in the province of Cacères, notably at Aldea Moret and in the Logrosan district. Phosphate has also been found at Millanes, Zarza la Mayor, Valencia de Alcantara, Casas de Millan, Torremocha, Albalet, and in Montaña. The rock of these deposits varies from 58 to 68 per cent. tricalcic phosphate. The only deposits worked to any great extent as yet are those at Aldea Moret, where two small superphosphate factories operate.

Tunis.—According to a correspondent of the *Mining Journal*, London, the shipments of phosphate rock in 1907 were as follows: Gasfa company, 750,000 tons; Kala Djerda, 240,000; Kala-Senaam, 100,000; La Floridienne, 30,000; total, 1,120,000. The railway from Metlauoi to the Gafsa company's Redeyef mines is now completed, and shipments of the rich rock of the latter (65 per cent. tribasic phosphate) will commence early in 1908. The increase in the Gafsa production over 1906 was 160,000 tons. During 1908 the total will probably reach 1,000,000 tons, the arrangements for the shipment of that quantity being already provided for. It is said on good authority that the sales effected for 1908 ensure a profit of 20 francs per ton.

The line from Sousse to Aïn Moularès is being steadily pushed forward and will be carrying the fertilizer from these beds a year hence. According to the terms of this concession the Gafsa company has to bring a minimum of 250,000 tons annually to Sousse. As far as the southern half of Tunisia is concerned, the Gafsa company has practically a monopoly, as it can exercise the option of taking over any ground conceded in that district on equal terms within a specified time of the adjudication. In this way it took over the Aïn Moularès concession on the terms of a bid made by another company.

Consul-General R. P. Skinner, of Marseilles, reports that some of the Tunisian phosphate companies are said to have contracted for much of their production up to 1915. Tunisians are exulting over the prosperity of the phosphate business, which reflects itself in the earnings of the Gafsa company. This company began with an output of 73,000 tons of phosphate rock in 1899 and sold 590,000 tons in 1906. The Société Anonyme La Floridienne has now entered the Tunisian field, promising to do a large business in Africa as well as in Florida. This is a Belgian corporation, with headquarters at Rue de la Loi, Brussels. The Florida company in its report of 1907 states that important quantities for delivery in 1909 had been disposed of at prices higher than those obtained for the output of 1907 and 1908. The Florida company own the Djebil Sal Salah deposit, one-half of royalties on the Aïn Moularès mine, and an interest in the Aïn Krimah mine.

PLATINUM.

The exceedingly high price for platinum that prevailed during the closing months of 1906, extending well into 1907, had the result of stimulating search for new productive localities in various parts of the world. In the United States, discoveries were reported from Yuba and Sacramento counties, California, from Lincoln county, Nevada, and from the neighborhood of Helena, Mont. Greater care was also shown in saving the platiniferous residues from gold placer mining in southern Oregon, in California, and, in fact, wherever platinum has been known to exist. Interest in the subject of platinum has been much more general throughout the West since the extensive series of tests on black sands was made by the U. S. Geological Survey, at the Portland Exposition. Heretofore the only platinum produced in this country was that, only a few hundred ounces in amount, recovered at the Mint in San Francisco, from the bullion yielded by certain Californian gold placers, and from the cyanide slime from certain western ores.

In foreign countries, new platinum localities were reported during 1907 in the Minas Geraes district of Brazil, near Grahamstown, in Cape Colony, and in the Hokitika district of New Zealand. Renewed activity was shown in Colombia, and new discoveries were reported from Russia. In the last mentioned country, which supplies 95 per cent. of the world's output, steps were taken by some of the most important producers to curtail the exports of crude platinum, in order to take better advantage of the prevail-

STATISTICS OF PLATINUM IN THE UNITED STATES.

Year.	Production. (a)		Imports.			Consumption.
			Unmanufactured.		Manufactured	
	Troy Oz.	*Value.*	*Troy Oz.*	*Value.*	*Value.*	*Value.*
1896	163	$944	83,080	$926,678	$106,338	$1,033,960
1897	150	900	83,080	960,299	43,921	1,005,120
1898	225	3,375	101,018	1,178,142	52,283	1,233,800
1899	300	1,800	187,778	1,462,157	55,753	1,539,710
1900	400	2,500	118,919	1,728,777	36,714	1,767,991
1901	1,408	27,526	85,438	1,673,713	24,482	1,725,721
1902	94	1,814	105,450	1,950,362	37,618	1,989,794
1903	110	2,080	114,521	1,921,772	135,889	2,059,741
1904	200	4,160	103,802	1,812,242	105,636	1,920,478
1905	318	5,320	104,196	1,985,107	188.156	2,176,263
1906	1,439	45,189	137,556	3,601,021	187,639	3,797,460
1907	....	....	74,208	2,509,926	175,651	....

(a) Statistics of the U. S. Geological Survey. (e) Estimated.

ing high prices, as well as to encourage the establishment of native refining works.

Toward the close of 1907, prices had returned more nearly to their former level, so that not all of the newly discovered localities are likely soon to gain much prominence. Potential sources of platinum will, however, always deserve careful investigation.

Platinum in the United States.

California.—It has recently been discovered that the old dumps of the Blue Point mine at Smartsville, Yuba county, carry platinum, none of this metal having been saved while the mine was being worked years ago. The property is being re-opened after a long period of idleness. The gravel in the dredge diggings in Folsom district, Sacramento county, has lately been found to carry more platinum than it was supposed to and devices are being adopted to save this metal as well as the gold.

Montana.—In Warm Springs gulch, nine miles from Helena, Wilton Browne, assayer and chemist of Helena, has discovered that the gold ores of the district carry platinum in the form of sperrylite.

Nevada.—Copper-nickel-platinum ores of Bunkerville district in northeastern Lincoln county, Nevada, occur in diabase eruptives in schist. There are three nearly parallel lenses, only the middle one of which has been exploited. The assay of the ore exposed is reported as 4 per cent. copper, 2.5 per cent. nickel and $\frac{1}{3}$ oz. platinum.

Oregon.—A considerable quantity of platinum was saved in southern Oregon, in the summer of 1907, all of it being taken from placer diggings. It occurs with black sand, and is saved by undercurrents and broad riffle tables connected with the gold sluices. The Deep Gravel mines, of Waldo district, save the greatest quantity of the metal.

Platinum in Foreign Countries.

Brazil.—According to Eugen Hussak, researches carried out during the last 30 years have lengthened greatly the list of Brazilian localities for platinum. That found in the Rio Abacte and its left-bank tributaries, probably derived from peridotites, is strongly magnetic, contains much iron, and is free from palladium. The platinum from Conceição and that from Condado, Serro, is in both cases non-magnetic, but the latter variety is rich in palladium, while the former is free from it. Along the eastern slopes of the Serra do Espinhaco platinum occurs in association with diamonds, derived from conglomeratic quartzites, but in such singular shapes that it must have been redeposited from solution—the outcome probably of the decomposition of platiniferous pyrites (such as are known to occur in the United States and Norway). Platinum has also been found in the

auriferous quartz-veins which traverse the crystalline schists of the Rio Bruscus, Pernambuco.

Canada.—Platinum occurs in small quantity in the nickel-copper ore of Sudbury and is recovered as a by-product from the matte. At another locality in Ontario, near Wabigoon, Rainy River, platinum was reported to have been found two or three years ago in some of the eruptive rocks.

British Columbia has for some time been known to contain deposits of platiniferous gravel, probably of commercial importance, those in the Cariboo district being the best known. The Consolidated Cariboo Hydraulic Mining Company annually reports the recovery of a marketable amount of platinum concentrate. The provincial mineralogist of British Columbia lately received from Lillooet, on the Fraser river, 2 oz. of black sand concentrate containing platinum. No information was received as to the source of the sand from which the concentrate was obtained, but it was probably the Fraser river. Heretofore no platinum has been reported as having been found in any other tributary of the Fraser than the Quesnel, which joins the Fraser near Barkerville, Cariboo, about 200 miles north of Lillooet.

Cape Colony.—Considerable interest was shown in 1907 in the discovery of platinum-bearing rocks in the Grahamstown and Port Elizabeth districts. The principal developments have occurred along the Assegai river, and in the district between the Kasouga and Kariega rivers. The rocks have been analyzed by eminent chemists in England, the general average being about 1 oz. platinum per ton, with additional gold content. Several syndicates have been organized, and development seems to be progressing actively.

Colombia.—The platinum industry in Colombia is attracting considerable attention from abroad. Recently a French company has made purchases of mining properties. Other capitalists have also secured large holdings, which they expect to develop scientifically.

The platinum of Colombia has been found only on the headwaters of the Atrato river and the Rio San Juan in the State of Cauca and near the Pacific coast of Colombia. Most of these deposits are also gold bearing and they have been worked for a good many years, principally for gold. Many important discoveries of platinum deposits have recently been made in the province of Lloro, but it seems, from the fact that the authorities have given orders to permit no mine denouncements within a region covered by a league on either side thereof, that the government intends to consider platinum a government monopoly, as provided in a recent disposition of the Colombian Congress.

New Zealand.—A discovery of platinum in the Hokitika district, by the Geological Survey, is recorded in Bulletin No. 1 of the Survey. The platinum occurs in quartz veins close to the Pounamu belt of mag-

nesian eruptive rocks. This mode of occurrence is of geological interest as the metal, when found in its native locus, usually occurs actually in the magnesian eruptive rocks. The only recorded occurrence of platinum in quartz in New Zealand was observed by J. A. Pond (*Trans.* New Zealand Inst., 1882), who isolated gold, silver, platinum, and iridium from quartz obtained at a depth of from 540 to 600 ft. in the Queen of Beauty shaft at Thames.

Platinum has been found in the present case in two lodes, the most important of these being that known as Harley's Creek, where the white, semi-vitreous, platiniferous quartz occurs in lenticular veins intercalated with the country rock, which consists of a dark shaly phyllite. Assays of two samples of this quartz yielded 0.167 oz. platinum and 1.2 oz. silver in one, and 0.054 oz. platinum and 0.4 oz. silver per ton in the other. The second locality is Taipo Gorge Reef, where the platinum occurs in a bedded quartz vein, which attains a maximum width of about 1 ft., and is enclosed in banded schists. The vein contains a small amount of pyrite and chalcopyrite. The assay of a sample of the quartz from this vein showed the presence of platinum, 0.05 oz., and silver, 0.3 oz. per ton, but no gold.

The physical condition of the metals in the above samples has not yet been determined, but it is interesting to note that in each case the silver accompanies the platinum in the ratio of 7 to 1, suggesting a combination of the two metals, Only in one of the three samples assayed does the platinum occur in paying quantities, and in this case the vein is small.

Russia.—The geographical and commercial aspects of the Russian platinum industry were described with great detail in Vol. XV of The Mineral Industry, and need not be repeated here. It was explained that the great majority of the Russian output is now sold in Paris and London on contracts dating from some time back, and at prices considerably below the recent market quotations. The producers, being shut off from participation in the benefits of an advancing market, have shown no inclination to augment their output, and will not do so until their present contracts shall have expired.

Discoveries of new platinum deposits, in the form of ore as well as of gravel, were reported from various parts of the northern Urals. One of the gravel deposits was found at a point six miles from the Barantchinsky works, on the Schoumikha river, a tributary of the Barantchi. Advices from Ekaterinburg, report the recent discovery of a platinum deposit on the Koiva. The platinum was discovered on the border of Prince Abamelek-Zararieff's, Count Schouvaloff's estates, and the Treasury domains, in the form of ore. It is added that it is found at no great depth from the surface. Platinum has also been found in the Zlatoustoff mining district, on the river Upudje, on the Treasury Estate of Artinsk. The discovery in the Artinsk district has caused particular surprise, since in

that part neither platinum nor gold has been known to exist up to the present time.

PRODUCTION OF PLATINUM IN RUSSIA.

(In troy ounces. Statistics derived from various sources.)

Year.	Ounces.	Year.	Ounces.	Year.	Ounces.
1869 (*a*)	60,480	1894 (*a*)	137,376	1901 (*a*)	168,048
1874 (*a*)	30,240	1895 (*a*)	159,624	1902 (*b*)	197,173
1882 (*a*)	108,000	1896 (*a*)	130,032	1903 (*b*)	193,000
1890 (*a*)	116,640	1897 (*a*)	147,744	1904 (*b*)	161,139
1891 (*a*)	111,715	1898 (*a*)	158,544	1905 (*b*)	168,508
1892 (*a*)	120,614	1899 (*a*)	151,200	1906 (*c*)	210,318
1893 (*a*)	134,482	1900 (*a*)	134,222	1907 (*d*)	160,105

(*a*) W. A. Dyes, *Chem. Ind.*, 1905, p. 387. (*b*) W. A. Abegg, Warsaw, privately communicated. (*c*) E. de Haupick, London *Mining Journal*, May 4, 1907. (*d*) Exports.

Markets.

New York.—In the first three months of 1907 the demand for platinum was strong. Early in April a break in the market occurred which was followed by a steady decline. In the last quarter the financial depression had a marked effect on the platinum industry in that manufacturers of jewelry and electrical supplies, using platinum, greatly curtailed their purchases; the demand for platinum chemical-ware was also reduced.

AVERAGE MONTHLY PRICES OF PLATINUM AT NEW YORK.

(In dollars per troy ounce.)

	1906		1907			1906		1907	
	Ordinary	Scrap.	Ordinary	Scrap.		Ordinary	Scrap.	Ordinary	Scrap.
January	20.50	16.00	38.00	31.50	August	26.00	21.50	28.125	22.625
February	25.00	19.00	38.00	31.75	September	32.10	24.00	28.70	23.30
March	25.00	19.00	37.00	30.75	October	33.00	25.50	27.125	21.25
April	25.00	19.00	32.50	24.75	November	35.50	28.38	26.312	18.937
May	25.00	19.00	29.50	21.125	December	38.00	31.25	26.00	17.62
June	25.40	19.75	26.20	20.30					
July	26.00	21.50	26.75	21.437	Year	28.04	21.99	28.183	22.310

Ekaterinburg.—The prices for platinum on the Ekaterinburg exchange decreased steadily throughout the last half of 1907, and the demand became very low. In October Rs. 6 and 6.50 were offered for 1 zolotnik of crude platinum of 83 per cent. In November the price for platinum was down to Rs. 4, or 4.60 per zolotnik. The platinum mine owners explained this by the financial crisis in America, which embarrassed American buyers, on the one hand, and by the great quantity of platinum offered by Ural merchants, on the other hand, who receive platinum stolen from the mines.

PRICE FOR CRUDE PLATINUM OF 82 PER CENT. IN THE URALS.

Year.	Rubles. Per Pood.	£ Per Oz.	s.	d.	Year.	Rubles. Per Pood.	£ Per Oz.	s.	d.	Year.	Rubles. Per Pood.	£ Per Oz.	s.	d.
1874	4,800	0	19	0	1899	7,000	1	7	8	1906 (Jan.).	22,000	4	7	9
1888	6,000	1	4	0	1901	16,200	3	3	9	1906 (Oct.).	34,000	6	15	8
1890	6,200	1	4	8	1902	17,300	3	9	4	1907 (Jan.).	30,000	5	18	11
1893	6,500	1	5	10	1903	18,500	3	13	3	1907 (Feb.).	29,000	5	15	8
1895	6,600	1	6	7	1904	21,000	4	3	8	1907 (Mch.).	28,000	5	12	0
1898	6,800	1	7	0	1905	22,000	4	7	9	1907 (Apr.).	27,000	5	8	0

London.—The average yearly quotations for ingot platinum for a series of years is given below:

AVERAGE PRICE PER OUNCE TROY FOR PLATINUM INGOT.

Year.	£	s.	d.	Year.	£	s.	d.	Year.	£	s.	d.
1874	1	5	2	1900	3	19	9	1906 (Jan.).	4	15	2
1888	1	13	8	1901	4	1	11	1906 (Oct.).	7	19	8
1890	1	15	8	1902	4	4	0	1907 (Jan.).	7	0	0
1893	1	17	9	1903	4	6	1	1907 (Feb.).	6	19	6
1895	2	2	0	1904	4	8	1	1907 (Mch.).	6	19	0
1898	3	13	6	1905	4	10	4	1907 (Apr.).	6	18	8

Technology.

Refining.—There are two methods of refining platinum ore. In the Wollaston method, after removal of the metals associated with platinum with the successive action of nitric and hydrochloric acids, the platinum itself is dissolved in aqua regia, from which it is precipitated by a solution of sal ammoniac in the form of sparingly soluble ammonium platino-chloride. This salt is washed and heated to redness, by which the chlorine and ammonia are expelled, leaving the metal in the form of a gray, spongy soft mass, known as spongy platinum. In this form platinum can not be fused into a compact form by ordinary furnace heat, but it can be welded at a high temperature. Accordingly, it is made into a thin paste with water, then introduced into a brass mold and subjected to a graduated pressure, by which the water is squeezed out and the mass rendered sufficiently firm to bear handling. It is then dried, very carefully heated to whiteness, or hammered or subjected to powerful pressure. The Deville-Debray method requires a simple furnace. Two flat pieces of quicklime are scooped out, like two cupels, and form the bottom and the lid of the furnace. The lower cupel has a notch cut in its side to serve as an exit for the liquefied platinum. The upper one is pierced at its center with a slightly conical round hole, through which the nozzle of any oxy-hydrogen blowpipe enters, so that the flame beats down on the metal within.

Oxidation of Platinum.[1]—Platinum is perceptibly oxidized by mere contact with oxidizing agents. Pieces of platinum foil (both pure and containing different quantities of iridium) were placed in a solution composed of 10 gm. of potassium permanganate, 10 c.c. of sodium hydroxide solution and 200 c.c. of water, control pieces being placed in a similar alkaline solution containing no permanganate. After 24 hr. at the ordinary temperature, the pieces of metal were thoroughly washed, and immersed in a hot hydrochloric acid solution of potassium iodide. After a few moments the solutions containing the pieces of metal which had been in contact with permanganate acquired the pink color characteristic of dilute solutions of platinum to which an iodide has been added. The presence of dissolved platinum is also indicated by hydrogen sulphide, the depth of color being comparable with that produced in an equal volume of a solution containing 0.00004 gm. of platinum. Similar oxidation of platinum is effected by potassium persulphate, bichromate, chlorate, and permanganate in sulphuric acid solution and by potassium ferricyanide in alkaline solution. Ferric chloride in acid solution and hydrogen peroxide in acid or alkaline solution appear to be without action on platinum. Strong nitric acid has also a considerable oxidizing action on platinum, especially when hot. So far as could be judged, pure platinum and 20 per cent. platinum-iridium were attacked more strongly than 1 per cent. platinum-iridium.

Platinum Silicide.—Silicon and platinum give, by direct union, a silicide corresponding to the formula SiPt. This compound, which can be obtained crystallized, has chemical properties somewhat like those of platinum, though more readily attacked by oxidizing agents.

Platinum Amalgam.—A note in the *Am. Journ. Sci.* (June, 1907) gives some interesting facts concerning platinum amalgam. It is well known that when mercury is shaken up with water, the two liquids separate as soon as the agitation is stopped. Moisan has observed that it is otherwise when the mercury contains platinum in solution. After a few seconds of agitation the platinum amalgam forms a semi-solid mass of buttery consistency with a volume about five times as great as that of the original amalgam. The emulsion thus formed is so permanent that it does not appear to have changed its volume after standing at rest for a year. It resists the action of heat, for it may be heated to 100 deg. C. without apparently changing its volume, and without any disengagement of gas. When the emulsion is subjected to a vacuum it diminishes in volume, a little water separates, and bubbles of gas are given off. The emulsion may be made by shaking 2 c.c. of distilled water to which has been added a few drops of 10 per cent. solution of platinic chloride.

[1] C. Marie, *Compt. rend.*, 1908, pp. 475-77.

POTASSIUM SALTS.

By Reginald Meeks.

Germany continues to be the chief source of potash salts. In 1907 its mines showed a net increase of 344,734 metric tons over the 1906 production. This increase was principally in miscellaneous potash salts and potassium chloride. The greatest decrease was in kainite, which showed a falling off of 96,182 tons from the production of 1906.

PRODUCTION OF POTASSIUM SALTS IN GERMANY. (a)
(In metric tons and dollars; 1 mark=$0.238.)

Year.	Kainite.		Potassium. Chloride		Potassium. Sulphate.		Potassium Magnesium Sulphate.		Other Salts. of Potassium.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
		$		$		$		$		$
1896	856,290	2,989,736	174,515	5,718,559	19,682	813,381	4,623	85,977	902,707	2,964,750
1897	992,389	3,486,007	168,001	5,764,423	13,774	565,720	7,812	149,079	953,798	3,030,143
1898	1,103 643	3,835,856	191,347	6,380,220	18,853	763,397	13,982	259,485	1,105,212	3,576,628
1899	1,108,159	3,838,250	207,506	6,801,250	26,103	1,027,500	9,765	195,000	1,384,972	4,202,000
1900	1,178,527	4,134,000	271,512	8,793,750	33,853	1,249,250	15,368	280,500	1,874,346	5,643,750
1901	1,500,748	4,327,250	282,750	8,782,250	27,304	1,460,000	15,612	286,500	2,036,326	5,443,250
1902	1,322,633	4,571,980	267,512	7,507,710	28,279	1,079,092	18,147	334,390	1,962,384	4,949,448
1903	1,557,243	5,208,154	280,248	8,125,320	36,674	1,389,444	23,631	441,252	2,073,720	4,993,478
1904	1,905,893	6,322,470	297,238	8,425,676	43,959	1,664,572	29,285	545,972	2,179,471	5,305,972
1905	2,387,643	7,976,808	373,177	10,580,528	47,994	1,804,040	34,222	614,754	2,655,845	6,396,250
1906	2,720,594	8,918,574	403,387	11,034,632	54,490	2,032,520	35,211	644,028	2,821,073	6,538,336
1907	2,624,412	8,579,206	473,138	12,639,704	60,292	2,216,494	33,368	631,652	3,124,955	7,314,930

(a) *From Vierteljahrshefte zur Statistik des Deutschen Reichs.*

EXPORTS OF SALTPETER FROM INDIA. (a)
(In tons of 2000 lb.)

Year.	Quantity.	Value.	Value per 100 lb.	Year.	Quantity.	Value.	Value per 100 lb.
1897–98.......	20,889	$1,329,155	$3.06	1902..........	21,882	$1,359,335	$3.11
1898–99.......	18,263	1,164,480	3.06	1903..........	23,105	1,450,980	3.14
1899–00.......	19,870	1,281,050	3.64	1904..........	21,894	1,331,745	3.04
1900–01.......	17,432	1,471,245	4.05	1905..........	17,535	1,178,615	3.36
1901–02.......	17,721	1,189,400	3.22	1906..........	19,446	1,352,735	3.48

a) From "Mineral Production of India," by T. H. Holland, Government Geologist.

The exports of saltpeter from India in the last five years indicate a gradual rise in value, but the industry shows no signs of real expansion. Most of the salt is obtained in the Province of Behar, where 50,469 workers were employed in the industry in 1906.

The Potash Mines at Stassfurt, Germany.

At a general meeting of the "Kalisyndikat'' held late in November 1907 at Magdeburg the Günthersthal, Thüringen and Heldrungen mines were admitted as members and a provisional arrangement with the Krugershall company was approved. At the present time the "Kalisyndikat'' is composed of 40 members and there are nearly as many independent producers. Ten more mines are expected to produce within two years.

The German government is one of the most important members of the syndicate and does much to maintain the policy of selling principally to German consumers. The chief outside producer is the Sollstedt mine which has been steadily underselling the syndicate with the purpose of obtaining American trade. In 1906 the German government acquired the Hercynian mine and the idea then prevailed that other mines also would be taken over; this additional acquisition, however, did not materialize.

In the early part of November, 1907, the owners of potash mines in Germany received formal notice from the authorities, that they must at once make their final representations concerning the proposed second shaft, which the Government engineers declare is necessary for the protection of the lives of the miners. The mine proprietors assert that these shafts will be so expensive and difficult of construction that the mines will be compelled to shut down, even if it be possible to sink the second shaft.

There are in all about 50 workings which must take the necessary steps to put down the second shaft. *Kohle, Kali und Erz* estimates that the average cost per shaft will be about $600,000; that is, the German potash industry must submit to a total expense of $30,000,000. Inasmuch as this expense must be borne at once and not distributed over a period of six or eight years the edict can have none other than a serious effect on the German potash industry.

The Refined Salt Market in 1907.

Caustic Potash.—Business was chiefly on contracts. The movement covering the first six months was above normal; that during the summer months was quiet; fall shipments were below those of the fall of 1906. Late in the year some spot buying developed where contracts had expired. Prices at the beginning of the year ranged between 5c. and 6c. In July a temporary scarcity influenced an advance to 5@6¼c. and in November values receded to 4@6⅛c. where they remained till the close of 1907.

Carbonate.—Prices prevailed at 4@4½c. for 80 to 85 per cent. calcined,

$4\frac{3}{4}$@$5\frac{1}{2}$c. for 96 to 98 per cent. calcined and $4\frac{3}{4}$@5c. for 80 to 85 per cent. hydrated. The general depression caused a diminution in business but no falling off in prices.

Chlorate.—Prices remained stationary all year and the movement was steady and confined chiefly to contracts. The quotation for 1907 was $8\frac{3}{4}$@$9\frac{1}{4}$c. and in October it was announced that the carload price for 1908 would be $8\frac{3}{4}$c. for crystals and 9c. for powdered.

Nitrate.—The market was steady all year and prices ranged from 3.90c. to 4.50c. for the crude and 4.75c. up for refined salt depending upon quantity and seller.

Bromide.—The market was steady at 16c. until November and December when cuts of 1c. and then 2c. per ℔. were made.

PRECIOUS STONES.

By George Frederick Kunz.

In the United States there was considerable activity in the mining of precious stones in 1907, especially the tourmalines of Oxford county, Maine, a number of which were cut by local lapidaries. Of special interest was the finding of small quantites of spodumene of variety similar to that found at Pala, California, called kunzite. This spodumene was also found of a sea-green color, which, when heated, would change to lilac and violet. The great event of the year was, of course, the discovery of diamonds in Arkansas. This is discussed at length in a subsequent part of this article.

Turquoise.—Turquoise was worked at four places during 1907, but, although a quantity of material has been obtained up to the present time, very little of the fine blue variety has been secured.

A new and interesting find of turquoise has been made near Las Vegas, Nevada. The turquoise is the blue and blue-green material usually found near the surface. The trachyte rock in which it occurs strongly resembles that occurring near Los Cerrillos, New Mexico. Only a depth of 4 ft. has been dug, but for the depth the color of the turquoise is excellent.

An interesting variety of turquoise, attached to a peculiar trachyte matrix, was found near Austin, Nev., and has been extensively cut in the form of cameos, some of which measure from 2 to 3 in. in diameter. The most charming contrast is obtained when the subject is cut in blue, relieved by a background of the fawn-colored matrix.

Amethyst.—A locality for amethysts was found a few years ago in Nelson county, Va., near Lowesville, which I have described elsewhere.[1] The find consisted of a large, decomposed vein, the gangue rock being almost entirely missing through the decay of rocks. The region covered about 100 acres, with pockets scattered over the same and apparently did not extend down to any depth. These amethysts were unusually fine in quality. They occurred in a pegmatite rock and the crystals were more or less irregularly colored, like the finer stones that are found in Chitanka (Perm), Ural mountains, or the water-worn amethysts from Ceylon, and differing from

[1] "Mineral Resources of the United States," p. 851, 1902.

those found in the amygdulas of igneous rocks in Brazil and Uruguay, and at Thunder Bay, Lake Superior. The color is more unequally distributed, but for this reason, when cut, they are much more brilliant and frequently show not only a purple, but a red light. The finest gem found here was shown at the Jamestown Exposition, and was a heart-shaped stone weighing 114 carats, of the most gorgeous purple, rivaling the richest-colored wine.

During 1907 this locality was extensively worked. Many hundred tons of gangue rock were removed and some thousands of gems were mined, cut, and sold to the jewelry trade. Many of these are of unusually rich color, the largest stones weighing from 50 to 150 carats. A great many gems were obtained that were a pale pink, or almost lilac, but possessing the same brilliancy as the darker purple stones and forming very beautiful and interesting gems.

Kunzite.—Kunzite was worked at several places in California, but the quantity obtained during 1907 was not commensurate with the demand. An attempt has been made to sell the stock of a mine of the almandite garnet crystals which occur so plentifully on the Stickeen River, Alaska, seven miles from Fort Wrangell. These garnets appear as unusually brilliant crystals, occurring in numbers, a dozen or more together, in a compact gray mica schist, and are sold extensively to visiting tourists. But a great number of these specimens have been tried by lapidaries, with the disclosure of imperfection in the interior, which is more or less opaque, or only slightly translucent. On account of this defect, their use as gems will be very doubtful.

Hiddenite.—Some development work was carried on at the emerald and hiddenite mine at Stony Point, Alexander county, N. C., during the summer of 1907. Some small beryls, emeralds, and hiddenite were found, in value not exceeding a few hundred dollars. Preparations were made for continuing the work at some future time.

Chalcedony.—The most splendid specimens of chalcedony, colored blue and green by silicate of copper, have been found in the Copper Queen mine in the Globe district, Arizona. This mineral is frequently in thin layers, with a coating of chrysocolla; sometimes, the entire material is one evenly-distributed mass of green chalcedony and then a layer of chrysocolla coated by quartz. This chalcedony is unusually beautiful, the most striking resembling turquoise in color, but it is somewhat harder and less likely to change its hue, as the chalcedony is not as absorbent as is the turquoise. This green chalcedony has been cut extensively in Arizona and New Mexico, and also by the jewelers in the East, in the form of ring-stones, seal-stones, and, in exceptional cases, of spherical beads, which are quite as beautiful as any of the best turquoise. Single pieces of this material weighing some ounces each have been found, worth \$100 to \$300. It is

known as "copper silicate" chrysocolla. I have offered "azurlite," or "azurchalcedony," as a substitute.[1]

In San Diego, Cal., there has been found a variety of chalcedony impregnated with a manganese mineral or stained by manganese, the color being somewhat similar to that of triphylite; for this, because of its peculiar purple tint, the name violet or mangan-chalcedony has been suggested.

Azurmalachite.—I have given the name "azurmalachite" to the natural mixture of azurite and malachite which sometimes occurs in concentric layers, as in the form of stalactites, or as botryoidal masses, which, when cut crosswise, show regular or irregular bands, rings or markings of the blue azurite combined with the green malachite. This material is found at Bisbee and at other copper mines in Arizona. The naming of the stone has been a result of the introduction of a quantity of this material in the less expensive forms of jewelry; the stones being cut for cuff-buttons, rings, scarf-pins, etc. Neck-pieces also, in the form of spherical beads, offer, in their irregular blending of blue and green, a pleasing and attractive effect.

Utahlite.—During 1907, there was found in Colorado a serpentine of the most intense chrome green color that has been observed anywhere, closely resembling the green of the utahlite, a variety of variscite from Dugway, Utah. However, this serpentine is translucent, and not amorphous in texture as is the latter.

Utahlite itself has been more or less worked at the new locality found two years ago, and a quantity of gems have been obtained which are excellent in color; these were frequently cut where a band across them produced an effect resembling that of the cat's-eye. Under the belief that another name than utahlite would insure a better sale these stones have been given a popular selling name (not a scientific name) by the dealers who call them "amatrice."

Californite.—The interesting californite, so closely resembling jade, but really a compact idocrase or vesuvianite, has been found at a new locality in Tulare county, several miles east of Exeter. The new California occurrence is of a pleasing apple-green color. It appears in veins that vary from 2 to 4 in. in thickness, associated with serpentine in a serpentinous rock. A very limited amount has been mined and sold, perhaps not exceeding in value a few thousand dollars.

Rose Quartz.—Rose quartz of magnificent coloring and size has been found in Riverside county, California, at a point in the Coahuila mountains where a lode, varying from 3½ to 6 ft. in thickness, remains to be developed. A new and unique use for this beautiful substance is that to which it will be put by Miss Agnes V. Luther, teacher of natural science in the Normal and Training School at Newark, N. J. She has obtained a mass of many

[1] New York Academy of Sciences, April, 1907.

hundred pounds at the Kinkels feldspar quarry near Bedford, Westchester county, N. Y., and has had it made into a tombstone and placed in the old Indian cemetery at Saybrook, Connecticut. Rose quartz, however, has the reputation of losing its color by exposure to sunlight, and for this reason has not generally been regarded as suitable for out-door uses; but I have seen glaciated surfaces of deep pink color in the Black Hills and the outcrops retained their color. Miss Luther's experiment will be interesting as a test.

Agatized Wood.—Agatized, chalcedonized, or petrified wood, as it is called, has been used by Charles Mead in building his home at New England, Hettinger county, North Dakota, in the neighborhood of which it occurs. The house stands on the edge of a bluff, overlooking the Cannonball river, and forms a striking object of color and brilliancy. The material has been left in its original condition, and the wall is unpointed; as the petrified wood contains cavities with coarse crystals, the play of light on its surface produces a beautiful effect. This material is not so fine as that found in Chalcedony Park, Arizona, but in one tract of three acres in extent the specimens are so thick that tons can be obtained with little difficulty.

Benitoite.—One of the most interesting of the new gems found in the United States is the mineral called benitoite, a stone from California, described by Prof. George Davis Louderback, of the University of California.[1] This remarkable blue gem resembles the blue spinel rather than the sapphire and, like the spinel, it has a peculiar brilliancy of its own. The mineral was discovered early in 1907 by Mr. Hawkins and T. Edwin Sanders, who were prospecting in the southern part of the Mt. Diablo range near the San Benito-Fresno county line, about latitude 36° 20′. It was first brought to Prof. Louderback's attention by Shreve & Co., a San Francisco firm, who had purchased one of the cut stones from a lapidary and who were later offered some of the rough material as sapphire. Investigation at the University proved it to be an undoubtedly new mineral species, and it was called benitoite, as it occurs near the headwaters of the San Benito river in the county of the same name.

The most striking outward characteristic of this mineral is its fine blue color, and selected crystals cut in the right direction produce a beautiful gem-stone that rivals the sapphire in tint and excels it in brilliancy. The color, however, although fairly characteristic, is not an essential property, for parts of a crystal are often colorless, while occasionally small crystals are entirely so. The color also varies in intensity in different crystals or in parts of the same one. When pale it is rather a pure blue; but when more intense it assumes a violet tint. Besides this variation in color in different parts of crystals, there is a difference at any one point, depending

[1] "Benitoite, A New California Gem Mineral," by George Davis Louderback, with Chemical Analysis by Walter C. Blasdale; University of California, Bulletin of the Department of Geology, Vol. V, No. 9, pp. 149-153.

on the direction in which the light passes; in other words, the mineral is strongly dichroic, the ordinary ray being colorless and the extraordinary being blue. A section cut parallel to the basal plane is practically colorless, while sections parallel to the principal axis show the deepest tint. To get the finest effect, therefore, gems should be cut with the table parallel to the principal axis; this is the reverse of the sapphire, which shows its color best when cut perpendicularly thereto. If such a section, cut so as to give the strongest color effects, be examined with a dichroscope, the contrast between the images is most striking. The image of the extraordinary ray, being freed from the colorless image of the ordinary ray, presents a remarkable intensity of color, very much deeper, of course, than can be seen by looking at the mineral in any direction with the unaided eye. In the lighter parts this color of the extraordinary ray is a slightly greenish blue inclining to indigo as it becomes darker, and is very similar to one of the axial colors shown by some colites (cordierites); but in the more highly-colored or thicker parts it is an intense purplish blue. The color is not affected by heat, up to the melting point of the mineral. Fragments brought to a rather bright red and maintained at that heat, just short of fusion, for five minutes, showed no change whatever on cooling.

Benitoite occurs generally in individual simple crystals scattered through a matrix of and varying from a few millimeters to about 2 cm. across. The matrix being translucent white, the blue transparent crystals stand out prominently, often showing well-defined faces.

Benitoite and carlosite, a new microclinic mineral with a hardness of between 5 and 6, occur as individual disseminated crystals in narrow veins in a basic igneous rock or in a schist which has been considerably altered by the solutions that formed the veins. The benitoite is apparently restricted to the veins, the carlosite also occurring in the neighboring parts of the wall rock. The chief gangue of the veins is a soda-rich zeolite.

Benitoite crystallizes in the hexagonal system, trigonal division. The observed forms are the basal plane, the plus and minus trigonal pyramid and the corresponding trigonal prisms. The normal angle between the basal plane and the pyramid is about 40° 14′. If the pyramid be taken as a unit pyramid of the first order, this would yield an axial ratio of 0.7327, if of the second order, 0.8460. The most common habit is pyramidal, one pyramid being the chief form, the other occurring as a small but regular and brilliant truncation. One or both prisms may be present as narrow truncations and also a small triangular basal plane. No tendency towards a prismatic habit was observed. The angles between two adjoining pyramid faces at one end of the axis is 68° 1′. There is an imperfect pyramidal cleavage. The fracture is conchoidal to subconchoidal. The hardness is 6¼ to 6½; distinctly above orthoclase and labradorite and below chrysolite and quartz; specific gravity, 3.64 to 3.65.

The refractive index is quite high, a feature which adds greatly to the beauty of the cut stone. For the ordinary ray it is about 1.77 (sodium light), for the extraordinary, about 1.80. The double refraction is therefore very strong and the mineral optically positive. Basal sections show a perfect uniaxial cross which gives a distinct positive reaction with the mica plate. This mineral fuses quietly to a transparent glass at about 3. It is practically insoluble in hydrochloric acid, but it is quite easily decomposed by hydrofluoric acid. Slowly attacked by melted potassium pyrosulphate, it dissolves readily in fused sodium carbonate. Benitoite has proved to be of much interest from the standpoint of its chemical composition which is new and unusual. The suggested formula is $BaTiSi_3O_9$, which yields the following calculated values: SiO_2, 43.71 per cent.; TiO_2, 19.32; BaO, 36.97; total, 100.

ANALYSES OF BENITOITE. (*a*)

	A.	B.	Average	Mol. Ratios.
SiO_2	43.56	43.79	43.68	.723
TiO_2	20.18	20.00	20.09	.250
BaO	36.34	36.31	36.33	.237
	100.08	100.10		

(*a*) As reported by Prof. W. C. Blasdale.

Benitoite is then a very acid titano-silicate of barium, and stands in a class by itself, both among acid silicates and titano-silicates. The possibility of the titanium acting as a base was considered, but the summation of the analyses and the fact that the crystals are often perfectly colorless seem to point definitely to the above interpretation. The blue color of much of the material may be due to a small amount of titanium in the sesquioxide condition.

Both chemically, and as a gem-stone this is a most interesting addition to American minerals. It is to be regretted that its hardness is less than 7, so that it is not as durable for gem purposes as sapphire or spinel. The stone has been found only very sparingly and up to the present can be considered mineralogically a rare gem.

Sapphire.—So far, the most noble blue gem, one of the four precious stones which has not been in as great favor as the diamond, the ruby, or the emerald, is at last coming into fashion. With it, the color blue is also becoming more in vogue. The result is that sapphires of every clime are worn and with them has come the demand for not only the finest Oriental, but also the finest American blue stones. Strange to say, however, the gem of from two to four carats is a rarity in the United States. Great quantities of beautiful blue stones from $\frac{1}{8}$ to $1\frac{1}{4}$ carats each have been found, and the two mines in Fergus county were productive until the panic of 1907.

Since then not so much mining has been done. The stones of various colors, found in Granite county and at Eldorado Bar were also much worked, but, with the slump in many industries, there was less demand for them.

During 1907 some rolled sapphire crystals from an unknown locality in Colombia were shown to me; white, colorless, yellow and pale blue; in hexagonal crystals, flat and barrel-shaped. In general character they resembled the Montana gems found on the Missouri river near Helena, at Eldorado Bar. They were quite perfect and would cut into gems of fancy quality. With them was some cyanite and also some spessitite garnets in small grains of ½ carat each. This is the first recorded finding of sapphire from this part of South America; their nearer locality could not be ascertained.

Emerald.—After many vicissitudes, and after having offered every possible form of concession in order to have them worked, the historic mines of emerald belonging to the Colombian government are in the hands of President Rey, who has under consideration the placing of a loan of about $2,500,000, which is to be secured by the mines, the interest to be met by the profits from these virtually unique mines of fine emeralds, with over three centuries of history.

Diamonds.

The financial crisis in the United States in the fall of 1907 affected very seriously the diamond industry of South Africa. America had been for several years the largest purchaser of diamonds; and the changed conditions here soon began to show themselves in a great falling off in the importation of luxuries. The result was alarming to the diamond companies, and steps were at once taken to reduce the output and prevent serious losses from an overstocked market. The De Beers company had been taking out 30,000 tons of rock per day, working its five mines for six days of 24 hours each week. In a short time the hours were reduced to one-half, and subsequently the days were made five instead of six. In April, 1908, the Dutoitspan mine was shut down; this had furnished over one-third of the company's output. By these various steps the total yield was reduced to 11,000 tons per day. All the other companies either restricted their output or suspended operations entirely, only the more important ones continuing work at all. The Premier, by agreement with the De Beers company in December, shut down portions of its plant, and lowered its output by about 30,000 carats a month, which would represent over 80,000 tons. In addition to these business difficulties, a series of labor troubles in the form of strikes in Antwerp and Amsterdam resulted in the going out of more than 10,000 diamond-cutters, who, on May 4, 1908, had not yet returned to work. All these causes have had a direct bearing on the

imports and created a tendency to steady the market; and there has been no break in the prices, nor is there likely to be, since the Syndicate intends to restrict the output until the visible supplies in the jewelers' hands are virtually exhausted. In the face of this fact, the strike of the diamond-cutters revealed that an effort was being maintained to keep up the price of polishing. Negotiations were started by the Federation of Employers with the Diamond Syndicate and the Premier Diamond Company for taking measures to restore confidence in the trade, and thus bring the diamond industry back to its normal aspect.

Diamonds in the United States.

No further diamond discoveries have been made in the drift of either Wisconsin or Indiana, nor is there knowledge of any diamonds having been found at the two places in California and Kentucky where prospecting has been carried on since 1906 and where there were supposed diamond-bearing deposits. We have, however, to note what is believed to be the first occurrence of diamonds in a matrix of kimberlite on the American continent (diamonds have never been found in the true matrix in the Guianas or in Brazil). This discovery was made near Murfreesboro, Pike county, Arkansas, in August, 1906, and together with Dr. Henry S. Washington I was permitted to investigate the locality. The geographical conditions, the character of the deposits and the methods of working will be briefly explained.[1]

Geology.—The geology as well as the petrography of this interesting locality has been well described by J. C. Branner and R. N. Brackett.[2] Briefly summarized, the igneous rock in which the diamonds are found is a vitreous peridotite, forming a stock or volcanic neck, which has broken up through Carboniferous and Cretaceous quartzites and sandstones. After an extensive period of erosion, during which an unknown portion of the neck and presumably a previously existent volcanic cone were removed, the surface was covered with thin beds of post-Tertiary conglomerate. The volcanic intrusion was accompanied by the formation of several small dikes of a rock much like that of the main body. One of these dikes cuts across the stock, while another cuts the Cretaceous sandstone, but is overlain by the conglomerate, thus giving a datum for the period of intrusion. So far as known, there was little, if any, metamorphism of the country-rock by the igneous magma, which probably followed an approximately vertical course, so that a more or less vertical extension downward of the igneous body to indefinite depths may be expected. This result should hold good, at least for the upper and most accessible portions, though

[1] "Diamonds in Arkansas," by George F. Kunz and Henry S. Washington; *Trans.*, A. I. M. E. (New York Meeting, Feb., 1908), pp. 187-194.
[2] *Am. Journ. of Science*, Third Series, Vol. XXXVIII, pp. 50 to 59 (1889); and *Annual Report* of the Geological Survey of Arkansas for 1890, Vol. II, pp. 377-391 (1891).

some departure from a strict verticality may be expected at greater depths.

As above remarked, the igneous rock is a peridotite which, in fresh hand-specimens, is tough, hard, distinctly porphyritic, and very dark greenish- or brownish-black. Microscopic study shows it to be composed of numerous crystals of olivine and some patches of biotite, imbedded in a ground-mass of very small crystals of augite, perofskite, and magnetite, with an abundant yellowish to colorless glassy base. In all the specimens examined the olivines are more or less completely serpentinized, and the glass is apt to show an aggregate polarization due to decomposition. The rock is evidently an igneous intrusive, which probably welled up in comparative quiet, and solidified not far from the surface. It is therefore in no sense a volcanic breccia, due to explosive eruptions, as are most of the South African occurrences. Chemically and mineralogically, however, it much resembles the South African rock, although there are certain points of difference—notably the absence of inclusions.

Peridotites are generally prone to alteration by weathering. In this instance the freshest rock is dense, hard and tough, and does not crumble markedly on exposure, as is shown by the fact that the highest points in this igneous area are outcrops of fairly fresh peridotite. The first state of pronounced alteration is the disintegration of the firm rock into a mass of hard, angular fragments, varying in size from that of a bean to that of a human fist, which apparently do not readily disintegrate on exposure to the weather. The second stage of alteration, due to further weathering, yields a compact mass, the so-called "green ground," showing various shades of light olive green, and often bluish in tint when moist, but becoming yellowish on drying. The third stage of alteration, found nearer the surface, furnishes, from still further oxidation of the ferrous iron, the so-called "yellow ground," which resembles the "green ground" in physical characters, but is, in color, distinctly brownish-yellow, with little or no trace of green. The green ground and the yellow ground are soft and friable, crumble readily between the fingers, and show soft, but sharply defined, serpentinous pseudomorphs of the original olivine crystals, with well-preserved outlines. This fact, supplemented by the general appearance of the texture, shows clearly that the peridotite has been decomposed in place, and that there has been little or no transportation of the material.

Both the green and the yellow grounds, if dry, crush under a gentle pressure to a fine powder, containing small gritty particles of the less decomposed minerals, which can be readily sifted out. If wet, the rocks disintegrate rapidly, especially with mechanical agitation, to a fine, somewhat sticky mud, which can be easily washed or otherwise treated.

The fresh, compact peridotite crops out at the surface, forms several small hills along the northwest border of the deposit, and is also visible

at other points; and the first fragmentary alteration-product shows itself at a few spots; but the green and the yellow grounds are found over by far the greater part of the igneous area, either on the surface or, more frequently, immediately beneath a thin layer of black, sticky, "gumbo" soil. The maximum and average depths of this mass of decomposed peridotite have not yet been exactly ascertained; but borings show it to be, in places, 40 ft. thick. This fact, together with other considerations, leads us to estimate the average thickness to be not less than 20 ft. Below this is found either the fragmentary, or a more or less compact, igneous rock. One drill-hole has penetrated the peridotite to a depth of 205 ft., another to 186 ft., and a third to 80 ft.—all remaining in igneous rock to the end, as was to be expected, in view of the geologic structure.

The surface original exposure of the igneous area forms a rough ellipse, about 2400 ft. in major and 1800 ft. in minor diameter. The area known to be underlain by peridotite is estimated at about 40 acres, though further prospecting of the neighboring alluvium-covered bottom-land to the south may possibly add to this amount.

General Conditions.—A variable supply of water, usually abundant, is furnished by the Little Missouri river, which flows a short distance to the southwest of the igneous area. This stream, though somewhat low at certain seasons, never runs dry, and may safely be counted on to provide a sufficient supply of water for all mining purposes. For certain installations, however, its rapid and sometimes serious rises must be taken into consideration. The owners of the igneous area possess, also, a large tract of land, along both sides of this stream, with the incident water-rights. A large portion of the land under control is well wooded; and extensive forests, chiefly of pine, with some oak, promise a good supply of cheap timber for some years to come. Coal may be readily purchased at a reasonable cost from the bituminous fields of Arkansas, Oklahoma, or Texas.

Although the region is not thickly settled, and the nearest towns are small, the experience of the lumber companies indicates that an ample supply of labor (chiefly white) will be available; indeed, the lumber-camps themselves may be an immediate source of supply. In this connection an obvious, and possibly serious difficulty may be mentioned; namely, the liability of the loss of diamonds through theft by the laborers. With the class of labor employed at the South African mines, a system of detention in compounds, thorough physical examination for hidden diamonds, and other methods for the prevention of theft or the recovery of stolen stones, can be carried out; but in the United States it may be impossible to employ, at least in a thorough-going manner, safeguards of this character. Up to the present time, a small force of picked men having been employed in the preliminary operations, and about 300 diamonds

having been found, there is little or no ground for the belief that any serious loss of this kind has occurred. But work on a greater scale, involving the employment of a large number of laborers of less trustworthy character, with increased difficulties of adequate supervision, will augment this risk, the prevention of which will be a serious problem.

Transportation facilities for coal, machinery and other supplies are furnished by two short branches from theIron Mountain Railroad. One (a private lumber road) leaves the main line at Prescott, and extends 26 miles to Nathan, about six miles from Murfreesboro, while the other runs from Gurdon on the main line about 30 miles to Pike City, distant about 10 miles from Murfreesboro. Only very rough roads now connect these terminals with the diamond bearing locality; but these roads will be improved and a new railroad to Murfreesboro is in process of construction and will be completed within one or two years.

Factors to be Determined.—Up to the present time about 300 diamonds have been discovered within the igneous area, while none has certainly been found outside of it, even in the immediate vicinity. All the stones have been found on the surface, except two, which were in the concentrates derived from washing large amounts of the green ground, and one, which was imbedded in the green ground itself about 15 ft. beneath the surface. My careful examination of this last specimen, confirmed by Dr. R. W. Raymond, leaves no doubt that the diamond is actually in place in the rock and was not inserted in the specimen. Consequently it furnishes a definite proof that the peridotite is the source of the diamonds, and that all the stones so far discovered have been derived from it. It would be well, however, to have this single piece of evidence corroborated by similar specimens. With regard to the quantitative relations of the diamonds to the inclosing rock, about 200 carats have been found on or immediately beneath the surface, where presumably there has been considerable concentration of the stones. From the nature of the deposit, the average yield per ton can be ascertained only by actual washing or other extraction from the rock on an extensive scale, commensurate with that of the contemplated commercial operations.

Additional factors of economic importance for which more extensive data are necessary, are the average size, color and quality of the stones, since these factors determine their value. From the 200 carats at present available for examination, it appears that the Arkansas locality compares very favorably with most, if not all, of those in South Africa. Although no stones larger than 6.5 carats have yet been found, the average size is fairly good. There is a large proportion of white stones, for the most part of a high grade in color, brilliancy, and freedom from flaws. Indeed, many are as fine as have ever been found. Some of the yellow ones, also, are of exceptional quality and color. As the white stones are among the finest

material found, there will be little competition with some of the African mines, where, as a rule, the quality is not of so high an order.

The method of extraction of the diamonds is of vital interest and importance. The green and the yellow grounds offer no difficulty, and are amenable to the methods used in South Africa. Indeed, in Arkansas, there is no need for prolonged exposure to the weather, since the freshly extracted material disintegrates and can be washed with ease. The amount of this easily worked material "in sight" is very large; yet it is not of indefinite extent downward, as is the "blue ground" of Kimberley; and, consequently, its extraction will form but a transient phase of future exploitation.

The economical extraction of the diamonds from the compact, and relatively fresh and hard, peridotite, underlying the "green ground" and forming the vast bulk of the mass, will involve study and experiment. But, apparently, there will be no greater difficulties than have been successfully overcome in South Africa. In view of the hard, tough, and fresh character of the peridotite which composes the highest points of the area along the northwest border, it might be thought that the material underlying the green ground would be of the same character and equally refractory; but the diamond drill shows that, at least for considerable depths, a large proportion of the underlying peridotite is far more decomposed than that which crops out at the border; is, indeed, so far altered that much of the material comes up as sludge, and no continuous cores longer than 14 in. have been obtained. Many of these cores were so soft as to be readily scratched with a knife. Probably this more compact material will disintegrate on exposure to the weather, like the South African "blue ground." If this be the case, a large proportion of the mass will not be difficult to work.

At some portions of the mass, however, as at the northwest border, and probably in depth beneath the rest of the area, fresher and much more refractory material will be encountered, the treatment of which will present practically the same problem as that of the hard portions of the African rock which do not disintegrate on exposure. While a certain amount of crushing, in order to extract the diamonds, is apparently unavoidable, this should be reduced to a minimum, on account of a high loss from breakage of the stones themselves. Several methods of treatment suggest themselves, which are at present under consideration; but the practical details, as well as the economic features, remain to be worked out and cannot be discussed here. The non-magnetic character of the diamond and its tendency to adhere to grease are obvious features which can undoubtedly be used at certain stages of the extraction for all classes of material in the Arkansas deposit, as in South Africa.

4. *Word of Warning.*—In view of the great local excitement over the

discovery of diamonds, which has extended over part of the State, and in view of the danger of the repetition here of the disastrous history of many mining camps which have undergone an unwarranted "boom," and the consequent rush, with loss of time and money by many innocent individuals, it should be distinctly understood by the public that the occurrence of diamonds near Murfreesboro is an isolated one, and that it does not resemble a mineral vein or lode in any respect. Consequently, there is not the least justification for any such claims as will undoubtedly be made by ignorant or unscrupulous parties, that "a continuation of the vein" has been struck. There can be no continuation of a vein when there is no vein.

In 1908 a new small artery of peridotite was located about two miles from the original deposit, and about three miles from Murfreesboro. Up to the present, no stones are said to have been found on the new exposure.

Should other similar igneous areas, which may possibly be diamond-bearing, be discovered elsewhere, any claims put forward for them should be received with the greatest caution. Fortunately, the characteristics of the peridotite (in which, by analogy, diamonds may be most reasonably expected to occur) are so easily recognizable by a petrographer, the localities will be presumably so isolated, and the outlines and extensions of the mass so well defined, that the report of a geologist or petrographer can surely prevent an unsuspecting or ignorant person from loss by investment in a property said to be a continuation of, or a connection with, the present deposit. Peridotites are not uncommon, but very few are diamond-bearing. Indeed, the great majority of these rocks found all over the world show no trace of diamonds. Even in South Africa, many peridotite pipes, resembling valuable ones, carry no diamonds, while in any given pipe some portions are found to be richer in diamonds than others.

As shown by J. F. Kemp,[1] many basic dikes have been found in Arkansas; but most of these differ petrographically from the Murfreesboro peridotite, and there is no reason to think that any of these, or any of the several syenitic areas of the State, is connected with diamond-bearing rocks. As has been noted above, two dikes of peridotite occur in connection with the Murfreesboro igneous area. Great stress is laid locally on these dikes, or "leads," as they are called, but without warrant, since there is no reason to think they contain diamonds, and in any case they are too small to be of economic value. From analogy with other igneous intrusions, it is probable that more dikes will be discovered in the neighborhood, radiating from the main stock; and in other localities the presence of dikes of similar rock, which could only be identified by petrographical means, would be an indication of the possible presence of a larger body of peridotite in the vicinity. If diamonds are present, they are to be looked for in the rock-mass itself, or in its products of weathering, and not only along the

J. F. Williams, *Annual Report* of the Geological Survey of Arkansas for 1890, Vol. II, pp. 392-406 (1891).

contacts, because they are integral portions of the igneous mass, and their presence is not due to the circulation of hot water and solutions along the contact between an igneous mass and the country-rock.

For the last 15 months active prospecting and developing have been going on in the search for diamonds in the Murfreesboro area. A washing-machine has been installed, and it is now proposed to make a trial test of some ten thousand or more loads to prove definitely to what extent the diamonds occur here.

Diamonds in South Africa.

De Beers.—From the 19th annual report of the De Beers Consolidated Mines, Ltd., for the fiscal year ending June 30, 1907, it appears that the receipts from diamond sales were £6,452,596 ($32,262,980); from this there was deducted a total expenditure of £3,845,356 ($19,226,780), leaving a net profit of £2,607,240 ($13,036,200). After the payment of dividends amounting to £2,550,000 ($12,750,000), the year's balance, which included something from the balance of the previous year, was £932,623 ($4,663,115).

The year's output was 9,010,686 loads as against 8,144,979 during the previous year; the figures for the amount crushed being 6,626,291 loads and 5,625,592 loads, respectively. The accumulated stock of blue ground was augmented by 2,622,477 loads and amounted in all to 9,391,603 loads at the close of the fiscal year. Development work for the year totaled 165,613 ft. of drifts, tunnels and raises, and 1703 ft. of rock and prospect-shafts.

DETAILS OF COST AND PRODUCTION, 1906-1907.

Mine.	Output of Blue Ground for Year.	Yield per Load.	Value per Carat.	Value per Load.	Cost per Load.		
					Min. (b)	Wash.	Total.
	Loads. (a)	Carat.					
De Beers	1,525,184	0.37	$15.55	$5.76	$1.21	0.74	1.95
Kimberley	578,669	....	9.87	....	1.72	0.99	2.70
Wesselton	2,104,308	0.32	10.45	3.16	0.86	0.51	1.37
Bulfontein	2,320,538	0.32	10.15	3.31	0.95	0.54	1.49
Dutoitspan	2,481,987	0.24	19.10	4.54	0.97	0.60	1.57
	9,010,686	....	....	....	....	....	....

(a) The load contains 16 cu.ft. and weighs about 1600 lb.
(b) Including the cost of handling waste rock.

Premier.—This immense mine, which has rapidly become a formidable rival of the De Beers company, made notable progress during 1907. The figures for the 12 months ending Oct. 31, 1907, are reported at 6,538,669 loads washed, with an average yield of 0.29 carats per load, and a total production of 1,889,986 carats of diamonds, valued at £1,702,630. While

these figures are enormous in amount, and compare well with those of De Beers, it is to be noted that the average value per carat of the diamonds found here (18s. 0.2d.) is only about one-fifth of that shown at the De Beers group as given in the above table. Still, the mine is one of extraordinary possibilities. The amount of blue ground in sight is estimated by the president, Mr. Cullinan, at 500,000,000 loads, or nearly eight times that at the De Beers mines, as reported in December, 1906. Mr. Cullinan, in his address, discusses the prospect of carrying on open working to a depth of 1200 or 1500 ft.; although shaft-sinking will in time become necessary. But the present open system can be continued for perhaps nine years, with the great improved plant now being installed, and may in that time deal with about 136,000,000 loads. Meanwhile, the year's record shows a marked reduction in costs of operation, which were 2s. 4.14d. per load, as against 3s. 5.7d. in the previous year, and 4s. 7.2d., or nearly double, in 1903. In the months that have passed since October, the cost have been further reduced to 1s. 9.31d. in January, 1908. This steady diminution of expense is attributed by the president to improved modes of treatment, on a large scale, with the most carefully studied methods and the best apparatus, and he predicts that it may be even further lowered.

The enormous Cullinan diamond (the Edward VII as it is to be known) found at this mine in 1905, has been presented as a gift from the people of the Transvaal colony to King Edward VII, and will be the most remarkable treasure among the crown jewels of Great Britain. This result was reached only after much discussion; but it was approved by a large majority of the colonists, and was undoubtedly the wisest disposal of such a unique object. The only alternative would have been to divide it into a number of large stones, thereby destroying its extraordinary individuality. The cutting is being done at Amsterdam, with the most elaborate precautions at every step, the stone having already undergone the process of cleavage. It has been cleaved into three large pieces. The largest piece is of such size that a dop nearly eight inches across and a polishing wheel fourteen inches in diameter are required to polish it. It will afford a brilliant of between 550 and 600 carats. Nothing will be done with the others until this is cut, as the greatest diamond known will be cut from one of the smaller pieces even if an accident occurs to the largest.

Other African Mines.—In 1905, the Roberts Victor Diamond Mine was opened at Boshof, Orange River Colony, and since that time the "yellow ground" has yielded many nodular masses of eclogite, a rock composed of garnet and an emerald-green pyroxene with cyanite. Much interest attaches to these occurrences, as similar masses of eclogite have been found at the Newlands mine in South Africa, and also at Ruby Hill, near

Bingara, New South Wales; and diamonds have been reported as occasionally found in them at both localities, certainly at the former.

It is now stated by G. S. Corstorphine that eight diamonds, one or two being well-formed octahedrons and weighing $\frac{1}{5}$ or $\frac{1}{6}$ of a carat, have been found in one of these nodules at Boshof. There has been considerable discussion as to the origin of these eclogite boulders, as they are termed; but, in Mr. Corstorphine's opinion, they are concretionary nodules, formed either by segregation or differentiation in the original magma, and hence are to be compared with the olivine nodules that appear in certain localities. The same view is strongly urged by A. W. Voit, in an important paper presented to the South African Geological Society in July last, on "Kimberlite dikes and pipes."[1] On the other hand, Mr. Johnson, in a communication to the same Society,[2] argues for a different origin of these eclogite masses. He describes in detail several types of them, some of which present a conspicuous banded, but not concentric structure. They "contain in proportionate abundance all the characteristic minerals of the eruptive diamond-bearing breccia"; and hence he is led to the view that they represent residual and more resistant portions of a rock that has gone to make up the ordinary kimberlite, and that this was the real home of the diamonds. The so-called boulders are often tabular in form and banded in structure, suggesting an origin from a deep-seated rock formation rather than from an igneous magma. When seen in place, they are sharply differentiated from the matrix, with no suggestion of segregation. Their surfaces, when taken out, are seen to be worn and smoothed, as though by attrition, the inclosed garnets being cut through and worn down. The banded structure is frequent, but it is linear and never concentric. Mr. Johnson therefore regards them as true boulders, broken from underlying rock and worn by attrition in the pipe.

Vaal River Diamonds.—An important decision by Dr. Hans Merensky has lately appeared in the *South African Mining Journal* (April 4, 1908), as to the origin of the diamonds found in the valley of the Vaal river. These are widely distributed in gravels and placers, which became noted as the "river diggings" before the discovery of the great "pipes" at Kimberley, and which are still extensively worked and yield diamonds of fine quality. They have been discussed a good deal of late, especially by T. Lane Carter, in 1903, and by Gardner S. Williams of the De Beers mines.[3] The general opinion has been that the Vaal diamonds must be derived from pipes similar to those of Kimberley, but not yet located; and Mr. Carter announced the actual discovery of such a pipe, and its partial exploitation. Dr. Merensky, however, presents a different view. He lays emphasis upon the well-known fact that the Vaal diamonds have an

[1] *Trans.* Geol. Soc. of South Africa, Vol. X, 1907 (read July 22, 1907) pp. 69-79.
[2] *Ibid.*, Vol. X. 1907 (read Aug. 12, 1907), pp. 112-114.
[3] *Eng and Min. Journ.*, Sept. 5, 1903; "Min. Res. of the U. S.," 1903, p. 918, and 1904.

aspect quite distinct from those derived from any of the "pipe" mines, and that the occasional pipes and fissures found by extensive search in the region have proved to contain but few and small diamonds, all of which are strictly of what he terms the "kimberlite type," different from that of the placers. Some other source must, he thinks, be sought for the latter.

The Vaal diamonds occur in deposits of debris from the disintegration of diabase and related rocks, concentrated here and there by stream action, but showing no signs of much transportation. They are associated with various forms of quartz, jasper, agate, etc., and with amygdules from diabasic rocks. The diggers, moreover, state that they find diamonds on steep kopjes, where river deposit is out of the question, and also that old diggings yield a fresh supply after a few years. In view of all these circumstances, Dr. Merensky believes that the Vaal diamonds have their source in the diabase itself, which is widely distributed over the entire area where they occur, and that they are still being released by the weathering of the rock. These diabasic rocks are referred by the South African geologists to the Ventersdorp system, which is earlier than the Karroo beds, with which all the "pipes" of kimberlite are associated; and hence the two modes of occurrence would differ not only in the nature of the enclosing rock, but also in its geological age.

This view, which Dr. Merensky presents as a tentative one, requiring further investigation, derives additional interest from the discovery of diamonds in a dolerite dike, in the Inverell district of New South Wales, reported in 1904. [1] It is true that the rocks are different in the two cases, but both are igneous; and the suggestion afforded by these indications in both Africa and Australia is that diamonds may prove to be a not very infrequent content, though rarely abundant, in basic igneous rocks of several kinds.

Diamond Mining in Brazil.

Much attention has been given recently to the diamond fields of Brazil and the possibilities of greatly enlarged production in that country. The area of diamond-bearing deposits is very extensive, but the obstacles and limitations have been so great that comparatively little has been done. These have consisted in two things, viz., inacessibility and lack of transportation on the one hand, and on the other, crude, primitive methods and lack of machinery. It appears that both of these conditions are soon to be changed, and then may come a very large development of diamond production in Brazil. American capital is becoming interested, and has already gained control of much of the richest country in the Diamantina region. The work already begun has led to a demand for more and better

[1] Min. Res. of the U. S., 1904.

means of transportation; and American engineers and American wagons have been called into the field. Railroad connections are to be opened to Diamantina, the mining center of the State of Minas Geraes, which has heretofore been reached only by rough wagon-roads or on mules or horses; and the superiority of American-built wagons, introduced by some of the companies, has already attracted much attention from the authorities of the State. Dredging machinery has been installed along the Jequitonhonia river; and with it will come a revolution in the diamond production of the region that, it is claimed, may be felt throughout the world. The stones will not average as high as the African in size and the yield may not be a permanent one, as it is all from new gravels.

Heretofore, the Brazilian output has gone to Europe; but with the new conditions above noted, it is now turning to America. The diamond exports from Brazil, so far as reported, at least, were $310,000 in 1906,—double the value of 1905. These figures, however, are believed to be much below the actual output. They include also the carbons, or carbonados, from the State of Bahia, which are in great and increasing demand, at high prices.

It is of interest in this connection to note that Dr. J. C. Branner, of Leland Stanford University, and formerly of the Brazilian and Arkansas geological surveys, spent some six months in Brazil in 1907-1908 and found in his explorations that the carbonade region covered at least one-third more area than had been supposed, and, as the new district was unworked, this means that the world's supply will be nearly doubled. Dr. Branner also found that the area of the old diamond-fields is much more extensive than had been supposed.

Although the total imports of precious stones into the United States for January, February, March and April, 1908, amounted to but $1,540,000, one-third of this sum ($500,000) represented the carbon, carbonado, or industrial bort, as it is known, from the province of Bahia, Brazil. This large importation was owing to the very high price of this material at present, viz., from $70 to $80 per carat of 205 milligrams,[1] or at the rate of $12,000 per oz.; while the crystalline bort used to pulverize into diamond dust, and also employed to some extent in marble-sawing, sells from $1 to $2 per carat, or only $\frac{1}{40}$ to $\frac{1}{80}$ of the price of the other much more highly prized amorphous material.

Caution is needed, however, in regard to the subject of Brazilian mining enterprises, as a number of projects have been presented in New York for the purpose of developing tracts of diamond-bearing earth in that country. It is claimed by those who are competent to judge, that many of these are

[1] The proposed metric carat of 200 milligrams first proposed by G. F. Kunz at the International Congress of weights and measures held at the World's Columbian Exposition, Chicago, 1903, is being agitated and adopted in some countries. For full discussion, see The Book of the Pearl, by George F. Kunz and Charles Hugh Stevenson, 565 pages, 100 plates, The Century Company, New York, 1908.

of problematical nature; and that with others the titles can be perfected only with the greatest difficulty, as the mines have passed through the hands of many owners, and the heirs, legitimate and illegitimate, frequently number more than a hundred, so that it is almost impossible to gain a clear title in a reasonable time, if indeed at all. It would seem that a process of condemnation by the Brazilian Government might be legislated, thus aiding in the development of an industry and preventing the frequently recurring losses to so many who needlessly suffer failure.

Other Precious Stones.

Sapphire.—The Queensland sapphire fields are in the Anakie district, Central Queensland, on the Central Queensland Railway. They are all found in the alluvial deposits and are "panned" or dry cradled in the absence of water. The production has been as follows, the value per ounce being stated in brackets: 1903, £7000 (15s.); 1904, £10, 700 (15s.); 1905, £5,255 (15s. to £1); 1906, £18,110 (£ 1.4); 1907, £30,000 (£1.4); total to date, £64,065 (15s. to £1.4). Unfortunately little of the material is a transparent blue; it is generally more or less dark, and has frequently been used for stones for seal rings. The population of the Anakie district now amounts to 1100, in five places, the principal of which are Sapphire town, Policeman creek, New Rush, and Tomahawk creek. Most of the material is sent to Germany and the United States.

Burmese Tourmalines.—C. S. George, deputy commissioner of the ruby-mines district in Burma, gives an account of the manner of working for tourmalines, of which the red variety, rubellite, the Chinese ruby, is found in that region and has always been greatly prized. It occurs in distinct crystals in the cavities in granitic rocks—probably pegmatite veins similar to those in which the same mineral is found in Maine, Connecticut, and California. The method of mining is to sink a vertical shaft 4 or 5 ft. square, into the upper decomposed portion of such a vein; this shaft is known as a *twinlon* mine, the same name that is applied by the natives to similar shafts sunk into the ruby-bearing gravel in the search for that gem. From this vertical shaft, the owner is usually granted the right to extend horizontal galleries underground to a radius of 30 ft. Of course, when any shaft has proved productive, others are sunk along what is supposed to be the line of the vein, but frequently without success. The best tourmalines are never found at a less depth than 50 ft., the average depth being 60 or 70 ft. The material excavated is brought up from the shafts in small bundles, elevated by extremely long bamboos set on a pivot and counterweighted; the tourmaline is then sorted out by hand.

Seven years ago, an important find of red tourmaline, or rubellite, was made at Hpai Baing (Milaunggon), where, in former times, the Chinese had

worked; a more recent one is at Htauka, between Milaunggon and Sanka. Early in 1905, another rich vein was discovered near Sanka village.

Australian Topaz.—Among the Australian gems brought to light in recent years are topazes weighing from ½ to 8 oz. each, blue, green and white in color. These have been found in various places on the Australian continent, notably near Torrington, New South Wales, where they were found in place with crystals of quartz; and at Stanhawk, Queensland, where they appeared in the form of water-worn pebbles, the crystalline faces being entirely obliterated; but they are of a beautiful deep-blue color, very similar to the large, water-worn crystals found at Oban in New South Wales, where they adhered to crystals of quartz, associated with wolframite in a matrix of clay at Heffanan's Lease. Some of the individual crystals measured 2 in. in hight, and 2½ in. in diameter. They were invariably striated and broken across. At the Gulf, near Emmaville, New South Wales, beryl has been found imbedded in and associated with quartz, one crystal being bluish green, over 2 in. in length and nearly 1 in. in diameter. It has also been found at Packenham, Victoria.[1]

Opal.—Among the many unusual forms of precious opal found near White Cliffs, New South Wales, there were specimens that were pseudomorphs of a mineral, either glauberite, gaylussite or gypsum; these were groups of crystals, which, from their peculiar forms, were termed "pineapples;" frequently, however, they consisted entirely of good opal material. There were also other pseudomorphs after wood, the woody structure being still visible, although replaced by precious opal. Besides these, were pseudomorphs from shells and from reptilian bones and teeth, especially saurian vertebrae; one of these measured 3 in. across. In addition to this, there has been found, within the past few years, a deposit of black opal of wondrous, splendid fire, the reds, greens, and blues predominating, usually of from ⅛ in. in thickness to 1 in. or more. But the largest part of the deposit was found to be either entirely black, or else with a decidedly smoky tint. This has given us an opal possessing a velvet black groundwork through which a brilliant display of fire is apparent. The best deposit of black opal was found at Lightning Ridge, New South Wales, at a depth of 30 ft. One mass discovered weighed 78 oz., and was valued at more than £1000. These are the first localities from which have been derived these dark stones, to which superstition might assign the property of bringing good fortune to the owner, in contradistinction to the ill-luck attributed to the more usual varieties by Sir Walter Scott and others. These black opals have been cut into gems weighing from ¼ to one carat each, and have formed a valuable addition to the gem family.

Jade.—During 1906 and the early part of 1907 there were discovered some immense masses of jade (nephrite) at the western end of New Zealand.

[1] C. Anderson, Mineralogist, Records of Australian Museum, Sydney, pp. 69-79.

These were great, water-worn boulders, exceeding in size any specimens ever found in Australasia; the largest two pieces weighed 4228 and 2968 lb. respectively. They were brought to the attention of Tiffany & Co., who imported them and, through the characteristic generosity of J. Pierpont Morgan, Esq., they were presented to the American Museum of Natural History. Here they are shown in connection with the immense piece which I discovered in Silesia in April, 1899, weighing 4712 lb. This specimen was found in the bed of an old serpentine quarry used for road material; it was of much greater toughness than the material in the quarry and the quarrymen had therefore passed it by. This mass is of the same variety as nephrite and there is enough material in it to make all the archaeological jade objects on the European continent five times over. A special interest was associated with this find because it was long believed that all the nephrite and jadeite objects found in Europe, in the Lake dwellings and elsewhere, had an Asiatic origin and had been brought to those places in the migration of the early races.

Jadeolite.—Several handsome and interesting green minerals, more or less resembling jade, and suitable for similar ornamental uses, have lately been brought into notice. One of these is a remarkable, deep-green, chromiferous syenite, found at the jadeite mine at Bhamo, Burma, and frequently cut into gem-stones. I have suggested the name "jadeolite" for this beautiful and interesting material.

Verdite.—During 1907 there was found on the south bank of the North Kaap river, in South Africa, two miles above Kaap station, another somewhat similar ornamental stone that has the deep green color of the chromiferous syenite, just mentioned as found in connection with the jade of Burma. It has a hardness of about three, and is susceptible of a high polish; the color is a rich chrome green and the stone contains a chrome-muscovite and some argillaceous material. Occasionally it has yellow or red spots. This stone is obtained in blocks weighing one ton or more and it is now sold in England at the price of £18 to £40 per ton. The name verdite has been suggested for it.

Style and Fashion—Commercial Conditions.

In regard to matters of style and fashion in jewelry, the demand for green stones has been unprecedented. The emerald, tourmaline, peridot, the rich green jade, amazon-stone, New Zealand jade or nephrite, and even malachite, have been more or less sought. Jade of every kind has been brought into the United States and worked and carved into an endless variety of forms. There is the rich carving of the Chinese, and the jade is frequently mounted in designs that are Chinese in character and are often executed by skilled Chinese jewelers.

During the latter part of 1907 the blue stones have been much in evidence, and the sapphire has once more come into positive favor, resulting in the use of sapphires of all shades and qualities, and the introduction of lapis-lazuli as well as the blue azurite, found in New Mexico. More attention is paid at the present time than ever before to the harmonious agreement in color with her costume of the jewels which a lady may wear, so that, while not suitable for rich decoration, many of the minor jewels are frequently worn in the form of beads, either round, facetted, ellipse-shaped, or flat antique in form. Such chains are sometimes varied by the alternation of a pearl or a piece of crystal cut flat, or else every second bead is followed by a pearl. Then again such combinations as amethyst beads alternated with a rich green jade from Burma, either plain or carved, are fashionable.

During 1907, what are known as calibre cut stones were extensively sold. These stones are usually brilliant or table-cut and range from a millimeter or slightly larger to greater sizes; they are generally square, or at least have two parallel edges so that they may be slipped into a continuous setting without any gold or other metal mounting between them, and the edges of the setting hold the stones securely. When the jewels are square, oblong, round or oval, they are mitered on two sides, the metal edges holding them in place. By this means it is possible to make unusually interesting and beautiful settings, such as continuous bands or circles of red, blue, or green, or else these same stones are alternated with diamonds, or two or three colored stones may be placed with a white diamond intervening.

From January until October, 1907, there was an unprecedented trade in precious and semi-precious stones in the United States. The importations were never greater, and the material was frequently such as commanded the most expensive and often extravagant prices. The demand was for every form of precious stone, and pearls in every quality. Emeralds from the finest to the poorest, and sapphires, the gem last to feel the dictates of fashion, were imported in greater quantities than ever before. At the same time diamonds, from the largest cut stones to the minutest, were in equal demand; down to the small stones of single brilliant cutting, generally of the whitest material, numbering 100 to 250 to the carat, in other words, from 15,000 to 37,000 to the ounce.

November and December saw a great falling-off in the imports of diamonds and other precious stones, and this continued through the winter and spring. The total imports of precious stones, cut and uncut, in 1908 for the months from January to May inclusive, amounted to but $2,000,000 in comparison with $12,000,000 worth imported in the same period of 1907. As elsewhere mentioned, $500,000 of this sum represented the imports of carbonado, or what is known as industrial bort; hence the

value of the precious stones imported was really only $1,500,000. This great decrease was due to several causes, the principal one being that the unprecedented imports of 1907 and the sudden cessation of speculative activity resulting from the panicky times induced smaller imports of jewels and caused retrenchment on the part of jewelers, whether cutters or retailers, who preferred to diminish their stocks until times were more propitious.

QUICKSILVER.

The production of quicksilver in the United States in 1907 amounted to 20,932 flasks, of 75 lb. each, with a total value of $780,506, as compared with 25,309 flasks, worth $1,035,138, produced in 1906. The year thus recorded another step in the downward progress that has characterized the quicksilver industry in the United States for the last four years. As heretofore, California contributed much the greater part of the total, although the output from that State was less in 1907 than it had been for the last 50 years. The Texas deposits have not yet gained much eminence, and the anticipated productivity of the one Oregon locality has not yet materialized. The production of the single Utah mine came to a close in 1907. Newly discovered deposits have been reported from Nevada.

STATISTICS OF QUICKSILVER IN THE UNITED STATES.

Year.	Production.					Exports.			Imports.	
	Calif. (*a*) Flasks.	Texas. Flasks.	Others. Flasks.	Total. Metric Tons	Value.	Flasks	Metric Tons.	Value.	Pounds.	Value.
1896...	30,765	....	....	1,061	$1,075,449	19,944	692	$618 437	....	$2,037
1897....	26,648	....	....	919	993,445	13,173	475	394,540	45.539	20,147
1898....	31,092	(*b*)	153	1,077	1,194,746	12,830	445	440,587	81	51
1899....	29,454	261	....	1,025	1,416,790	16,518	573	609,586	131	83
1900....	26,317	1,700	233	974	1,279,436	10,702	353	425,812	2,616	1,051
1901....	26,720	2,932	75	1,031	1,382,305	11,219	389	475,609	1,441	789
1902....	29,552	5,252	....	1,208	1,515,714	13,247	459	575,099	*Nil.*	....
1903....	32,094	5,029	....	1,288	1,564,734	17,575	610	719,119	*Nil.*	....
1904....	28,876	5,336	700	(*c*)1,204	1,348,185	21,064	731	841,108	212	160
1905....	24,655	5,000	1,050	1,045	1,217,652	13,460	458	497,470	2,690	1,710
1906....	19,516	4,517	1,276	861	1,035,138	6,455	220	244,299	84	50
1907....	(*d*)17,532	3,000	400	712	780,506	5,132	175	192,094	16,566	6,719

(*a*) Reported by the California State Mining Bureau. (*b*) Included in "Other States." (*c*) Estimated; the weight of the flask was changed from 76.5 lb. to 75 lb. within this year. (*d*) Figures collected by *The Mineral Industry.*

QUICKSILVER PRODUCTION OF THE WORLD.
(Metric tons.)

Year.	Austria.	Hungary.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1896..................	564	1	186	218	491	1,524	1,036	4,020
1897..................	532	1	192	294	616	1,728	965	4,328
1898..................	491	7	173	353	362	1,691	1,058	4,135
1899..................	536	27	205	324	360	1,357	993	3,802
1900..................	510	32	260	124	304	1,095	983	3,308
1901..................	525	33	278	128	368	754	1,031	3,117
1902..................	511	45	259	191	416	1,425	1,208	4,055
1903..................	523	44	314	188	362	968	1,288	3,687
1904..................	536	45	357	(*e*) 190	393	1,130	1,192	3,843
1905..................	519	36	370	(*e*) 190	318	853	1,045	3,331
1906..................	526	50	418	(*e*) 200	210	1,568	963	3,935
1907..................	....	....	423	....	130	....	712	....

(*e*) Estimated.

Quicksilver Mining in the United States.

California (By Charles G. Yale).—The condition of the quicksilver mining industry in California is not at all encouraging for the opening of new mines, or increased development upon old ones, unless the deposits have been proved. The output is lessening year by year. In nearly all the old mines the ore is diminishing in quantity and grade, showing a virtual exhaustion of the profitable deposits. A number of properties have been closed down, and hardly any are coming into productiveness to take their places. Of course in mining these deposits there is always the chance that good ore may be met again, so this encourages the operators to keep on work even though little profit is being made. There is in this feature also some encouragement for men to open new prospects and test their ores with small benches of retorts. But unless the grade of ore is reasonably good there is little money in quicksilver mining under present conditions, with the exception of those cases where extensive reduction works are already built to handle ores carrying below 1 per cent. of metal.

In 1905 California produced 24,655 flasks of quicksilver, worth $886,081. In 1906 the yield of the State was 19,516 flasks valued at $712,334. A still further reduction occurred in 1907, when the production amounted to 17,532 flasks, valued at $653,126, or the lowest output in 50 years.

The most productive mine in the State, the New Idria of San Benito county, has been increasing its output since September, another new furnace having been put in commission. The product is about 800 flasks per month. This mine and the Napa Consolidated, under the same management and ownership, yielded over 10,000 flasks in 1907. The Napa produced at the rate of 200 flasks per month, which is better than in 1906. The New Idria also materially increased its output as compared with 1906. Most of the other mines of the State showed rather a marked falling off in yield, though a few held their own.

Aside from these about the only mine in the State which shows improvement worthy of note is the Helen, of Lake county. This mine has been worked for some years in a small way, but has never made much production. Of late, however, owing to vigorous and intelligent development, it has been turned from a good prospect into a good mine, and is expected to cut quite a figure in the market in 1908. A good furnace has been erected at the mine.

Nevada.—A cinnabar deposit, upon which development began early in 1907, gives promise of developing into a producer. The owners are endeavoring to equip the mine with a plant, including a smelter. The mine comprises an area of 480 acres, situated in the Ione, or Union mining district, about 90 miles northwest of Tonopah. A lode yielding mercury has been

traced by outcrops through the whole length of the property. Prospecting shafts are said to show that the lode is a well-defined contact deposit.

Oregon (By William B. Dennis).—The reduction plant at Black Butte mine, Lane county, was not operated during 1907. The year was consumed in installing a new plant. Some underground work was done, chiefly preparations for stoping. The mine itself had been extensively developed during the years previous and large ore reserves established. The property includes about 2000 acres of timbered lands and lies in the foot-hills of the Calipooia range, at the southern extremity of the coast fork of the Willamette valley.

Several clearly defined lodes of cinnabar-bearing rock have been proved. The ore occurs chiefly along strongly defined fracture planes, which show marked persistency in lineal and vertical extension. Along these fractures the ore-bearing solutions have penetrated the walls laterally for a great distance, forming wide orebodies.

One main central fracture, outcropping along the apex, virtually cuts the mountain in two longitudinally. This fracture has been traced in a continuous line for two miles, showing ore-bearing rock at every exposure. Considering the system of parallel and tangential fractures as a whole, it may be said that the entire region forms one huge lode of cinnabar-bearing rock. The richer portions of the lode usually lie along the walls of the fractures, forming payable orebodies ranging from 8 to 80 ft. wide, and carrying an average of from 0.25 per cent. to 0.95 per cent. mercury.

The rocks of the region are of volcanic origin, ash-rocks being the most abundant. The extent of alteration is so great that it is difficult to determine the original composition, but geologists who have examined the district have generally agreed that they were originally andesites.

The problem in the operation of the Black Butte mine has been to treat the large low-grade orebodies at a profit. Former owners had erected a 40-ton Scott-Hunter furnace of the California type. The operation of this plant proved unprofitable, as it saved only about 33 per cent. of the metal. The fir wood of the district used for fuel produced an enormous amount of soot, which retarded condensation and made a second treatment necessary in order to free the entangled quicksilver from the soot.

In 1906, I erected an experimental furnace on new and original lines, and conducted a series of experiments covering the period of a year. At the end of this time patents were taken out. Perfect combustion was accomplished, soot entirely dispensed with, and a high percentage of recovery secured. The roasting period was cut down from 24 to 6 hours, thereby greatly increasing the furnace capacity per unit of hearth area. Along the lines of the new process a plant, including alterations to portions of the old plant, was erected during 1907, and on Feb. 1, 1908, the fires were

lighted. The ultimate success of the new plant still remains to be demonstrated but in the early part of March the outlook was favorable.

The new plant is equipped with a hydro-electric power plant which supplies current for lighting the buildings and mine, and power for the operation of the crushers and the two 70-in. Sturtevant exhaust fans. These fans furnish artificial draft for the furnace, as well as for the wood-gas producer, which is also one of the new features of the plant. A Sturtevant coarse ore steel breaker and a Gates fine ore crusher have been installed and bin capacity for 700 tons has been provided. The ore is delivered to the furnace by an aerial tram 3000 ft. long. The condensing plant has been erected along new lines and forms one of the novel features of the plant. The new dryer, constructed of concrete, steel and brick, has a net capacity of 125 tons.

The management declines to furnish any estimates of the capacity of the plant, probable output, or treatment costs, preferring to wait until the end of the year for the finished record. Should it prove successful Oregon will become one of the regular producers of quicksilver.

Quicksilver Mining in Foreign Countries.

Mexico.—The failure which has thus far followed almost all attempts to produce quicksilver in Mexico is believed to be due not so much to lack of sufficiently rich ores, as to the primitive and wasteful smelting methods that have usually been employed in recovering the metal. The quicksilver deposits in two different parts of Mexico have been described in a recent publication.[1]

The Chiquilistlán deposits occur between the villages of Tapalpa and Chiquilistlán, in the state of Jalisco, 118 miles southwest of Guadalajara. The nearest railway stations are distant about a day's journey on horseback. In 1843, a company began working the deposits, but suspended operations in the following year; since then various isolated attempts have been made to work some of the mines, but now a new company is about to reopen the workings on a fairly large scale. They occur at the foot of a sierra, which ranges southeast and northwestward. The ores are chiefly found at the meeting-points of the fissures which traverse in all directions the hard, gray, Middle Cretaceous limestones of the district. The fissures are mineralized with cinnabar, azurite, malachite, chalcopyrite and limonite, associated with calcite and a little gypsum. The average metallic mercury contained in the ores is 0.80 per cent. Occasionally the deposits assume the form of strings of pockets, simulating true beds by their horizontal extension. Much exploration work remains to be done,

[1] Descripcion de los Criaderos de Mercurio de Chiquilistlán (Jalisco). By Juan D. Villarello. Memorias de la Sociedad cientifica "Antonio Alzate," 1904, vol. xx., pp. 389 397.

but the prospects which await judiciously conducted mining operations are favorable.

The Guadalupana mercury mine, in San Luis Potosi,[1] is said to be one of the richest mercury deposits worked in Mexico. The mine is about 19 miles by a good road from the Moctezuma station of the Mexican National Railway, and this distance will presently be diminished, by means of a new short cut, by fully five miles. The smelting works are about 1¼ miles distant from the town of Moctezuma and 11¼ miles from the mine. The works could not be set up in the neighborhood of the mine itself owing to the deficiency of water supply. The deposit occurs among highly folded calcareous slates (or marls), and forms a belt from 20 to 33 ft. broad, having much the appearance of a vein that strikes generally north and south, with an average dip of 55 deg. westward. But there is no precise evidence of the existence of a fissure, of which such a vein might be said to form the filling. The ore consists entirely of cinnabar, which either impregnates the marls or shoots out in small leaders among them, and is associated with calcite and a certain amount of finely crystalline gypsum. The absence of all other sulphides but those of mercury is characteristic of this deposit, and is another circumstance which favors the working, even of low-grade ore. In 1904 a depth of nearly 500 ft. had been reached, and the deposit had been followed for over 650 ft. along the strike. So far as could be ascertained from exploration work there was no diminution in the richness of the ore. The average weekly output is 10 tons, assaying 10 per cent. of metallic mercury of which at least 10 per cent. is lost in the process of treatment.

Russia.—The quicksilver output of Russia, according to figures of the Auerbach company, the only producer in Russia, has suffered a practically continuous decline in output since the beginning of 1906. The production in January, 1906, was 1178 poods, while in January, 1907, it was only 780 poods. The output for July, 1906 and 1907, was 1000 and 598 poods respectively. The production for the first seven months of 1907 was 5177 poods, as against 7817 poods in the corresponding period of 1906. The total production for 1907 was 8055 poods, as compared with 12,848 poods in 1906.

Spain.—The financial condition of the quicksilver mine of Almaden in Spain has not been satisfactory of late and a commission was instituted to inquire into the situation. The following were the conclusions of the commission. Above the twelfth level there are blocked out about 400,000 tons of mineral, corresponding to about 1,000,000 flasks of mercury representing supplies for about 25 years. Below this level about 800,000 flasks are available. Recent investigations have demonstrated the probable continuity of the deposits toward the west, as well as in the old

[1] Alberto Capilla, *op. cit.*, pp. 423-27.

workings. The reforms recommended as necessary for placing the mines on a sound footing are as follows: The adoption of machine drills, modern explosives, artificial ventilation, renewing equipment in the shafts and workings, bettering the means of surface transportation, installation of a new group of Cermak-Spirek furnaces, abandoning the Bustamente furnaces, reducing the working and administrative force, and the appointment of an engineering chief at the mines, as well as four other engineers to be subject to the orders of the directors.

A British concern, Oviedo Mercury Mines, owns deposits of cinnabar at Mièrès, in the province of Oviedo, its property having an area of 802,712 square meters. The mines have been in operation since 1846, with unfailing financial success. The veins have a width of 10 to 40 m., and are developed by two shafts, one of which is 140 and the other 126 m. deep. The company's smelting plant is thoroughly modern, and includes 10 furnaces, of which four are of the Idria type, one is a tubular furnace, three are retort furnaces and two are of the Gascue-Rodriguez pattern. Arsenious oxide is recovered as a by-product. The average monthly output is 4000 kg. of mercury and 15,000 kg. of arsenious oxide, having a combined value of 38,000 fr.; the total cost of operating is about 15,000 fr. per month.

Transvaal.—Fairly extensive quicksilver deposits have been discovered in the Transvaal, which promise to be capable of supplying the needs of the gold-mining interests. Quicksilver claims in the eastern Transvaal and near Hector Spruit on the Delagoa Bay line, have been worked to a small extent. The most promising deposits, however, belong to the Campbells Mercury, Ltd., and are situated about 18 miles from Hector Spruit.

Market and Trade.

San Francisco.—Prices for quicksilver improved in the last few months of 1907 owing to the fact that the California operators ran short of stocks, and no exports were made. At the prices prevailing until late in 1907 it would not pay to export to China and Japan to meet the competition of the Rothschilds' quicksilver sent to those countries from Austria and Spain. The cost of mining is higher than formerly in California, so there is no profit in exporting to the Orient, especially as the production is now about equal to the domestic consumption. When this was not the case the producers here had to seek an outside market. As the Rothschilds make the price of quicksilver, when the California men were shipping to China and Japan, the price was kept down. When they stopped exporting, in September, the price went up.

QUOTATIONS FOR QUICKSILVER IN LARGE LOTS.

Month.	1904. New York.	1904. San Francisco. Domestic.	1904. San Francisco. Export.	1905. New York.	1905. San Francisco. Domestic.	1905. San Francisco. Export.	1906. New York.	1906. San Francisco. Domestic.	1906. San Francisco. Export.	1907. New York.	1907. San Francisco. Domestic.	1907. San Francisco. Export.
Jan...	$45.75	$44.50	$41.50	$40.00	$40.42	$39.17	$40.25	$39.13	$37.63	$41.25	$39.50	$37.50
Feb...	45.50	44.50	41.50	40.00	38.88	37.63	41.00	39.50	38.00	41.25	39.00	37.37
Mar..	45.40	44.50	41.50	38.95	38.15	36.90	41.00	39.50	38.00	41.00	38.50	37.25
Apr...	45.00	44.50	41.50	38.25	38.00	36.70	41.00	39.50	38.00	41.00	38.50	37.25
May..	44.94	44.19	41.81	38.38	38.25	37.00	41.00	39.50	38.00	41.00	38.50	37.25
June..	44.75	43.30	42.70	38.50	37.85	36.50	41.00	39.50	38.00	41.00	38.50	25.37
July...	43.81	43.50	41.94	41.25	39.00	37.75	41.00	39.50	38.00	41.00	38.50	25.37
Aug...	43.50	43.50	41.75	40.50	39.00	37.75	41.00	39.50	38.00	40.00	38.00	36.75
Sep...	41.40	42.45	40.85	40.00	39.00	37.75	41.00	39.50	38.00	40.00	38.05	36.70
Oct...	40.00	41.75	41.75	40.00	39.00	37.75	41.00	39.50	38.00	40.50	38.19	36.50
Nov..	40.00	41.75	41.75	40.00	39.00	37.75	40.75	39.50	37.50	45.00	45.00	43.50
Dec...	40.00	42.25	41.00	40.00	39.00	37.50	40.75	39.50	37.50	45.00	45.00	43.50
Year..	$43.34	$43.39	$41.63	$39.65	$38.80	$37.52	$40.90	$39.47	$37.89	$41.50	$39.60	$38.17

London.—The quicksilver trade of Great Britain with foreign countries is shown in the accompanying table, the range in the price per flask also being shown:

BRITISH IMPORTS AND EXPORTS OF QUICKSILVER. (*a*)

Year.	Imports Flasks.	Exports Flasks.	Price. Highest. £ s. d.	Price. Lowest. £ s. d.	Year.	Exports. Flasks.	Imports. Flasks.	Price. Highest. £ s. d.	Price. Lowest £ s. d.
1907.......	39,448	29,465	8 5 0	6 15 0	1902.....	33,192	19,519	8 17 6	8 14 6
1906.......	38,823	27,712	7 7 6	6 17 0	1901.....	35,341	26,863	9 2 6	8 17 6
1905.......	34,034	21,330	7 15 0	7 1 0	1900.....	32,725	25,869	9 12 6	9 2 6
1904.......	33,218	27,277	8 5 0	7 14 0	1899.....	51,696	32,239	9 12 6	7 15 0
1903.......	34,886	18,846	8 15 0	8 5 0	1898.....	54,563	34,014	7 15 0	6 16 0

(*a*) Reported by Alex. S. Pickering, London.

The Treatment of Quicksilver Ores.

The method employed for reducing quicksilver ore depends mainly on the character of the ore and of natural conditions. The processes comprise two main steps: Volatilization, freeing the mercury from associated elements; and condensation, reducing the mercurial vapors to liquid form and collecting this product. This treatment is accomplished either in closed retorts or in furnaces. Briefly stated, "retort reduction is essentially a simple distillation in which the mercury vapor is kept separate from the flame and smoke of the fire box," while in furnace reduction "all the gases enter the condensers together". Retorts are suited only for treating high-grade material, or for limited operations. In this country fine-ore furnaces, of the shaft type, are now most commonly used.

In California, where the ore is regarded as generally decreasing in grade both retorts and furnaces are used. More than half a dozen properties,

including the Cambria, Napa, Oceanic and Socrates, operate 50-ton Scott furnaces, while other properties use smaller furnaces of the same type. The new Almaden operates six intermittent furnaces, and the Great Western a Litchfield furnace.

In Texas, where the serious factor is scarcity of fuel, and, with one or two exceptions, scarcity of water also, the Scott furnace is in almost universal use. The Marfa & Mariposa use two Scott furnaces of 15-tons capacity each. The Chisos leases the 10-ton Scott furnace at the Colquitt-Tigner property, 6 miles distant, but is now building a 10-ton furnace of the same make on its own property. The Big Bend company and the Terlingua company each has a 50-ton Scott furnace operated only at intervals. In 1906 the Texas-Almaden company completed a 20-ton Scott furnace of special improved design. The condensers are constructed with close lining of a brick of special quality to minimize the escape of quicksilver, and are maintained at a steady temperature by means of an 18-in. ventilator over each.

In Utah, at Mercur, on the property of the Sacramento Gold Mining Company, high-grade ore is treated at the mine in six retorts (each taking four pans).

In the Monte Amiata district, Italy, where the ores now mined are of very low grade, the treatment has been brought to a very high degree of refinement. The Czermak-Spirek continuous, automatic, reverberatory roasting furnace is almost universally used. The Spirek shaft furnace is also in operation at several plants. The largest mines use roasting furnaces which have a capacity of 24 tons of low-grade ore and 50 of high-grade, while the medium-size furnace handles 12 tons and various smaller ones 6 to 3 tons. The Czermak-Spirek roasting furnace is also used in Idria, Austria; Almaden, Spain; Nikitovak, Russia; Vallalta, Tyrol; Taghit, Algiers, and Smyrna, in Asia Minor.

SALT.

The production of salt in the United States during 1907 showed a substantial increase over that of 1906. Michigan and New York continued in the lead as producers, these two States furnishing approximately two-thirds of the entire output. An important feature in the industry during 1907 was the completion of several new-process plants, which have materially decreased the cost of production and increased the output capacity. The accompanying table shows the production of salt in the United States since 1900. Of the total amount of salt mined, the chief product was dry salt, although a large quantity was used as brine in chemical works without concentration.

PRODUCTION OF SALT IN THE UNITED STATES. (*a*)
(In barrels of 280 lb.)

Year.	California.	Illinois.	Kansas.	Louisiana.	Michigan (*c*)	Nevada.	New York (*c*)	Ohio, W. Virginia and Pa. (*b*)	Utah.	Other States.	Total Barrels.
1900..	621,857	(*d*)	2,233,878	(*d*)	7,210,621	(*d*)	7,897,071	1,669,156	249,128	987,631	20,869,342
1901..	601,659	99,700	2,087,791	451.430	7,729,641	13,781	7,286,320	1,385,257	334,484	569,092	20,566,661
1902..	682,680	90,009	2,158,486	399,163	8,131,781	14,829	8,523,389	2,318,579	417,501	1,112,824	23,849,221
1903..	629,701	(*d*)	1,555,934	568,936	4,297,542	(*d*)	8,170,648	3,043,135	212,995	489,238	18,968,089
1904..	821,557	(*d*)	2,161,819	1,095,850	5,425,904	(*d*)	8,600,656	3,030,829	253,829	639,558	22,030,002
1905..	664.099	(*d*)	2,098,585	1 055,186	9,492,173	(*d*)	8,359,121	2,728,709	177,342	1,390,907	25,966,122
1906..	806,788	(*d*)	2,198,837	1,179 528	9,936,802	11,249	8,978,630	3,436,840	262,212	1,361,494	28,207,743
1907.	626,693	(*d*)	2,667,459	1,157,621	10,786,630	6,459	9,657,543	4,007,390	345,557	464,143	29,719,493

(*a*) Statistics of the U. S. Geological Survey except for New York during 1906 and 1907, which were taken from report of the State Geologist. (*b*) The production of Pennsylvania in 1906 and 1907 is included in "Other States." (*c*) Includes brine used in manufacture of alkali. (*d*) Included in "Other States."

CONSUMPTION OF SALT IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
1897...........	2,236,248	$4,920,020	209,025	$565,038	5,797	$52,320	2,439,476	$5,432,738
1898...........	2,465,769	6,212,554	185,530	588,653	8,640	63,624	2,642,659	6,737,583
1899...........	2,759,206	6,867,467	189,051	579,682	12,600	86,465	2,935,657	7,360,684
1900...........	2,921,708	6,944,603	199,909	634,307	7,511	65,410	3,114,106	7,513,500
1901...........	2,879,332	6,617,449	201,733	676,324	9,433	86,414	3,071,632	7,207,359
1902...........	3,338,892	5,668,636	184,764	647,554	5,094	55,432	3,518,562	6,260,758
1903...........	2,655,532	5,286,988	165,981	495,948	12,750	95,570	2,808,763	5,687,366
1904...........	3,084,200	6,021,222	166,140	467,754	13,964	113,625	3,236,376	6,375,351
1905...........	3,635,257	6,095,922	161,159	492,189	34,238	239,223	3,762,178	6,348,888
1906...........	3,944,133	6,658,350	170,505	502,583	33,988	274,627	4,080,650	6,886.306
1907...........	4,160,729	7,439,551	153,435	452,227	30,802	232,895	4,283,362	7,658,883

PRODUCTION OF SALT IN FOREIGN COUNTRIES.
(In metric tons.)

	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907
Algeria......	21,300	17,378	18,325	18,518	27,263	26,329	18,563	27,000	22,615	(c)
Austria.....	341,959	342,059	330,277	333,238	310,807	359,014	369,877	343,375	378,912	(c)
Canada.....	51,853	53,847	56,296	53,927	57,203	56,671	62,411	41,170	69,291	65,946
France.....	999,283	1,193,532	1,088,634	910,000	863,927	967,531	1,153,754	1,130,000	1,335410	(c)
Germany...	1,370,341	1,432,181	1,514,027	1,563,811	1,583,458	1,693,935	1,701,654	1,777,557	1,870,212	1,950,689
Greece......	37,125	22,411	22,411	23,079	25,200	26,000	27,000	25,201	25,167	(c)
Hungary....	178,551	182,593	189,363	(a)211,321	174,882	183,327	187,620	195,410	201,369	(c)
India (d)....	1,043,828	977,240	1,021,426	1,120,187	1,056,899	908,911	1,188,900	1,212,600	1,176,324	(c)
Italy.......	29,745	28,842	367,255	435,187	458,497	488,506	464,326	437,699	496,872	(c)
Japan.......	646,719	390,433	669,694	659,118	620,820	657,489	701,965	483,506	(c)	(c)
Russia......	1,505,600	1,681,362	1,768,005	1,705,922	1,847,019	1,658,938	1,908,275	1,844,678	1,730,934	(c)
Spain.......	479,358	598,108	450,041	345,063	426,434	427,394	543,674	493,451	541,978	(c)
U. Kingdom.	1,908,723	1,945,531	1,873,601	1,812,180	1,924,273	1,917,184	1,921,899	1,920,149	1,996,593	2,038,072
U. States....	2,382,197	2,522,610	2,651,278	2,612,204	2,409,174	2,408,646	2,797,461	3,297,285	3,578,061	3,773,781

(a) Sales by the royal monopoly, including imports entered for consumption. (c) Statistics not yet published. (d) Does not include the untaxed output of certain native States.

Market Conditions and Prices.—Market conditions were favorable during the greater part of 1907; the financial depression, however, coupled with the customary falling off in the demand at the approach of cool weather, caused salt manufacturers to inaugurate a policy of retrenchment, and the output was curtailed from 20 to 25 per cent. by many of the principal producers.

The prices fetched during 1907 were extremely variable, the product bringing all the way from 25c. to $1.25 per bbl., depending on quality, delivery, etc. In October, it was reported by the *Journal of Commerce* that an understanding had been reached between the International Salt Company and certain independent manufacturers in regard to regulating the future course of the market for evaporated salt. This action had been taken in consequence of the higher cost of labor and supplies, as well as the over-production of salt. As a result of this understanding prices advanced from 50c. to $1 per ton from the low figures prevailing in August and September, when sharp price cutting was in progress. Notwithstanding this advance, quotations were still below the top level of 1907, common bulk salt being quoted at $3.80 to $4.50 per ton at the point of production. According to the same authority, the International Salt Company, early in January, 1908, made a radical cut in the price of various grades of evaporated salt, ranging from 5 to 15c. per sack, or 50c. to $1.30 per ton. The extent of this cut came somewhat in the nature of a surprise in view of the fact that it was the dull season in the trade.

Salt Mining in the United States.

Michigan (By A. C. Lane).—The salt inspection of Michigan is once more in the hands of a practical salt man, Temple Emery, a former manufacturer at East Tawas, and one of the earliest in Michigan to

make bromine. His report for the year ending Nov. 30, 1907, shows, with its output of 6,298,317 bbl. of 280 lb., that the Michigan industry is by no means decadent, nor dependent on the lumber industry for its continued existence, for it is the largest in the history of the State and 10 per cent. greater than in 1906. Nor do these figures by any means include all the salt produced; there is still to be added the large amount made into soda, and the amount used in salt baths and medicines. The accompanying table summarizes some of the most important items of Mr. Emery's report.

SUMMARY OF IMPORTANT ITEMS OF THE REPORT OF THE MICHIGAN SALT INSPECTOR.

District.	County.	Number of Plants.	Annual capacity Bbl.	Number of Wells.	Approx. extreme depth of wells. Feet.	Steam blocks operated	Number of Grainers	Number of Vacuum Pans.	Number of Employees.	Bbl. salt inspected 1907.
No. 1.....	Saginaw..	9(a)	500,000	26	....	8	41	....	134	328,083
No. 2.....	Bay......	4	400,000	31	1,050	4	20	2	51	294,791
No. 3.....	St. Clair...	7	1,700,000	17	....	....	46	5	546	1,932,969
No. 4.....	Manistee..	5	3,000,000	20	2,000(b)	5	43	5	619	1,966,335
No. 5.....	Mason....	3	1,650,000	8	2,264	4	28	3	238	974,861
No. 6....	Wayne...	6	2,000,000	22	1,600	....	25	7 (c)	435	1,101,424
	Total....	34	9,250,000	124			203		2,023	6,298,863

(a) Also one solar plant, with 1170 covers and 18 settling vats. (b) Average depth of wells. (c) Also 10 open pans.

Statements regarding the geology and occurrence of Michigan salt are often incorrect because they are too general. The six districts into which the State is divided differ materially in the quality of the salt, the methods of manufacture and the geological conditions. Districts 1 and 2, Saginaw and Bay counties (and Midland and Isabella counties, where the bromine is made) are geologically similar and draw their brine mainly from the Marshall sandstone, a sub-Carboniferous sandstone whose base is usually from 750 to 1000 ft. beneath the surface along the Saginaw river. The plants are mainly grainers, but there is still one solar evaporation plant which is run in connection with the waste heat of some other industry. It is not always as formerly a lumber plant, but may be a piano factory, a coal mine, an electric power plant, or a glass works. The Saginaw plate glass plant is one of the newest and most improved and is worth a word of description. It was designed by the Wilcox Engineering Company.

The brine has about the following composition: Sodium chloride, 228 grams per kg.; calcium chloride, 45; magnesium chloride, 17; magnesium bromide, 0.3 to 0.1; calcium sulphate, 0.8; ferrous carbonate, 0.15; ammonium chloride, 0.05; silica, 0.02; alumina, 0.02. A brine like this, even when evaporated to a bittern and much of the salt expelled, has not enough gypsum in it for saturation, as Hahn has remarked. From this arises the characteristic feature of the Saginaw salt, that it does not

contain gypsum and is not harsh and hard and caking. On the other hand, unless thoroughly drained it tends to be moist and will deliquesce, owing to the presence of the calcium and magnesium chlorides.

This brine is treated with calcium hydrate, i.e., with quicklime, or white-wash. This combines with the carbon dioxide and throws out the iron, and very likely a little magnesia. At any rate, that is the effect of an excess, which produces magnesia hydrate. The ammonia is also left in such condition that in a later stage, in which the brine is aerated by being allowed to trickle down through a pile of brush, it is also liberated. But in the settling, greenish salts of iron are also thrown down. From the settling tanks and brush tower it passes to the grainers, which are built of concrete and steel. Thence it is raked automatically, and allowed to drain in huge piles, after being carried by conveyers, before barrelling. The quantity of bittern run off from the grainers and stock piles, largely calcium and magnesium chlorides, with a notable proportion of bromine, which is saved at St. Charles from a similar brine, offers a tempting field to the chemical engineer. More or less investigation has been carried on in this direction.

In the Saginaw valley district I believe the North American is the only concern using vacuum pans. In districts 3 and 4, Detroit and Wayne counties, a great highway of commerce follows the course of the St. Clair and Detroit rivers. The salt supply here comes from rock salt beds, of about the same age as the New York salt beds, the Salina, near the top of the Silurian. These are dissolved by river water, pumped down upon them. Off-color salt has occasionally resulted from not getting this water pure enough. The salt beds aggregate several hundred feet in thickness. Toward the south they are found at a depth of 800 and 1700 ft., and deepen going north. A shaft which is being sunk just south of Detroit is now more than half-way down, and a contract for its completion has been taken by the Wallace Company, of Duluth. Serious difficulties in the way of creviced and cavernous limestone, charged with sulphureted water, have been surmounted. For packers' use, a coarser salt would be more desirable than the recrystallized brines, and that is understood to be the chief reason for sinking this shaft. With the salt occurs anhydrite, and the characteristic impurity, which of course in the purer grades is reduced to extremely minute proportions, is sulphate of lime, which makes the salt feel harsh.

In this district is the soda-ash works. The necessary limestone of high grade is found just south in outcrops of the Dundee (Corniferous) limestone, at the Sibley quarries. The district around Alpena, where it would seem probable that a similar conjunction of salt, limestone and shipping facilities might be found, has not yet been developed, but is understood to be controlled by persons who understand the possibilities.

The fourth and fifth districts, Manistee and Mason counties, comprise the industry centering in the towns of Manistee and Ludington. Here the time honored association of lumber and salt still holds sway, though it is passing even here. The source is rock salt, as around Detroit, but instead of the many hundred feet there seems to be but one salt bed some 20 ft., or more, thick. Either the ordinary pump or air lift is used. In many cases instead of letting down water from the surface, natural underground brines are used to dissolve the salt. However, this introduces impurities and the advantage is questionable. The vacuum pan process is used and has been clearly described by J. J. Hubbell in a paper before the Michigan Engineering Society, entitled "A Barrel of Salt."

A recent improvement, as practiced at the Peters plant, is the placing of the vacuum pans in tandem, somewhat as in a compound condensing engine. Two have been run in this manner and it is now proposed to run three. Saw mill waste will not last long for fuel. Coal is already being introduced, and there has been some testing for oil or gas, of which there are signs and possibilities.

New York.—The salt strata in the Warsaw valley are at great depth, and are worked by means of wells. These wells run from $5\frac{1}{2}$ to 8 in. in diameter, and are cased with iron pipes down to the salt. Inside this pipe is then inserted a second pipe 2 in. in diameter, having perforations at its lower end. Between the two pipes fresh water is run down from the surface, which dissolves the salt, forming a strong brine, which being heavier, sinks to the bottom of the well and is pumped up through the second or inner pipe.

(By D. H. Newland.) The salt production of New York in 1907 amounted to 9,657,543 bbl., as compared with 9,013,993 bbl. in 1906, thus continuing the progress that has for some time been a feature of the industry. The value of the output was $2,449,178, exceeding that of the previous year by $317,528. There were six counties represented in the returns, with Onondaga county in the lead, though its output consisted mostly of salt used for soda manufacture. Livingston county made the largest quantity of marketable grade, chiefly rock salt.

Texas.—Salt occurs in lagoons along the Gulf coast and in many salt lakes, or "salines," throughout Texas, from which much is taken annually. The region of present commercial importance is in Van Zandt and Anderson counties, where salt is made from artificial brines drawn from wells which enter heavy beds of rock salt in Cretaceous strata at several horizons.

Utah (By L. H. Beason).—Owing to the abnormal precipitation, the wet season extending well into the summer months, the production of salt in Utah during 1907 was considerably lower than usual. The output is estimated to have been approximately 30,000 tons. The yield comes

from the water of the Great Salt Lake, which is pumped into large evaporating ponds. The principal refinery is situated near the Saltair bathing resort, 15 miles west of Salt Lake City. An enormous deposit of pure salt is reported to have been found on the west side of the Utah desert, not far from the Nevada State line.

SILICA.

Pure silica (SiO_2), an oxide of the element silicon, occurs in abundance both in pure and impure forms. Among the silicious materials which are produced and marketed may be mentioned silica sand, flint, quartz, ganister, infusorial earth and tripoli.

The production of silica sand has increased greatly of late years, due largely to the expansion in concrete construction in which the cheaper grades of sand are used. The chief use for pure silica sand is in the manufacture of glass.

Flint is the commercial term applied to quartz, the production of which in 1907 was 75,561 short tons, valued at $407,699, against 66,697 short tons, valued at $243,012 in 1906. The domestic flint used in the United States is amorphous and crystalline quartz, and is found in Maryland, Pennsylvania, Alabama and Connecticut. Flint which has been calcined is used in chemical works for lining Glover and Gay-Lussac sulphuric acid towers, and for condensers for other acid gases. The quantity used for pottery purposes is decreasing, for the reason that many potters prefer ground silica sand as it is usually freer from iron than the vein quartz. The greater part of the flint consumed in America is imported from England and France, and is used for pottery work, but a considerable quantity is now used as flint pebbles for grinding in ball mills in metallurgical and cement plants.

Ganister is a silicious fire clay and is extensively used for lining furnaces and converters.

Infusorial earth and tripoli include all porous silicious earths, composed of organic fragments, such as infusoria or diatoms, which have accumulated either on ocean bottoms or in ponds. Such deposits are common in the coastal plain area of Maryland, Virginia, Georgia, and Alabama, where they form beds several feet in thickness, generally interstratified with the Tertiary sands and clays. Tripoli is also found in quantity in the coast ranges of California. Deposits in Missouri are different in their nature, consisting of the residual silica left by the leaching of an impure limestone. The largest and best known deposits are those in northern Germany, where they are found from 15 to 18 ft. below the surface in a bed varying from 18 to 45 ft. thickness. This is exported to all parts of the world. Infusorial earth and tripoli are used chiefly for polishing powders and scour-

ing soaps, while their porous character makes them valuable as an absorbent for nitro-glycerine. As a non-conductor of heat they are of value for steam-boiler packing, for wrapping steam-pipes, and for fire-proof cement. The tripoli of Missouri is used for water-filters. Common absorbent grades bring 1½c. per lb., while the better grades fetch as high as 3c. per pound.

Volcanic Dust and its Present Production in Nebraska.

By Edwin H. Barbour.

Volcanic dust of surprising purity and whiteness occurs in extensive deposits throughout Nebraska, and the supply may be said to be inexhaustible. Because of its fineness and white color the attention of everyone seems to be attracted by it. The name geyserite which was erroneously given to this substance in the outset clings to it still, and of late certain trade names, such as diamond polish, gibson grit, etc., have been used. However, it is now pretty well recognized under the name volcanic dust, volcanic ash, or native pumice.

The Nebraska Geological Survey has many samples of the material from Indian Territory, Oklahoma, Kansas, Colorado, Wyoming, Montana, South Dakota and Iowa. Iowa is plainly the eastern limit of this volcanic dust, for it occurs in but one county as far as known, and is in a fine powdery form. In eastern Nebraska volcanic dust is not common, but to the westward the beds increase rapidly in number, extent and thickness, while the ash grows coarser in texture. Still farther west in Colorado there are beds which have a reported thickness of 600 to 800 ft., the texture being proportionally coarse and the color gray. Beds of fine ash, flour-like in color and fineness, may be traced as far north as the Big Bad Lands of South Dakota and to the Black Hills. In geologic range this volcanic dust may be traced from the White river Oligocene through Loup Fork. Since the beds of this dust increase in frequency and in coarseness of texture toward the southwest, that region is presumably the source of our supply. After the dust was driven into the upper air by volcanic explosions it was laid down as aeolian deposits in some places, and as aqueous in others. In the latter case it is finely stratified and contains in certain places occasional concretions. In western Nebraska, in Wyoming and Montana it often contains leaves and plants, as well as fresh-water shells.

In Harlan county, on the Kansas line, a number of good deposits are known which require but little stripping. These are the best known and the most fully developed of any deposits in the State. Many tons have been shipped from this county to such points as Denver, Lincoln, Omaha,

Chicago, Burlington, Philadelphia, Rochester, and New York. At the present time more is produced and marketed than ever before.

The uses to which this dust has been put have varied from year to year. At first it was put up in small packages and sold locally as a scouring powder. Considerable quantities have been shipped of late years and used in stove factories. While the native Nebraska pumice is not as vesicular in most cases as the imported, yet its low cost, $2 or less a ton, makes it an abrasive worth considering. Many manufacturing concerns can doubtless use it to advantage when the deposits are better known. For prudential reasons the producers of this volcanic ash, or pumice, are often reticent about reporting the amounts marketed. However, it is known that 5,000,000 ℔. of volcanic dust are now being used annually by Nebraska institutions alone in the manufacture of scouring preparations. What are considered inexhaustible supplies have been bought by these interests in Lincoln, Antelope, and Harlan counties. In these large holdings provision has been made to supply the market for years to come.

SODIUM AND SODA SALTS.

The production of metallic sodium has become an important branch of the metallurgical industry, although the product is used only to a small extent in the metallic form, it being employed chiefly for conversion into certain salts of sodium, principally sodium cyanide, sodium peroxide and caustic soda. In The Mineral Industry, Vol. XV, it was stated that the annual production of sodium in the United States, Great Britain and Germany was at that time approximately 3500 tons, divided nearly equally among the three countries. Of this amount about 1500 tons were used for sodium cyanide, 1500 tons for sodium peroxide, and 500 tons were sold as metallic sodium. In a paper in the *Transactions* of the Faraday Society, March, 1908, Dr. F. Mollwo Perkin stated the present production of metallic sodium to be about 5000 tons annually. It is produced by the Castner Kellner Company, at Weston Point, Cheshire, England, and at Wallsend-on-Tyne; in Germany by the Farbewerke vorm. Meister, Lucius and Brunning at Höchst, near Frankfurt; by the Elektrochemischen Werke Natrum, in Rheinfelden; in France by the Cie. d'Electro-Chimie at Gavet; also in America by the Electro-Chemical Company at Niagara Falls.

All of the above companies employ the Castner process, in which caustic soda is electrolyzed between electrodes of copper and nickel. Another process for the production of sodium by the electrolysis of sodium hydroxide is that of Rathenau and Suter, which has been in operation since 1895 at the Elektrochemischen Werken at Bitterfeld, and also in a modified form by the Aluminiumaktiengesellschaft at Neuhausen. The Becker process is used in France at the Usines de Rioupéroux. This process differs from the Castner in that a mixture of sodium carbonate and hydroxide is electrolyzed.

In the United States, metallic sodium is produced by the Electro-Chemical Company at Niagara Falls, N. Y., and the Virginia Electrolytic Company, at Holcombs Rock, Va. The latter company commenced the manufacture in 1908, and from now onward will doubtless be an important factor in the market. We have not received official statistics as to the production at Niagara Falls in 1908, but estimate the amount at 2000 tons, valued at 25c. per ℔. Dr. Perkin gives the present price in Great Britain as 8d. per ℔. In The Mineral Industry, Vol. XV,

it was estimated that the cost of production at Niagara Falls, N. Y., is 10@14c. per pound.

Nitrate of Soda in 1907.

By Reginald Meeks.

During the first quarter of 1907 prices ranged about the same as at the close of 1906. In the second quarter, there was an advance and the market was firm until about mid-year. Prices fell off in August, September, October and November about 5c. per 100 lb. each month, and there were numerous small parcels of surplus nitrate, due to simultaneous arrivals which should have been spread over four months, thrown on the market at any available price. Of course this had a demoralizing effect. December brought a steadier tone and the year closed with prices a little firmer. According to James S. Burrough & Company, New York, the highest price for the 95 per cent. grade was \$2.67½ per 100 lb. in April; the lowest, \$2.27½ in August; the average for the year was \$2.424. The 96 per cent. grade varied from 5@20c. higher.

NITRATE OF SODA STATISTICS. (a)
(In tons of 2240 lb.)

Year.	Shipments from South America.	Consumed in Europe.	Consumed in United States.	Consumed in World.	Stocks in Europe.	Visible Supply at close of year.
1899	1,373,000	1,140,000	160,000	1,330,000	236,000	741,000
1900	1,429,000	1,126,000	175,000	1,324,000	221,000	794,000
1901	1,238,000	1,154,000	192,000	1,364,000	243,000	617,000
1902	1,360,000	1,028,000	214,000	1,259,000	263,000	660,000
1903	1,435,000	1,127,000	265,000	1,412,000	155,000	654,000
1904	1,476,000	1,131,000	275,000	1,447,000	162,000	672,000
1905	1,623,000	1,190,000	308,000	1,547,000	183,000	674,000
1906	1,700,000	1,243,000	355,000	1,636,000	190,000	733,000
1907	1,628,000	1,257,000	349,000	1,662,000	202,000	691,000

(a) Statistics of W. Montgomery & Co., London.

IMPORTS OF SODIUM NITRATE INTO THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Quantity.	Value.	Value per ton.	Year.	Quantity.	Value.	Value per ton.
1898	147,495	\$2,298,240	\$15.58	1903	272,947	\$8,700,806	\$31.88
1899	146,492	3,486,313	23.80	1904	228,012	9,333,613	40.93
1900	182,108	4,935,520	27.10	1905	321,231	11,206.548	34.89
1901	208,679	5,999,098	28.75	1906	372,222	14,115,206	37.92
1902	205,245	5,996,205	29.21	1907	364,610	14,844,675	40.71

Some of the disturbing influences in the market were: (1) A strike at Iquique in December which spread to the Tarapaca *pampas* and then became general, causing some loss of life and a general shutting down of *oficinas*; (2) a considerable shortage of labor; (3) the suspension

of an important Chilean bank and the continued weakness of Chilean exchange; (4) the financial depression in the United States, affecting consumption in this country and making buyers hesitate to place orders for future deliveries; (5) the stocks on the West Coast were reported to be 573,000 tons at the close of 1907, against 429,000 tons on Dec. 31, 1906, which caused weakness in the markets for all positions.

According to the *Chemical Trade Journal*, the consumption of sodium nitrate in Europe showed a slight increase, but in the United States there was a decrease of 1.7 per cent. Germany and Austria consumed less nitrate of soda than in 1906 for two reasons: (1) a dock strike in Hamburg, the leading market for Chilean niter, caused a stoppage of shipments up the Elbe; (2) the German and Austrian sugar-beet growers to some extent replaced nitrate of soda with ammonium sulphate obtainable on more advantageous terms. The United Kingdom imported 4070 tons in excess of the usual requirements, all of which was consumed by explosives manufacturers in Scotland. While Germany and Austria took 13,140 tons less, France took exactly as much more. The deliveries from Rotterdam, mostly for the chemical works in West Germany, were 15,430 tons more than in 1906, whereas Belgium's quota fell off 13,500 tons; Italy took 4680 tons less and Spain 1150 tons more than in 1906.

During 1907 there were 14 new producers in Chile, which were given an initial quota of 500,000 tons. Most of the new companies began production in the second half of the year and their output was not more than 85,000 tons. The total on Jan. 1, 1907, was 155 officially listed *oficinas*.

The labor troubles in Chile in 1907 developed strikes at most of the loading ports. The workmen, not unreasonably, demanded compensation for the depreciation of the currency in which they are paid, and several thousands marched down to Iquique, from the works in Tarapaca, to submit their grievances to the authorities; operations were suspended at about 30 *oficinas*, and troops were despatched to maintain order. An engagement ensued in which many lives were lost. The men went back to work on Dec. 26, 1907, and shipping was resumed at all nine ports, from Pisagua to Taltal.

Consul Alfred A. Winslow, of Valparaiso, states that from 1830 until 1907 the nitrate fields of Peru and Chile produced 36,443,327 tons of nitrate, valued at $1,112,728,765, gold. About two-fifths of this was produced during the last 10 years. There has been much said about the exhaustion of the nitrate beds, but from the best information obtainable they are good for 200 or 300 years more, even at double the present production.

The "combination" has handled the nitrate situation with a strong hand and the price of the salt has been maintained at an average of about

8s. 6d., f.o.b. Chilean ports. The *Chemical Trade Journal* estimates that at this price the profit is at least 3s., or 150 per cent. of the cost on the drying floors. In spite of the efforts of the nitrate producers to regulate production and consumption, the stocks held in Chile at the close of each year have increased annually as follows: 251,000 tons in 1902; 269,000 in 1903; 290,000 in 1904; 368,000 in 1905; 429,000 in 1906; and 573,000 in 1907.

Soda Salts in the United States.

California.—It is reported that the Pacific Coast Soda Company has purchased a tract of land at Inglewood, Cal. and will build a factory to produce crude soda, caustic soda and chloride of calcium from the natural soda deposits in Death Valley.

The deposits of sodium nitrate in California lying along the Colorado river where the Santa Fe railroad crosses it in San Bernardino county, south to a peak, known locally as Vivet Eye, in Riverside county have been investigated by H. W. Turner.[1] Special examinations for nitrates were made at Topoc, West Well, on the south border of the Colorado River Indian reservation, in Arizona and in the Vivet Eye area. Nitrates were found in the forms of efflorescence, chiefly on porous volcanic breccias; in the shales of lake-beds; and as deposits associated with other salts on hill slopes. Only in the area about Vivet Eye are the deposits of any importance and these in no way compare with the Chile fields. A number of cuts were made in the cement-breccia and the content of sodium nitrate was found to diminish rapidly with depth. At some points, however, nearly pure deposits of white salts, a foot or more thick, were observed and subsequent analyses showed the presence of sodium sulphate, sodium chloride, calcium chloride, magnesium sulphate, potassium sulphate and the nitrates of sodium and potassium. Two samples showed 7.8 and 14.4 per cent. sodium nitrate respectively. The area around Vivet Eye comprises about 10 square miles.

Kansas.—The Hutchinson Alkali and Chemical Company is erecting a plant at Hutchinson, Reno county, to manufacture soda ash from the great deposits of salt in the Hutchinson district. The plant is to cost $375,000 and will have an initial capacity of 120 tons of soda ash per day. The company is capitalized at $600,000.

The Markets for Soda Salts in 1907.

The market for soda salts in 1907 was steady and prices remained practically stationary during most of the year. Bicarbonate was sold

[1] *Min. and Sci. Press*, May 18, 1907.

on a basis of \$1.30 for bulk and \$1.50 for kegs with the usual 20c. advance for New York delivery and the customary discount terms. There was an absence of underselling and about the same shipments were made in 1907 as in 1906. The demand for caustic soda was heavy in the beginning of the year and large contract shipments were made. Later on consumers bought only for current needs. Prices held steady on the basis of \$1.75@1.80. The 60 per cent. grade sold 10c. higher per 100 lb. Bleaching powder was in good demand and sold during the first half of 1907 for \$1.30@1.50 per 100 lb. In the second half of the year the price receded to \$1.25@1.40. Sal soda held steady at 70@80c. all the year. Dry weather and the financial depression caused a number of paper mills to close and the demand for this salt fell off considerably in the last few months of the year. Salt cake opened quietly at 65c. but in May prices were cut to 40c. During the latter part of the year strikes closed several glass plants and the demand declined. At the close of the year salt cake could be bought for 40@42c. Chlorate of soda was steady and experienced none of the violent fluctuations which marked the market in 1906. Prices throughout 1907 were on a basis of 8½c. for carloads and 9½c. for smaller quantities. Late in the fall heavy contracts for 1908 delivery were made at the same prices.

SULPHUR AND PYRITE.

Most of the sulphur produced in the United States is furnished by the Union Sulphur Company. Sulphur is also produced in California, Colorado, Wyoming, Utah and Nevada, but in none of these States has the industry reached large proportions. During 1907, discoveries of sulphur were reported from several points in Colorado and from Jefferson county, Texas; plans were also made for the erection of a sulphur refining plant in Gunnison county, Colorado, and one in Fremont county, Idaho.

CONSUMPTION OF SULPHUR IN THE UNITED STATES.
(In tons of 2240 lb.)

Source.	1901	1902	1903	1904	1905	1906	1907
Sulphur—Domestic production	6,866	7,443	35,098	193,492	215,000	294,000	307,806
Imports	175,310	176,951	190,931	130,421	84,579	64,646	18,124
Total	182,176	184,394	226,029	323,913	299,579	358,646	325,930
Exports	207	1,253	967	2,493	1,713	14,419	35,925
Consumption	181,969	183,141	225,062	321,420	297,866	344,227	290,005
(a) Sulphur contents	178,330	179,478	220,560	314,992	291,909	337,342	284,205
Pyrites—Domestic production	234,825	228,198	199,387	173,221	224,980	225,045	261,871
Imports	403,706	440,363	427,319	413,585	515,722	533,346	627,985
Total	638,531	668,561	626,706	586,806	740,702	758,391	889,856
Exports	....	3,060	1,330	....	....	....	....
Consumption	638,531	665,501	625,376	586,806	740,702	758,391	889,856
(b) Sulphur in domestic	103,323	104,071	87,730	76,217	98,991	99,020	115,223
(c) Sulphur in foreign	189,742	205,532	200,215	194,385	242,389	250,673	295,153
Total sulphur content	189,742	309,603	287,945	270,602	341,380	349,693	410,376
Grand total sulphur consumption	471,395	489,081	508,505	585,594	633,289	687,035	694,581

(a) Includes crude and refined sulphur. Sulphur content of crude is computed at 98 per cent. (b) Computed at 44 per cent. (c) Computed at 47 per cent.

WORLD'S PRODUCTION OF SULPHUR. (a)
(In metric tons.)

Year.	Austria. (d)	Chile.	France. (c)	Hungary	Germany	Greece.	Italy. (b)	Japan.	Spain.	United States.	Total.
1895..	830	....	4,213	102	2,061	1,480	370,766	15,557	2,231	1,676	398,916
1896..	643	940	9,720	138	2,263	1,540	426,353	12,540	1,800	3,861	459,798
1897..	530	664	10,723	112	2,317	358	496,658	12,013	(b)3,500	1,717	528,592
1898..	496	1,256	9,818	93	1,954	135	502,351	10,339	3,100	2,770	532,312
1899..	555	989	11,744	116	1,663	1,150	563,697	10,241	1,100	1,590	592,290
1900..	862	2,472	11,551	123	1,445	891	544,119	14,439	750	4,630	581,282
1901..	4,911	2,516	6,836	137	963	2,336	563,096	16,548	610	6,977	604,930
1902..	3,721	2,636	8,021	105	487	1,391	510,333	18,287	450	7,565	552,996
1903..	4,475	3,560	7,375	135	219	1,266	553,751	22,914	1,680	35,660	631,035
1904..	6,288	3,594	5,447	143	209	1,225	527,563	25,587	605	196,588	767,249
1905..	8,407	3,470	4,637	135	205	1,126	568,927	24,652	610	218,440	830,609
1906..	15,125	(e)	2,713	133	178	(e)	499,814	27,589	700	298,704	844,956
1907..	(e)	(e)	(e)	(e)	176	(e)	(e)	28,381	(e)	312,731	....

(a) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to between 20,000 and 30,000 long tons annually, is not included. (b) Crude. (c) Crude mineral; limestone impregnated with sulphur. (d) Crude rock. (e) Not yet reported.

IMPORTS OF SULPHUR INTO THE UNITED STATES.
(In tons of 2240 lb.)

Kind.	1904		1905		1906		1907	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
Crude	128,885	$2,463,779	83,201	$1,522,005	72,603	$1,282,873	20,399	$355,944
Flowers	1,332	39,133	572	16,037	1,099	29,565	1,458	41,216
Refined	163	4,373	79	19,960	709	17,928	606	14,589
Precipitated	41	5,403	27	3,352	28	3,224	60	8,426
Total	130,421	$2,512,688	84,579	$1,561,354	74,439	$1,333,590	22,523	$420,175

Industrial Conditions.

The year 1907 opened with a good demand for sulphur at prices ranging from $22.12 to $22.50 per ton, according to quality and point of delivery. Business continued good until June, when there were signs of a lessening demand; during July and August there was a material shrinkage in the demand. In the latter part of August it was learned that offers had been received here to sell Sicilian sulphur at $19.50 per ton ex ship. This, coupled with the unsettled condition of the industry in Italy, precipitated an industrial war. It was a case of an open world market wherein the supply was considerably larger than the demand. Conditions appeared to favor the Union Sulphur Company, which was not overburdened with an enormous stock, nor was it laboring under the serious disadvantage of being forced to consider the welfare of some 30,000 people who were dependent on the industry for their livelihood, as was the case in Italy. The financial situation abroad was none too good and the banks which had advanced money on sulphur warrants to the extent of four-fifths of the market value of the ore, refused to loan in excess of three-fifths, although to offset this to some extent an additional 4,000,000 lire was granted to the Consorzio. Nevertheless, fears were entertained that unless production was reduced substantially, additional credit would become necessary before long. To sustain the market and eliminate an enormous accumulation of stocks, production would have to be curtailed; but this move would deprive hundreds of Sicilian laborers of a livelihood and might eventually lead to a revolution.

The solution of the problem was extremely difficult. Both parties maintained the utmost secrecy regarding further moves and consumers naturally became skeptical about entering into long contracts under the unsettled conditions. In consequence, the demand for large lots decreased, while the demand for small lots increased. At times it looked as if an agreement had been reached between the rival producers and that the war would be declared off, but of this there was no definite knowledge. It can only be said that sulphur moved quite freely for immediate delivery and was held steady in price. During the last four months of the year,

prices ranged from $22.50 to $19.50; these figures represent the high and low prices for the year.

Sulphur in the United States.

Colorado.—During 1907, the Colorado Sulphur Company began active operations on the sulphur deposits on Trout creek, in Mineral county. The Colorado, Good Hope and Vulcan properties, in Gunnison county, were operated during the year and plans were made for the erection of a new sulphur refining plant. The discovery of a sulphur deposit of great purity was reported in the neighborhood of Victor.

Idaho.—It was reported that the Wyoming Sulphur Company made plans for the erection of a small milling-plant in the vicinity of Thermopolis, in Fremont county.

Louisiana.—As previously mentioned, the Union Sulphur Company, operating in Calcasieu county, controls the sulphur industry in this State. The methods of winning the sulphur, treatment, storing, etc., have been fully described in previous volumes of The Mineral Industry. About the end of May, 1907, the property of this company, situated at Sulphur City, near Lake Charles, was visited by a flood, brought on by heavy rains. The early reports of the damage done were greatly exaggerated. As a matter of fact, production was checked for only four days, and shipments for only two days. There was no loss of wells through settlement of the ground. Up to the present time settlement has occurred over an area of about one-fourth of a square mile, the maximum settlement having been 12 ft. The natural elevation of the land at this place is about 18 ft. above sea level.

During 1907, drilling for sulphur was done on the Kerr land, about one mile north of Sulphur. It was reported that the first hole, which went to a depth of 1000 ft., failed to disclose sulphur; however, the work of drilling is being continued.

Texas.—During the summer of 1907, the discovery of extensive sulphur and salt deposits were reported not far from Beaumont. The deposits were said to cover an area of about 400 acres, the sulphur being from 10 to 11 ft. below the surface and the salt about 15 ft. Plans were made for the operation of these deposits but no production was reported from this source during the year.

Wyoming.—The producing mines of this State are included within an area only a few acres in extent, situated about three miles west of Cody, along the base of Cedar mountain, on the south side of the Shoshone river. The sulphur occurs in small yellow crystals and in gray streaks in the rocks, and is found in irregular beds in the limestone and travertine, associated with fine white crystalline aggregates, filling crevices 2 to 8

in. in diameter, and also disseminated through the limestone. Mining is conducted in open pits; the rock is hand-sorted and all ore estimated to contain more than 30 per cent. sulphur is sent by wagon or tram to the smelter, near the mouth of the cañon. The handling and treatment of the ore were discussed in Vol. XV of The Mineral Industry.

Sulphur in Sicily.

During the last six years the production of sulphur in Sicily has considerably exceeded the consumption, as is shown by the statement of stocks on hand at the close of each year. The unsold stocks on Dec. 31 are reported as follows, in metric tons: 1902, 339,113; 1903, 361,220; 1904, 396,541; 1905, 462,437; 1906, 525,115; 1907, 576,377. From the close of 1902 to the end of 1907 the stocks increased by 237,264 tons. The exports, added to the increase in stocks, give a production of 385,276 tons for 1907, so that the stocks at the close of the year were equal to about 1½ years' production. This does not appear to constitute a condition favorable to an advance in prices; but the Sicilian producers seem to rely upon government aid and the operations of the Consorzio Obbligatorio for the restoration of their declining trade. Moreover, they do not seem even yet to appreciate the importance of the American production, and its probable permanence; nor the advances which the use of pyrites has made, largely as a consequence of the high prices exacted for brimstone.

Mr. Frasch, of the Union Sulphur Company, has been in conference at Paris with the representative of the Consorzio, Signor Luzzatti, who was formerly the Italian minister of finance. Concerning this conference, Messrs. Fog & Sons report, under date of Feb. 1, 1908: "So far, nothing has transpired as to the result of negotiations; but as both parties have a common interest in avoiding a mutually detrimental price-war, it seems beyond doubt that some agreement will be come to. Even if Mr. Frasch could not be persuaded to buy annually a large quantity of brimstone from Sicily, he is sure to agree as to the necessity of maintaining prices. As a preliminary measure the Consorzio has increased prices for America from $19.50 to $22, which means that prices for Europe, although they have been raised 5c. per month from February until June, are now cheaper than those quoted for export to the United States. Actual price for California is now $16.44, and for Australia $16.68 per ton, f. o. b.

"As the Italian government at the same time renewed its promises of assistance, there is little doubt that the State banks will receive orders to provide also in future the large amounts required to carry the stocks inherited from the Anglo-Sicilian Sulphur Company forward until disposed of. The apprehension of disorders among the miners is so great that the economical considerations have been pushed into the background for the

time being. The Consorzio, therefore, will not lack the necessary financial aid to hold the stocks over until the increasing consumption of sulphur will gradually absorb the actual excess of supplies."

SHIPMENTS OF SULPHUR FROM SICILY TO THE UNITED STATES.
(In tons of 1030 kg.)

Port.	1902		1903		1904		1905		1906		1907	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Refined. (c)
New York....	76,383	26,842	70,800	21,201	41,429	10,547	26,782	16,270	18,106	13,009	3,098	254
Philadelphia..	3,500	10,399	4,910	8,500	1,325	5,825	800	1,848	554	200	....	275
Baltimore....	9,065	2,400	10,900	2,000	3,370	1,400	.....	....	21	....	....	......
Boston......	2,204	2,300	5,508	2,450	11,397	1,749	5,477	1,009	959	1,750	....	169
Portland, Me.	26,328	.....	23,855	.....	23,638	.....	17,882	.....	4,993	....	....	....
Other ports (a)	8,498	1,000	5,872	.....	.....	.....	.....	.....	1,419	272	(b) 5,500	180
Totals.....	125,978	42,941	121,845	34,151	81,159	19,521	50,941	19,127	26,052	15,231	8,598	878

(a) Norfolk, Mobile, New Orleans, Savannah, San Francisco, Bangor, Portland, Ore., St. Louis and Canada. (b) All to San Francisco. (c) No thirds shipped.

TOTAL EXPORTS OF SULPHUR FROM SICILY, 1900-1907. (a)
(In tons of 1030 kg.)

Country.	1900	1901	1902	1903	1904	1905	1906	1907
Austria................	21,594	18,842	19,086	17,926	23,374	25,111	22,756	24,597
Belgium..............	9,721	7,471	12,323	15,233	13,627	14,442	13,940	8,853
France................	103,647	74,394	67,249	74,372	103,040	96,170	67,536	59,725
Germany..............	28,702	23,448	25,906	32,553	31,613	28,319	34,967	37,100
Greece and Turkey......	19,647	21,702	20,548	22,133	25,376	25,069	26,560	27,608
Holland..............	18,595	10,848	8,648	5,157	8,122	4,425	5,539	11,379
Italy..................	101,073	74,516	45,603	45,572	79,619	99,633	79,519	58,926
Portugal..............	10,937	11,335	10,614	14,064	8,373	13,196	12,302	12,778
Spain................	6,187	2,979	2,249	4,099	4,064	2,478	3,120	
Scandinavia (c)........	22,681	24,486	24,918	28,292	20,120	18,288	21,608	25,155
Russia................	22,090	15,110	17,295	15,068	15,141	16,673	16,181	15,210
United Kingdom........	23,973	22,468	25,477	19,210	18,108	18,847	20,883	16,561
United States..........	162,505	144,817	168,919	155,996	100,000	70,332	41,283	9,476
Other countries (b).....	6,810	9,484	18,484	25,833	25,167	23,277	21,238	26,646
Totals.............	558,162	462,299	467,319	475,508	475,745	456,260	387.432	334,014
Stock in Sicily, Dec. 31.	221,204	302,410	339,113	361,220	396,541	462,437	525,115	576,377

(a) In 1900 and 1901 by A. S. Malcolmson, New York; for following years, by Emil Fog & Sons, Messina. (b) Mainly South Africa, Northern Africa, Australia and the East Indies. (c) Including Norway, Sweden and Denmark.

Pyrites.

By J. J. Rutledge.

During 1907, many of the pyrites mines in the United States made a smaller output than in 1906. Some new work was reported, while several companies ceased operations during the year. New York, however, showed a substantial increase in production, and on the whole, the production of the United States in 1907 showed an increase as compared with that of 1906.

PRODUCTION, IMPORTS AND CONSUMPTION OF PYRITE IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production.		Imports. (b)		Consumption.	
1897	133,368	$404,699	259,546	$847,419	392,914	$1,252,118
1898	191,160	589,329	171,879	544,165	363,039	1,133,494
1899	178,408	583,323	310,008	1,074,855	488,416	1,658,178
1900	201,317	684,478	322,484	1,055,121	523,801	1,739,599
1901	234,825	1,024,449	403,706	1,415,149	638,531	2,439,598
1902	228,198	971,796	440,363	1,650,852	668,561	1,622,648
1903	199,387	787,579	425,989	1,628,600	625,376	2,416,179
1904	173,221	669,124	413,585	1,533,564	586,806	2,202,688
1905	224,980	752,936	515,722	1,780,800	740,702	2,533,736
1906	225,045	767,866	597,347	2,138,746	822,392	2,906,612
1907	261,871	851,346	656,477	2,637,485	918,348	3,488,831

(a) These statistics do not include the auriferous pyrite used for the manufacture of sulphuric acid in Colorado. (b) Net imports, less re-exports of 3,060 tons in 1902 and 1,330 tons in 1903.

WORLD'S PRODUCTION OF PYRITES.
(In metric tons.)

Year.	Belgium.	Bosnia	Canada.	England.	France.	Germany.	Hungary.	Italy. (a)
1896	2,560	.....	30,580	10,177	282,064	129,168	52,697	45,728
1897	1,828	.....	35,291	10,752	303,488	133,302	44,454	58,320
1898	147	3,670	29,223	12,302	310,972	136,849	58,079	67,191
1899	283	.....	25,112	12,426	318,832	144,623	79,519	76,538
1900	400	1,700	36,308	12,484	305,073	169,447	87,000	71,616
1901	560	4,570	31,982	10,405	307,447	157,433	93,907	89,376
1902	710	5,170	32,304	9,315	318,235	165,225	106,490	93,177
1903	720	6,589	30,822	9,794	322,118	170,867	96,619	101,455
1904	1,075	10,421	29,980	10,452	271,544	174,782	97,148	112,004
1905	976	19,045	29,713	12,381	267,114	185,368	106,848	117,667
1906	908	13,474	35,927	11,318	265,261	196,971	112,623	122,364
1907	(b)	(b)	34,494	10,357	(b)	196,320	(b)	(b)

Year.	Japan.	Newfound-land	Norway. (c)	Portugal. (d)	Russia.	Spain.	Sweden.	United States.	Total.
1896	(b)	27,267	60,507	207,440	11,550	100,000	1,009	111,031	1,071,778
1897	7,626	32,790	94,484	276,738	19,380	100,000	517	133,502	1,252,472
1898	8,726	32,335	89,763	302,686	24,570	70,265	386	194,219	1,341,383
1899	8,376	26,154	95,636	347,234	23,250	107,386	150	181,263	1,446,782
1900	16,166	*Nil*	98,945	402,870	23,154	34,638	179	204,538	1,464,512
1901	17,589	7,532	101,894	443,397	30,732	33,953	*Nil*	238,582	1,568,999
1902	18,580	26,000	121,247	413,714	26,465	145,173	*Nil*	231,849	1,713,654
1903	16,149	42,674	129,939	376,177	22,780	155,739	7,793	202,577	1,692,812
1904	24,886	61,166	133 603	383,581	31,667	161,841	15,957	175,992	1,696,099
1905	25,569	51,534	162,012	352,479	30,689	179,079	20,762	228,580	1,789,816
1906	36,038	28,583	197,886	350,746	(e)30,000	189,243	21,827	228,646	1,841,815
1907	36,124	(b)	(b)	(b)	(b)	(b)	(b)	266,061	....

(a) Cupriferous in part. (b) Reports not yet available. (c) Both iron and copper pyrites. (d) Copper pyrite. (e) Estimated.

Alabama.—The Alabama Pyrites Company, mining near Pyriton, Clay county, ceased operations in 1907 and is now dissolved.

Georgia.—A new pyrites mine near Acworth began operations in 1907. The shaft is now 200 ft. deep, and a mill equipped with Blake crushers, Harz jigs and Wilfley tables went into commission.

Massachusetts.—The Davis Sulphur Ore Company's mine at Davis, Franklin county, reached a sufficient depth in 1907 to turn the 19th level. The persistence of this deposit is probably unequalled elsewhere in

this country; the vein has a pitch of 70 to 80 deg. Work was lately resumed in No. 3 shaft which, for some years past, has been idle. As in previous years, much the larger part of the output from this mine was lump ore.

New York.—Work at the Frank Cole mine, in St Lawrence county, was suspended late in 1907. The St. Lawrence Pyrites Company, operating the Stella mine, made very important improvements in its property during the year. Some diamond drilling was done and a number of shafts were sunk. This company completed, in 1907, probably the largest and best equipped pyrites concentrating mill in the United States. Its equipment includes a Gates crusher, rigid rolls, Hancock jigs and Harz jigs, with the necessary conveying and elevating appliances. This is, so far as known, the first application of the Hancock jig to pyrites concentration, and its operation will be noted with interest by owners of fine-ore properties.

(By D. H. Newland.) There was a large increase in the pyrites production in 1907, the quantity reported amounting to 49,978 long tons, against 11,798 long tons in 1906. The output came entirely from St. Lawrence county, from the mines of the St. Lawrence Pyrite Company, and the American Pyrites Company. The property of the former company is situated one mile north of Hermon and includes the old Stella mine, the first opened in the district. This is not worked at present. The ore is taken from two new shafts southeast of the Stella mine, developed since 1904 when the present company was formed. An average of from 250 to 300 tons daily is mined. The whole output goes to the mill where it is concentrated to a product assaying from 47 to 48 per cent. sulphur. Concentration is effected by Hancock jigs, with retreatment of the tailings after crushing on Harz jigs. The slimes are passed over Overstrom tables. The mill is new throughout and has a nominal capacity of 500 tons a day. The National Pyrites Company, which formerly worked the mines at High Falls, has retired from business, and the property has been taken under lease by the Oliver Iron Mining Company, a branch of the United States Steel Corporation. It is now being prospected in a thorough manner with the diamond drill. The American Pyrites Company was active during the first half of 1907 at the Cole mine, near Gouverneur, but later suspended operations with no prospect for immediate resumption. The difficulty is said to be due to the heavy exactions by owners of the property in the way of royalty. The mine yields an ore above the average in richness, a part of the output being suitable for shipment in the crude state. There was a good demand for pyrites from this section, as the local consumption of sulphur by pulp mills greatly exceeds the output, while owing to the absence of arsenic and other harmful impurities, the mineral finds a ready market among sulphuric acid makers.

Virginia.—The mines in Louisa county were active throughout 1907,

and at the end of the year all were working to their full capacity. The Arminius mine has reached a depth of about 1000 ft. As in former years, the greater part of the output was milled and shipped as fines. Important improvements are being made in the surface plant. The Smith mine resumed shipments after having been idle for several years. The output is fines altogether. The west shaft is being sunk to the 250-ft. level. The Sulphur Mines reached a depth of about 800 ft. and were in active operation during 1907. Some important changes in the surface equipment were made during the year. The Cabin Branch mines at Dumfries, Prince William county, have reached such a depth that the 16th level is ready to drive. This mine is working in probably the thinnest profitable vein of pyrites in the country. Within recent years the proportion of chalcopyrite has increased notably. A distinct innovation has been installed recently. A Traylor, 42-in., circular, copper-smelting furnace of 50 tons daily capacity has been erected and is now in operation. It is planned to use the cinders after the burning of the pyrites as a flux for the smelting operation. The copper ore is roasted in heaps of 500 tons each before smelting.

Canada.

During 1907 pyrites was developed in several new Canadian localities. Important deposits were opened in the Temagami forest reserve, in eastern Ontario. The Canadian pyrites is fairly rich, in some localities almost equalling the Spanish ore, and it likewise yields a high proportion of lump ore, when mined. At present, much of the best Canadian pyrites is remote from transportation, but within a few years Canada will undoubtedly become an important producer.

Prices.

Early in 1907, domestic non-arsenical, furnace size, was quoted at 11@11½c. per unit; fines, 9@10½c.; imported non-arsenical, furnace size, 13@13½c.; arsenical, 12@12½c.; imported fines, arsenical, 8½@9c.; non-arsenical, 10½@11c. The prices are given per unit of sulphur. An allowance of 25c. per ton is made when delivered in lump form. In June, domestic non-arsenical fines were quoted at 10@10½c., the prices for the other grades remaining unchanged. At the end of the year prices were practically the same, with the exception of imported material, which had advanced about ½c.

Import Business.

The chief importers of Spanish pyrites into the United States are Naylor & Co., Ladenburg, Thalmann & Co., and the Davis Sulphur Ore Company, all of New York, and the Pennsylvania Salt Mfg. Company, of Philadelphia.

The last imports between 150,000 and 200,000 tons of Rio Tinto pyrites per year, a portion of which is used in the manufacture of sulphuric acid at the company's works at Natrona, Penn. This company's imports exceed its consumption, so that considerable of the pyrites is sold to other acid makers. After burning off the sulphur, the cinder is returned to the works at Natrona, where the copper is extracted by the Henderson process. The residue is then sold to manufacturers of pig-iron, who purchase it for the iron which it contains. Ladenburg, Thalmann & Co. ships its ores from Pomaron, Portugal; Naylor & Co., the Davis Sulphur Ore Company and the Pennsylvania Salt Mfg. Company ship from Huelva, Spain. The Newfoundland Syndicate brings pyrites into the United States from Newfoundland. The total quantity of pyrites brought into the United States in 1907 from Spain, Portugal and Newfoundland was about 625,000 long tons. As a rule the shipments from Huelva and Pomaron constitute the entire cargoes of the vessels, and the freight rates are very moderate.

Only a small part of the pyrites from these countries carries workable amounts of copper; the greater part contains only 0.30 to 0.50 per cent. copper. Most of the foreign pyrites comes in lump form, but some of it is in the form of washed fines, from which the copper has been extracted. As a rule, domestic fines are needed to assist in the burning of the foreign fines. The foreign ore is richer in sulphur and supplies plants along the Atlantic coast from New Orleans to Boston, and goes as far west as Chicago and Detroit.

In 1907 foreign business was active until November, when a decline set in. In spite of this, the total imports were about 20 per cent. greater than in 1906. From present indications, imports during 1908 will reach only three-quarters of those during the past year.

Improvements in the Technology of Pyrites.

Breaking.—Until recently most of the pyrites mines have preferred to break the ore to furnace size, 2½ to 3 in., by hand at the mines, owing to the fact that most mechanical breakers have been found to produce too large a proportion of fines. A number of mines producing lump ore have lately made arrangements with the acid plants whereby the latter break the ore to furnace size themselves, the lump ore being sent directly from the mine to the acid works. The process followed at the acid plants for breaking lump ore to furnace size consists of two stages, using crushers of the Blake type. The fines by this process do not average more than 20 per cent., and the cost varies from 20 to 30c. per ton, as against 30 to 50c. per ton by hand labor. The apparently excessive employment of hand labor at pyrites mines has usually been justified by the smaller proportion of fines made in this way.

Concentrating.—Until recently, Harz jigs of three or four compartments, and Wendt-Bacon two-compartment jigs have generally been employed for concentrating pyrites fines. The introduction of the Hancock jig, with its much greater capacity, will probably lead to its general adoption by mines whose chief output is fine ore. It has not usually been thought worth while to install tables for the treatment of an ore of such small value as pyrites, but at least one plant has recently introduced Wilfley tables, and others are contemplating the use of Overstrom, Wilfley and similar machines. One Southern mine is planning to use Huntington mills in the treatment of its ore.

Pyrites in Alabama.

By Eugene A. Smith.

Along the eastern flank of Talladega mountain there is a narrow outcrop of green schist, which may be followed almost without interruption from Chilton county to the Georgia line. This is known throughout this section as the copper lead, although very little copper has been extracted from any part of it. On the other hand, beds of pyrites of considerable importance occur at several points along this outcrop. Near Gold Branch, in Coosa county, and in the vicinity of Dean, in Clay county, mining operations of some magnitude have been carried on. For several miles northeasterly from Dean the bed of pyrites appears at its best, being several feet in thickness and quite free from impurities. This bed was first worked for copper, of which it carries a small percentage, at the old Montgomery Copper Works, where during the civil war some blue-stone and perhaps some other copper salts were extracted. The remains of the furnaces and other buildings of this works were standing till quite recently, when they were torn down and the bricks used about the mines at Pyriton.

At the present time several companies are engaged in the production of pyrite in the vicinity of Pyriton. The Alabama Sulphur Ore and Copper Company, at Pyriton, has in operation a complete concentration plant, the capacity of which is 60 tons daily; when the improvements now in progress have been made, the capacity will be doubled at least. The pyrite is marketed now in Atlanta, but will probably be used in Alabama in great part when new contracts are made. The average of sulphur is given at 44 per cent., with 0.80 per cent. copper. From the cinder, after burning off the sulphur, this small percentage of copper may be extracted with profit. The vein about Pyriton will average perhaps 8 ft. in thickness, but it varies, being sometimes as much as 20 ft. One of the slopes is down 500 ft. and the other about 200, the dip being about 40 deg. to the southeast. This plant is under the management of Percy Smith.

About a mile southwest of Pyriton is the mine of the Southern Sulphur

Ore Company, under the superintendence of W. T. Williams. This mine is on the same vein as that just mentioned. Its average thickness is about nine feet and the average content of sulphur 40 per cent. The dip is the same as at Pyriton, and the slope is now down about 200 ft. The ore from this mine is first sorted by hand, the marketable ore is then crushed and shipped to the plants of the Tennessee Fertilizer Company, at Montgomery and Albany, Ga., which are run in connection with the mine. The low-grade ore is sent to the dump to be worked over later by concentrating machines. The present daily output of marketable ore is 50 tons. As at Pyriton, there are here portions of this vein which contain some copper. The mine is near the old Montgomery Copper Works, and a mile or two further southwest Mr. Zell is preparing to make another opening on the same vein.

In the southeastern part of Clay county, near Hatchett Creek P. O. and at the old McGhee copper mines, are other occurrences of the ore; from the first named locality considerable ore has been shipped. The cost of hauling the ore in wagons over the mountain to a railroad station has been too great to justify the continuation of these operations.

Pyrites Mining and Milling in St. Lawrence County, New York.

By Felix A. Vogel.

Pyrites is found extensively in the northwestern foothills of the Adirondack mountains; mining of the ore, however, has been confined to a limited area, extending from Gouverneur to Pyrites, St. Lawrence county. This is mainly accounted for by the low sulphur contents of a number of these deposits, their distance from shipping points and, most important, the large expenditure necessary to transform the ore into a marketable product.

The pyrites deposits in the northern part of the State vary considerably as to extent and quality. They occur mostly as bedded veins, conformable to the gneisses, schists and crystalline limestones, which strike southwesterly and northeasterly and pitch from 25 to 45 deg. to the northwest. The veins are apparently related to certain intrusions and dikes. Their exact nature has not as yet been determined. Secondary enrichment has occasionally been demonstrated, due probably to the intrusions. Some veins show characteristics which would rather class them as fahlbands. All of them have been subject to oxidation and when exposed form large outcrops of reddish or brownish color; these outcrops, however, have been largely eroded and are now mostly covered.

The sulphur in the ore varies from 15 to 40 per cent., some of the highest-grade ore being found at the old Stella mine. This particular deposit is also notable on account of its freedom from such injurious accessories as arsenic, zinc, copper, and pyrrhotite; the latter occurs occasionally but

only in small pockets. A large deposit has been found at High Falls, three or four miles in a northeasterly direction from the Stella, but it contains a good deal of pyrrhotite. Some 11 or 12 miles southwest of the Stella, at the southern extremity of the belt, the America mine is situated. (This deposit has been described heretofore as the "Cole Mine.") The ore mined is quite free of impurities but is rather low in grade. The physical characteristics of these ores are similar, all being crystalline and intermixed with decomposed gneiss or clay; there is but little quartz in evidence as gangue.

Practically all the ore produced in St. Lawrence county is subjected to concentration, there being but a limited amount of crude ore shipped. The latter is marketed as "Spalls," i. e., ore broken from 1½ to 2½ in. size. These "spalls" carry approximately 32 to 33 per cent. sulphur. The concentrates vary from 38 to 48 per cent., the higher grades being greatly in demand on account of their purity and self burning qualities; they form the highest grade of pyrites on the market. The cinders from these concentrates contain less than 1 per cent. sulphur.

For a number of years mining operations have been conducted at the High Falls, Stella and America mines (the last situated at Richville), certain of these having been started 30 years ago. All of them, however, have had financial difficulties. It is only quite recently that the old properties have come into life again, having been taken up by some large financial interests.

The St. Lawrence Pyrites Company was the first to enter the field in 1905 by acquiring the old Stella property, to which it has added considerable holdings aggregating today about 25,000 acres. In the fall of 1906, the American Pyrites Company was organized to acquire the old Cole mine which had previously been operated by the Adirondack Pyrites Company. The American Pyrites Company is controlled to a large extent by the same financial interest as the St. Lawrence Pyrites Company. In the summer of 1907, the Oliver Mining Company obtained control of the old High Falls mine. At present the St. Lawrence Pyrites Company only is active in that district.

The American Pyrites Company, which operated the America mine during the first half of 1907, has since closed down, as the ore was playing out. Since then, more exploratory work has been resorted to, in order to prove the continuance of the ore formation on some adjoining property where it was found, but it is questionable whether the American Pyrites Company will again enter the market as a producing factor. The mill is inadequate and the erection of an entirely new concentrating plant would be imperative.

The Oliver Mining Company, which assumed control of the High Falls and part of the adjoining Crane property, discovered by diamond drilling a large continuous orebody near the Grasse river. Some of the ore was

subjected to mill tests in the old High Falls mills. Recently the property has been closed down, and it is understood that operations will not be resumed in the near future.

The St. Lawrence Pyrites Company started operations by unwatering the old Stella No. 2 mine, which hàd been the main producer in times past. On the ore deposit, which pitched about 30 deg. to the northwest, a shaft some 800 ft. deep had been sunk, from which drifts at 50 to 70 ft. intervals were started, the ore being removed by breast-stoping, leaving pillars to support the roof. The mine was thoroughly explored, a large amount of ore being thus exposed. Connection was also made with the so-called Stella No. 1 mine, situated about 800 ft. to the south, which proved that both mines were on the same vein. Active exploratory work was also started on the Anna vein, underlying and about 1600 ft. southeast of the Stella. The inclined shaft, known as No. 7, was sunk about 450 ft. and the orebody further developed by five levels 75 to 100 ft. apart. This deposit averages 24 per cent. sulphur and is 20 ft. in width.

In view of the results obtained, it was deemed advisable, in the spring of 1906, to lay out plans for the production and concentration of 250 tons of ore per day. The problems confronting the company were manifold. The proper location of the power plant and mill, adequate storage for tailings, the concentration of the ore, and the transportation of the finished product, presented difficulties which were hard to surmount, for the mines were considerable distances apart and the topographic and climatic conditions were both unfavorable. It was finally decided to establish a central power station which would supply electric energy to the shaft houses, mill, pumping station, etc. The original Stella siding was rebuilt and extended, so that the crude ore could be transferred from the two principal producing shafts (Anna No. 4 and Stella No. 1) to the mill, the tailings from the mill deposited along the railroad, and the finished concentrates sent to De Kalb junction for shipment. It was also considered advisable to extend the road into Hermon, so that a general transportation service could be carried on.

A site for the mill was selected on the slope of a ridge in close proximity to the water supply, easy of access to the railroad, and with a view of obtaining all the space required for the clarification of the mill waters. Actual construction work on the entire plant was started in the summer of 1906 and the first unit was ready by March 15, 1907, when the mill was put into commission.

Since then, active diamond drill work has proved the presence of a large additional amount of ore, so that the daily production of the plant will now be increased to 400 or 500 tons.

Power is conveyed to the central sub-station from the Hannawa Falls Power Company's plant (situated on the Racquette river about 18 miles

distant) in form of three-phase currents at 20,000 volts potential and is "stepped down" by two 250-kilowatt General Electric transformers.

Mining.—The Stella mine is operated from No 1 shaft, which is equipped with a good sized shaft-house; the capacity of the ore bin is 250 tons. The ore is hoisted in $1\frac{1}{2}$-ton skips which unload over a 3-in. grizzley, the oversize being broken in a 20x10 Blake crusher; the undersize and the broken material from the crusher falls into the bin, whence it is loaded in drop-bottom, 25-ton, standard ore cars. A Lidgerwood single drum hoist is used.

The Stella orebodies are quite flat, averaging 28 deg. from the horizontal, so that levels must be run at close intervals for the purpose of securing the ore, which is mined in breast stopes not exceeding 50 ft. in hight. The hanging wall is well defined and solid so that few pillars are required to support it. The mine makes but little water and this is taken care of by a small, electrically-driven centrifugal pump; a Dean triplex pump handles the water of the Stella No. 2.

The drills are operated by compressed air supplied by an 18x11x18 Laidlaw-Dunn-Gordon duplex, class 18 compressor, located at the Stella engine house; an Ingersoll-Sargeant, 16x18, type "A," steam driven machine is used as an auxiliary. Ingersoll-Rand D-24 drills are used.

The shaft and mine levels are lighted by electricity. The ventilation is very satisfactory, due to the connection between No. 1 and No. 2 shafts. The No. 1 shaft is now being sunk and a connection will be made between No. 1 and No. 2 mines on the 700-ft. level, so that ore can be removed from the lower mine and hoisted through No. 1 shaft. The Anna mine is accessible from shafts Nos. 4 and 7. No. 4 serves for hoisting purposes exclusively, while No. 7 is used for pipe and cable lines and the movement of the crews. No. 4 shaft is also equipped with a large shaft-house with the same bin capacity as the Stella, a 3-in. grizzley, and a 24x14 Blake crusher. The skip and hoisting engine are the same as the ones used at the Stella. The shaft is on a 45 deg. slope and is down about 250 ft. The levels are approximately 75 ft. apart. Mining is done by stoping and milling. Several parallel veins on the hanging wall side of the Anna have been intersected by a crosscut, started from No. 4 shaft, and these are now being developed. All the ore mined will be tributary to that shaft. The mine makes little water and this is taken care of by an electrically-driven centrifugal pump and an air-driven Cameron pump.

Experiments have been made of late with hammer drills which, should they prove successful, will change the method of mining to overhand stoping, provided the deposits do not flatten to any extent. This would mean a considerable saving in the cost of mining. The compressed air is furnished and conveyed from the Stella engine house. The equipment of each mine is capable of handling from 400 to 500 tons of ore per day.

Milling.—The ore is carried from the mines to the mill by the railroad, where it is dumped into a receiving bin. The mill, as already mentioned, was originally built with a capacity of 250 tons of crude ore per day, but this has since been increased. From the crude ore bin, which has a capacity of 1200 tons, the ore is drawn over a pair of grizzleys, spaced with 1-in. openings, the undersize falling into the boot of a Gates elevator, while the oversize goes through a No. 5, style "K," Gates crusher, which reduces the material to $1\frac{1}{2}$ in. By means of the Gates elevator, the crushed product as well as the undersize of the grizzleys is elevated into the crushed-ore bin, after the 1-in. oversize has been eliminated, by means of a trommel screen, whence it is fed to Allis-Chalmers 14x24 "C" roughing rolls. The reduced material is then returned to the crushed-ore bin by means of the Gates elevator. From this bin the ore is drawn by a 12x24 plunger feeder which delivers it, by means of a shaking screen, to a set of 36x15 type "A" rolls, which are set at about one half-inch. Water is used to assist the sizing of the material. The reduced material and the fines of the shaking screen are delivered, by means of the No. 1 mill elevator, to two conical trommel screens; these screens are arranged in pairs and are covered with punched steel plate with $\frac{3}{16}$ in. holes. The oversize is fed to three 2-compartment 24x36 roughing jigs, while the undersize is spouted to a pair of cylindrical trommels covered with punched steel plate with $\frac{3}{16}$-in. holes.

The roughing jigs make a middlings product only and this is reground by a 36x15 rigid type roll and is then returned to the No. 1 mill elevator, while the low-grade tails are eliminated from the system. Another advantage of the roughing jigs is that they will separate any foreign material which comes into the mill, such as spikes, bolts, pieces of iron, copper wire, etc.

The oversize of the No. 2 trommels is fed to the No. 2 Hancock jig; the undersize is carried to a centrifugal pump and elevated to a five-compartment spitzkasten, where the coarse material is separated from the slimes. The No. 2 Hancock jig will make clean concentrates on the first four hutches, while the fifth hutch produces the middlings, which require recrushing. The sixth hutch makes the tails.

The middlings from the fifth hutch are reground in a set of 36x15 rigid type rolls, the pulp being raised by the No. 2 elevator to a three-compartment spitzkasten. The coarsest material, which is drawn from the first and second hutches of this No. 2 spitzkasten, as well as the material from the No. 1 spitzkasten, is fed to the No. 1 Hancock jig, which will also turn out clean concentrates in the first four hutches, while the fifth hutch will produce middlings and the sixth hutch tails. The middlings from this No. 1 Hancock jig are recrushed on the 36x15 rigid rolls and follow the same rotation as the pulp of No. 2 jig. Four 4-compartment 18x30 Harz jigs have been so placed in the mill that they will concentrate either the

material fed to the No. 1 or No. 2 Hancock jigs in case of break-downs, or will help to take care of any excessive amount of feed.

The slimes from the last hutches of the spitzkasten are fed to the Overstrom tables and sometimes to one of the Harz jigs. All of the concentrates turned out by the different machines are spouted into the No. 3 elevator, which raises them sufficiently high to run them into the concentrating bins; these bins have a storage capacity of 700 tons. No. 3 elevator is equipped with perforated buckets which give the concentrates an opportunity to partly drain before they reach the storage bins; these bins allow for further drainage. (In winter the drying of the concentrates is accomplished by means of coils of steam pipes placed in the bins; this also prevents freezing.) The concentrates are spouted from these bins into box cars, or gondolas, for shipment to the trade. They usually retain from 2 to 3 per cent. of moisture and vary from 40 to 48 per cent. in sulphur.

The tailings are taken care of by either a centrifugal pump or an elevator, whence they are spouted onto the tailings pile, located beyond the mill. They may also be loaded in drop-bottom cars and deposited along the railroad. These tails retain from 5 to 10 per cent. sulphur, the extraction varying from 70 to 80 per cent.

All the mill waters are centered into a large settling pond, where they deposit a large amount of slimes. A considerable amount of the water so clarified is returned to the mill by means of two directly-connected Worthington centrifugal pumps of a combined capacity of 700 gal. There are also several auxiliary pumps installed in the mill for reclaiming purposes. The water is delivered into two tanks, which are located on the two upper floors of the mill, and from there it flows to the different machinery.

The mill is heated by steam which is generated in two 80-h.p. boilers, located in a boiler house adjoining the mill. This battery of boilers furnishes the steam for the auxiliary compressor, also for a couple of steam-driven auxiliary pumps. The mill is a spacious frame building, well lighted and covered with asbestos (this has been found to be very efficient under the existing climatic conditions). Special features of the mill are the large bin capacity for crude and crushed ore, and concentrates, and the fact that all the heavy machinery, such as crushers, rolls, etc., is located on the bottom floor, which has a solid concrete foundation. The machinery in the crushing and concentrating department is driven by independent motors, and the elevators are also run by a separate motor.

A very close check is kept of the sulphur contents of the material delivered to the mill and of the product of the different machines. An up to date sampling outfit and laboratory, and a well equipped warehouse, shops, etc., form part of the plant.

The concentrates are mostly shipped to the East and middle West; however, some shipments have been made as far west as Duluth, via the Great Lakes.

TALC AND SOAPSTONE.

By Claude T. Rice.

Under this heading are comprised two distinct varieties of mineral, viz., the fibrous talc which is produced only in New York, and the amorphous steatite, which is produced elsewhere. Outside of New York, talc is produced in North Carolina, Georgia, Massachusetts, New Jersey, Pennsylvania and Vermont. Virginia continues to be the largest producer of soapstone slabs.

STATISTICS OF TALC AND SOAPSTONE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.						Imports.		
	Fibrous Talc.			Talc and Soapstone. (b)					
	Sh. Tons.	Value.	Per Ton	Sh. Tons.	Value.	Per Ton.	Sh. Tons.	Value.	Value Per Ton
1896..........	51,816	$256,080	$4.94	21,448	$207,085	$9.66	1,950	$18,693	$ 9.60
1897..........	52,836	283,685	5.37	27,068	259,948	9.60	779	8,423	10.54
1898..........	54,807	285,759	5.21	27,974	237,280	8.48	445	5,526	10.70
1899..........	57,120	272,595	4.77	26,682	241,267	9.04	254	3,534	13.91
1900..........	45,000	236,250	5.25	26,726	249,777	9.35	79	1,070	13.50
1901..........	69,200	483,600	6.99	28,643	424,888	14.83	2,386	27,015	11.32
1902..........	71 100	615,350	8.65	26,854	525,157	19.36	2,859	35,336	12.36
1903..........	60.230	421,600	7.00	26,671	418,460	15.69	1,791	19,677	10.99
1904..........	65,000	455,000	7.00	27,184	433,331	15.94	3,268	36,370	11.13
1905..........	67,000	519,250	7.75	40,134	637,062	15.87	4,000	48,225	12.06
1906..........	64,200	541,600	8.43	58,972	874,356	14.82	5,643	67,818	12.02
1907..........	59,000	501,500	8.50	....	....	....	10,060	126,391	12.56

(a) Statistics for 1902 and subsequent years, are as reported by the United States Geological Survey, except that fibrous talc is as reported by the New York State Geological Survey. (b) The value of these products has not much significance owing to the diverse conditions of the material reported.

Prices and Market Conditions.—During 1907 the prices of talc were fairly steady. The prices according to quality and quantity were $15 to $25 per ton for American talc; $18 to $25 for French talc; and $25 to $40 for Italian talc. During the first nine months of the year the market conditions were normal, and the quantity of talc sold compared favorably with that sold during the same period of 1906. Everything seemed to point toward a normal year and a generally healthy condition of the trade, but in the fall came the panic and the accompanying industri al depression. Consequently the demand for talc fell off largely during the last three months.

Talc and Soapstone in the United States.

Arkansas.—No talc was produced in 1907; this was due to the fact that the light railway being built to connect the deposits in Saline county with the Rock Island system road at Maumelle was not completed during the year. These deposits are owned by the Arkansas Soapstone and Talc Company. This talc has been pronounced by G. P. Merrill, of the Smithsonian Institution, to be a very fine grade of mineral, much resembling the French.

Georgia.—The Cohutta Talc Company at Dalton produced rough talc, and manufactured slate pencils and ground talc. During 1907 the Georgia Talc Company was organized by Wayne Dumont, C. L. Newman and M. V. Cahill, of Paterson, N. J.

Maryland.—The Steatite Corporation has been organized to develop soapstone deposits near Marriottsville, in Carroll county. The property is being equipped with machinery capable, it is said, of producing 5000 sq.ft. of sawed slabs, and 40 to 50 tons of powdered talc per day. The slabs will be used mainly for making electrical supplies and laundry tubs. The main offices of the company are at Baltimore; R. S. Baldwin is president and C. A. Williams, general manager. The Soapstone Products Company has been organized, as a sub-company, to market the product from the quarries. The Cecil Mineral Company was organized during 1907 to produce slab talc, crayons and pulverized talc from deposits near Conowingo, in Cecil county. John W. McCauley of Baltimore is president of the company and also the engineer in charge.

Massachusetts.—The Massachusetts Talc Company was the only producer in this State in 1907. The New England Talc Company at Worcester ceased producing about two years ago. The mines of the Massachusetts Talc Company, which was organized in 1905, are situated on the southeastern side of Round Top mountain in the town of Rowe, Franklin county, and about four miles from Zoar station, on the Fitchburg division of the Boston & Maine railroad. The rocks in the vicinity are schists and gneisses of Silurian age. The talc vein has a northeast-northwest strike, and dips 55 to 60 deg. to the southeast. It is said to be 18 to 28 ft. wide, and has well-defined walls of schist. The deposit has been developed by a two-compartment, square-set shaft, 14x7 ft. in the clear, sunk to a depth of 200 ft. following the footwall of the vein. On the 200-ft. level a drift 100 ft. long has been driven following the vein. The talc is said to be of good grade. The property is equipped with a modern mill of good design, having a 100-ton gyratory crusher, four vertical pulverizing mills, whose capacity is four tons per hour, three elevators, conveyers, etc. This mill is built in two units; each unit is driven by a 125-h.p.

engine, but during 1907 only one-half of the mill was running. The entire output in 1907 was pulverized.

New York (By D. H. Newland).—The mining of fibrous talc continues to be limited to St. Lawrence county, N. Y., which has furnished for many years the entire quota of that form of talc for domestic consumption, as well as for export. The annual product averages about 65,000 tons. During 1907 it was somewhat less, owing to curtailed milling capacity. The largest mill in the district, the Hailesboro, owned by the International Pulp Company, was destroyed by fire in 1906, and although the company immediately added to the equipment of its mill at Dolgeville, the output could not be maintained at the usual rate. The decreased production had a favorable effect upon prices which were higher than for several years past, ranging around $8.50 per ton at Gouverneur, the usual shipping point.

The erection of a new mill at Hailesboro has been under way and it will be completed early in 1908. It is planned for a normal capacity of 100 tons of ground talc per day, and will more than restore the former milling facilities of the International Pulp Company.

The Ontario Talc Company is now the only independent producer in the district, as the Union and United States companies have been consolidated with the International. The present mill of the Ontario company is situated at Fullerville, but reports have been current that a new mill is to be built at Gouverneur during the coming spring. Additional mining capacity has already been provided for by the opening of the Potter mine on the Van Namee farm, 1½ miles below Fullerville.

While fibrous talc is the main product of the mines, the foliated variety is also found in quantity. This is prepared separately, requiring long continued grinding to reduce the flakes to the proper fineness, and finds special use in the paper trade, where it is employed in the place of ground mica for giving a lustrous surface to wall paper. The fibrous talc is marketed among manufacturers of book and writing papers in the United States, Germany, England, France, Austria and other countries. Numerous other uses are made of St. Lawrence county talc, but the requirements outside of those specified are comparatively small.

An occurrence of amorphous or earthy talc, near Natural Bridge, Jefferson county, was under exploration during 1907, but no commercial shipments were made.

North Carolina.—Near Glendon, the American Talc Company, the Glendon Mining and Manufacturing Company, and the Croatan Talc Company operated during 1907, but the Croatan company did not market any mineral, being engaged in experimenting with its talc as to powders of different colors and mesh, slabs, bricks and briquettes. The mill is almost ready to run. The North Carolina Talc and Manufacturing Com-

pany at Hewitt, N. C., was the largest producer; some of its talc was sold as crude, but most of it was manufactured into slabs, or ground into powder. During the year G. W. Hinshaw sold his half interest in the talc mine near Glendon, formerly owned by him and Edward Binney of New York, to Mr. Binney. The Southern Talc Company was incorporated by J. H. and W. A. Stoddard of Chicago, Ill., and F. G. Hoffman of Beta, to work a talc deposit near Addie. At Crabtree, W. H. Silver & Co. are preparing to develop their talc deposit.

Vermont.—The International Mineral Company, formerly operating at Moretown, closed down its plant in 1907. The United States Talc Company did not produce during the latter part of the year owing to the burning down of its mill at Rochester. The Stockbridge Company, working a deposit near Stockbridge, also closed down during 1907. The Vermont Talc and Soapstone Company produced a small quantity of rough talc during the latter part of the year, but did not ship any. At the end of the year this company was about to commence the milling of its talc.

Talc in Foreign Countries.

The chief sources of talc in Europe are in the Pyrenees and in the Italian Alps. The French talc is generally shipped from Bordeaux, while the Italian is shipped from Genoa. This foreign talc is imported in the form of rough rock, powdered talc, talc stones for making gas burners, and tablets for use in metallurgical work. The supply in France is abundant, and in Italy, although less talc is produced than formerly, it is still plentiful.

Austria.—It is reported that large deposits of white talc of good grade have been found in the province of Styria, in the commune of Floing. The deposits have been purchased by M. Elbogen, of Vienna, who owns the concessions of all the talc mines in Austria. During the year M. Elbogen made arrangements to increase greatly the capacity of the grinding mills belonging to him so that he can increase the output of his mines.

Brazil.—During 1907 both European and American dealers in talc showed considerable interest in the talc deposits of Brazil. The deposits are said to be extensive and the output depends mainly upon their accessibility. Practically all the Brazilian output of talc comes at present from deposits in the mountains of the State of Sao Paulo, near the cities of Sao Paulo and Loreno, especially from the deposits near the latter city. The talc is graded into three qualities according to color; all three grades are of good quality. The rough stone sells in Rio Janeiro at from $25.42 to $37.20 per ton.

France.—The most important talc deposits are near Luzech. As these deposits are situated in the mountain at an altitude of 1500 to 1800 m., they are worked only in good weather, or about five months out of the year. During 1907 the production was much larger than in 1906.

Occurrence, Properties, Uses and Methods of Mining Talc and Soapstone.

Occurrence.—Talc, steatite, or soapstone, as it is differently called, is a hydrous silicate of magnesium, having a greenish, whitish, or gray color. It is derived from the alteration of pyroxene amphibole, muscovite, enstatite and other magnesium silicates, and frequently occurs associated with dolomite, serpentine or magnesite. Iron, aluminum and calcium sometimes occur as impurities in the talc; of these aluminum and iron are deleterious. Talc is of two kinds, the fibrous kind, used in the manufacture of paper, and the more compact form generally known as soapstone. The first soapstone produced in the United States is said to have been quarried in New Hampshire about 100 years ago. In 1812 slabs of this soapstone were being shipped to Boston to be used in making stoves. The main supply of slab soapstone in the United States comes from Albemarle county, Virginia, where the quarrying of soapstone began about 1883.

The best soapstone and talc powder comes from Italy but the Italian output has decreased somewhat in recent years. The talc from France is considered the next best in quality; the supply of talc in France is said to be large. The most important deposits of the United States are in St. Lawrence county, New York. But talc deposits are found in many isolated occurrences along the east slope of the Appalachian mountain system. In Georgia, where is the most southern of these deposits, the talc is soft, but the mineral hardens to the north until in Campbell county, Virginia, it reaches a maximum. Outcrops occur in Carroll county, Maryland, near Easton, Penn., and also in New Jersey, Connecticut, New Hampshire, Vermont, and finally near St. Johns, Newfoundland.

Properties and Uses.—Owing to its chemical composition, talc is highly resistant to all ordinary acids except hydrofluoric. It is highly refractory, and is unaffected by heat, for it neither expands nor contracts; besides, it is a good non-conductor of heat as well as a non-absorbent of liquids, but it absorbs oil and grease. It does not discolor with age. It has a great dielectric strength, for it takes 30,000 to 40,000 volts to pierce a piece ½ in. thick. When highly heated it loses the small amount of water that it contains, and becomes much harder, assuming a greenish color and becoming capable of taking a high polish.

While talc is marketed as rough blocks, sawed slabs and powder, it is

used only in manufacture as slabs and as powder. In the form of slabs talc, or soapstone, is used for making laundry tubs, hearthstone, crucibles, mantels, sinks, griddles, slate pencils, pencils for marking on glass, tailor's chalk, gas burners, acid tanks, fire-bricks, laboratory tables, foot-warmers, and in electrical work, for switchboards, etc.

In powdered form, talc is used in the manufacture of paper, paint, soap and dynamite, for foundry facings, for making non-conducting pipe-covering and plaster, for polishing glass, for removing grease spots from fabrics, for dressing skins and leather, as the basis of many face, foot and tooth powders, as a lubricant, and as a sizing in cotton cloth. Talc is also used as an adulterant in sugar, baking powders and flour, especially to give yellow flour a lighter color. Such a use of talc was the cause of elaborate investigation by the French government in 1907.

In China a good deal of soapstone is used in making images, for, when it is unbaked, talc is soft and easily shaped. These images upon being baked, become hard so that they take a good polish, and assume that greenish color characteristic of Chinese images. Talc is also used in the manufacture of tiles and enameled bricks. New uses are found for talc from time to time or old uses revived so that, although the output of talc is increasing rapidly, the demand increases with equal rapidity. In recent years the main increase in the use of talc as a powder has been in the making of paper, while the use of soapstone in slab form is rapidly increasing in the electrical industry. For paper manufacture talc is rapidly driving out the more expensive china clay, which formerly was the main filler used. The fibrous and foliated varieties of talc are especially valuable for this purpose, because 75 to 90 per cent. of the amount of talc added to the pulp is retained in the manufactured paper as against 30 to 35 per cent. when china clay is used. Besides, the fibrous nature of the talc (which, although apparently granular when ground to 100-mesh size, is readily seen, when viewed with a microscope, to retain its fibrous character) increases the strength of the paper and does away with the brittleness, characteristic of paper weighted with clay.

In China and in Japan, it is said that powdered talc has long been used in paints to preserve woodwork, and buildings or monuments made of stone liable to disintegrate in the atmosphere. This use of talc as a pigment is being revived in the United States and Europe, for the protection of iron, steel, wood and sandstone, for, inasmuch as talc is unaffected by heat, fire, frost, air, or ordinary gases and acids, it is excellent for such purposes. When mixed with any quick-drying varnish of good tenacity and hardness, it gives to the surfaces to which it is applied an enamel-like finish that is both handsome and enduring. Talc paint flows easily, and is said to take hold of iron and steel surfaces with great strength.

The value of powdered talc depends on its color and other properties,

so that physical and not chemical qualities are used in grading it. The better grades have a bluish tinge, while the poorer have a yellowish shade. Talc in slab form is more valuable than powdered talc.

Mining Methods.—Talc or soapstone generally occurs in large, flat-lying deposits without any well-developed cleavage. Consequently to obtain a large proportion of slabs from the deposit it must be quarried rather than mined. But, when the talc is used mainly for making powder, the usual mining methods are employed in breaking the mineral as in St. Lawrence county, New York, where almost all the talc now comes from underground workings.[1]

In the other deposits of the United States quarrying methods have been more generally used than mining methods. At some of the Virginia and Maryland deposits channeling machines are used in quarrying the soapstone, because the talc has no cleavage and therefore has to be cut on all sides into blocks in order to obtain a good percentage of the talc in slab form. These channeling machines are run by steam; two types are used, undercutters for horizontal and swivel-head machines for vertical cutting. Recently a combination machine has been put on the market which is adapted to both kinds of cutting; owing to the light blow struck, this is said to be especially good for working deposits near the surface where the floor is uneven and the rock soft.

The blocks after being channeled are taken to the gang saw where they are sawed into slabs, $1\frac{1}{2}$ in. thick. These slabs are cut into different sizes, and shipped in that condition, or are planed, grooved into their proper shape, and then finished on the rubbing bed. The pieces are then assembled and cemented together; in order to hold the slabs securely in place wooden screws are used, although their presence would never be suspected in the finished article.

---

[1] *The Mineral Industry*, XIV, p. 529.

TANTALUM.

Tantalum in its metallic form was until recently almost unknown in commerce. Its use in the preparation of filaments for incandescent lamps, however, has brought it into prominence. The most important minerals of tantalum are tantalite and columbite. The former is a tantalate of iron and manganese, in varying proportions, in which the iron is frequently entirely replaced by manganous oxide, and the tantalic oxide by tin, zirconium or niobic oxide. Columbite is usually found associated with tantalite, and is essentially a niobate of iron and manganese. There are numerous other minerals carrying tantalum, but none of them has become of commercial importance.

In the United States tantalite and columbite are known to occur in the mica and tin deposits of the Black Hills, South Dakota, in Fairfield county, Conn., in North Carolina and in Colorado. In the last named State the occurrence is in Fremont county, near Parkdale. The tantalite occurs in a vein about 4 ft. in width, with a schist hanging and a granite footwall. Tantalum-bearing minerals occur in the tin-bearing district of the Northern territory, South Australia; also in the alluvial wash at Greenbushes, Bunbury, West Australia.

The minor uses for tantalum cover many industries, especially those in which a metal of great hardness, high melting point, and immunity from attack by most chemical reagents is required. By far its most important use, however, is in the manufacture of filaments for incandescent lamps, on which subject much has been written during the last year. The advantages claimed for the tantalum filament lamp are its high efficiency, the whiteness of the light produced, and its ability to withstand a current considerably in excess of the normal.

The production of tantalum ores in the United States rarely amounts to more than a few tons per annum, and most of this is shipped to Germany. It is scarcely possible to give any definite information on the commercial value of tantalum minerals, as there are few manufacturers using them and the market is limited. It appears that the ores are bought on a basis of at least 60 per cent. of tantalum pentoxide; niobium oxide should not exceed 3 per cent. and chromium should be entirely absent. Ore conforming to these requirements should bring in the neighborhood of $1 per kilogram.

TIN.

By W. R. Ingalls.

The tin-mining industry in Alaska, South Dakota, and elsewhere in the United States, made no material progress in 1907. The only production was from Alaska and South Carolina. The total amounted to 63 tons of ore, valued at $15,200, against 10 tons, valued at $3043, in 1906. No metallic tin was produced in the United States in either year, except a few tons made experimentally.[1] It is to be hoped that the smeltery at Bayonne may be put in operation sooner or later on foreign ores, for which it was originally intended, so that the United States will at least smelt part of the large amount of tin which it annually consumes. There were the usual reports in 1907 of rich tin discoveries in Mexico, but, as I have repeatedly emphasized, these are only small pockets of the rich *guijilos* that occur in rhyolite tuff, and there is slight prospect that Mexico will ever be a tin producer of any consequence, at least not from the rhyolite-tuff formation, which predominates in Durango and Zacatecas.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1899...	71,248,407	$16,748,107	1902...	85,043,353	$21,263,337	1905...	89,227,698	26,316,023
1900...	69,989,502	19,458,586	1903...	83,133,847	22,265,367	1906...	101,027,188	37,446,508
1901...	74,560,487	19,024,761	1904...	83,168,657	22,356,896	1907...	82,548,838	32,075,091

Consumption.—One of the large uses of the world's production of tin is in the manufacture of tinplate in Wales and the United States. There are no absolutely reliable figures available as to the output of tinplate in Wales, but it is estimated by the *Ironmonger*, of London, at 650,000 long tons, requiring 11,600 tons of tin. The production of tinplate in the United States was about 600,000 tons, requiring 10,800 tons of tin. Consequently, the tinplate trade of Wales and the United States consumed approximately 22,400 tons of tin. The production of tinplate in the United States in 1906 was 577,000 tons, against 494,000 in 1905 and 458,000 in 1904.

[1] This refers only to virgin metal. A considerable quantity of tin is recovered from waste products.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (*a*)
(In tons of 2240 lb.)

	1899	1900	1901	1902	1903	1904	1905	1906	1907
English production	4,013	4,268	4,566	4,392	4,282	4,132	4,468	4.522	4,700
Chinese exports	......	......	......	......	......	......	4,482	4,052	(*b*) 4,000
Straits to Europe and America	44,460	46,058	50,339	51,831	52,212	57,419	56,840	57,143	52,520
Straits to India and China	1,484	1,785	2,655	1,882	3,123	3,261	1,484	1,292	3,140
Australia to Europe and America	3,337	3,235	3,345	3,199	4,934	4,846	5,028	6,482	6,612
Banka sales in Holland	9,066	12,631	14,978	14,978	15,070	11,363	9,960	9,286	11,264
Billiton sales in Java and Holland	5,057	5,882	4,387	3,897	3,650	3,215	2,715	1,968	2,229
Bolivian arrivals on Continent	813	1,900	9,670	10,150	9,630	12,978	14,245	16,394	15,500
Bolivian arrivals in England	3,940	5,065							
Totals in long tons	72,170	80,824	89,940	90,329	92,901	97,214	99,252	101,139	99,965
Totals in metric tons	73,325	82,117	91,379	91,774	94,387	98,769	100,840	102,757	101,564

(*a*) Compiled from commercial reports. There is also a small production in Germany. The apparently large increase in the total for 1905 is due to the inclusion of the Chinese exports for the first time. (*b*) Estimated.

Tin Mining in the United States.

Alaska.—There were some small shipments but no significant production of tin ore from this Territory in 1907. The Bartels Tin Mining Company confined its operations wholly to development work. In the North Star mine it completed 1000 ft. of underground work, consisting of two adit levels, two shafts and connecting levels. The main adit is now 394 ft. long. A new adit was run during the summer of 1907 for a distance of 157 ft. What is believed to be a large orebody was crosscut in the main adit and in one of the levels another vein 6 ft. wide was cut. Various improvements were made on the surface to facilitate the transportation of ore from the mines to the mill. The three-stamp, experimental mill was run 13½ days, the mill test showing an average of 3½ per cent. tin. The company plans the installation of a 200-ton concentrating mill in 1908. The United States Alaskan Tin Mining Company took out about 40 tons of ore in the course of development work and expects to make a commercial product in 1908. Developments were also made by the Seward Tin Mining Company, the Alaska Tin Corporation (which obtained a small quantity of stream tin in 1908) and the American Tin Mining Company (which made a small shipment of ore).

California.—English capitalists, who had an option on the Temescal mines, caused an examination of them to be made.

South Carolina.—Capt. S. S. Ross made some small shipments from his mine near Gaffney. The mine is worked through a shaft 150 ft. deep. The product is shipped to Swansea, Wales, via New York.

South Dakota.—Nothing of consequence was done in tin mining in this State in 1907. The Gertie mine has been tied up in litigation, which has now been settled in favor of E. C. Johnson, who is making plans to operate the mine. The Tinton Tin Company is also contemplating some development work. A discovery of cassiterite in association with amblygonite, near Keystone, was reported in 1907.

Washington.—The discovery of tin ore at Silver Hill, southeast of Spokane, in 1907 attracted considerable attention. According to Arthur J. Collier of the U. S. Geological Survey, who was detailed to investigate the discovery, the cassiterite found at Silver Hill is nearly black and without definite crystal outlines. It is distributed through a nearly white, fine grained rock, in grains or masses ranging in size from that of a pin head to that of a boulder several inches in diameter. Tin ore of this type has been found at four localities within the area described by Mr. Collier. In the opinion of Mr. Collier, the development at Silver Hill indicates that the tin ore occurs in detached masses, whose relations to each other cannot yet be determined.

The Spokane Tin Mines Company, of Spokane, mined during 1907 about 120 tons of ore that is expected to average between 5 and 7 per cent. metallic tin. The company as yet has no milling facilities, so has not marketed any of its product. The ore was taken out in development work. On the 100-ft. level there is said to be an ore shoot 30 ft. long and 2 to 5 ft. wide, but at the depth of 140 ft. the showing was not so good.

Tin Mining in Foreign Countries.

Australia.—The production of tin and tin ore in New South Wales in 1907 was 1914 tons, against 1671 tons in 1906. The production of tin in Queensland was 5140 tons in 1907 against 4823 tons in 1906. The production of black tin in Western Australia in 1907 was 1624 tons; in Victoria, 100 tons; in the Northern Territory, 400 tons; in Tasmania, 4343 tons. The total value of the tin production of Australia in 1907 was £1,497,582, divided as follows: New South Wales, £293,305; Victoria, £10,275; Queensland, £496,766; Northern Territory, £36,907; Tasmania, £501,681; Western Australia, £158,648.

Northern Territory: According to an Australian paper, quoted by the *Mining Journal*, Dec. 21, 1907, there are about 100 men on the Western Arm tin field at present, where wages for white miners average 11s. 8d. per day, and for Chinese 7s. to 8s. Within a radius of eight miles there are 20 payable claims, and new discoveries are frequently made. The climate is said to be much better than that of Cairns and Townsville, while food is good and the supply of vegetables and game plentiful.

As regards individual properties, little or no development has been done, with the exception of the John Grant, Good Hope, and Rocky Bar claims. In other cases the work done has been conditioned by the discovery of rich patches which have been picked out at the expense of future working. Thus, in one Chinese claim 300 tons of tin have been extracted, valued at £30,000, from a depth of less than 40 ft., without the installation of any machinery whatever. This method of working is caused partly by lack of

capital, and partly also, it would seem, by the character of the formation met with. The tin is said to occur, not in lodes, but in "blobs," yielding up to 100 tons of ore, after which the occurrence pinches out until another patch is met with.

An important find of tin in granite is reported from Ferguson river, 24 miles south of Pine creek.

New South Wales: The chief reason for the increase in the tin output of this State was the larger production of the dredges. Tingha is the principal center of tin mining, its output in 1907 having been 1812 tons, of which the 28 dredges in operation furnished 1439 tons. The Emmaville district produced 413 tons. It is expected that this district will make a much larger output in 1908 as the result of the operations of the four dredging plants which have been installed. The number of men employed in tin mining in New South Wales in 1907 was 3173.

Tasmania: Tin mining in this State is thoroughly discussed in a special article by Mr. Lewis, which follows this. At a recent meeting of the directors of the Mount Bischoff Tin Mining Company in Launceston it was decided to re-form the company for a new period of 100 years and to alter the articles of association with the view to enable the company to enlarge the scope of its operations by acquiring or leasing other tin mines. The rich surface alluvial deposits of the Mount Bischoff mine are practically worked out, but the manager reports that by cleaning up the old workings the plant can be profitably employed for the next six or seven years. In addition to the alluvial ore the mine contains large deposits of low-grade sulphide ore but no method has yet been found for working this class of ore profitably.

Banka.—The shipments of Government tin were 10,945 long tons in 1907, against 9807 in 1906, 9344 in 1905 and 11,749 in 1904. The shipments of private tin were 2432 tons in 1907, against 1935 in 1906, 2237 in 1905 and 3077 in 1904.

AUCTIONS OF BILLITON TIN HELD AT BATAVIA.

Date of Tender.	1905		Date of Tender.	1906		Date of Tender.	1907	
	Quantity Tons.	Av. Price Per ton.		Quantity Tons.	Av. Price Per ton.		Quantity Tons.	Av. Price Per ton.
		£ s. d.			£ s. d.			£ s. d.
Jan. 4..	245	125 19 11	Jan. 3..	123	156 8 10	Jan. 3..	184	184 0 9
Feb. 8..	244	124 5 1	Feb. 7..	123	161 9 6	Feb. 6..	184	186 0 10
Mar. 8..	244	126 9 3	Mar. 7..	123	158 0 7	Mar. 6..	186	186 0 10
Apr. 5..	306	130 4 2	Apr. 4..	123	165 11 7	Apr. 3..	184	178 0 2
May 3..	306	130 3 4	May 2..	184	176 4 0	May 8..	186	181 6 1
June 7..	246	130 9 5	June 6..	184	174 4 11	June 5..	186	176 7 0
July 5..	246	134 0 6	July 4..	184	172 8 6	July 3..	186	177 12 3
Aug. 9..	246	143 9 7	Aug. 8..	184	177 2 2	Aug. 7..	184	171 10 1
Sept. 6..	239	141 3 9	Sept. 5..	187	178 5 2	Sept. 4..	184	161 0 8
Oct. 4..	123	144 16 0	Oct. 3..	184	187 19 10	Oct. 2..	186	152 18 5
Nov. 8..	123	145 18 9	Nov. 7..	184	190 2 11	Nov. 6..	186	136 17 9
Dec. 6..	123	154 9 3	Dec. 5..	184	190 18 0	Dec. 4..	186	128 10 19

Bolivia.—The statistics of tin production in this country in 1907 and in previous years are given in the accompanying table. The general status of the tin mining industry is reviewed in a special article by Georg Hohagen, which follows this.

EXPORTATION OF TIN FROM BOLIVIA.
(In metric tons.)

Year.	Barrilla. Tons.	Metallic Tin. (a) Tons.	Year.	Barrilla Tons.	Metallic Tin. (a) Tons.
1898.....	4,327	2,596	1903....	21,785	13,071
1899.....	9,134	5,480	1904....	20,369	12,221
1900.....	15,088	9,053	1905....	27,690	16,614
1901.....	21,573	12,943	1906....	29,370	17,622
1902....	17,340	10,404	1907....	27,678	16,607

(a) Tin content of the barilla (black tin concentrate), computing the latter at 60 per cent. metallic tin.

According to H. A. Watson & Co., Liverpool, Eng. the arrivals of tin at Hamburg were 2390 long tons in 1907 against 1827 tons in 1906. At Havre the arrivals amounted to 924 tons in 1907 against 691 tons in 1906. The total arrivals of ore and metal at Liverpool from 1900 to 1907 are given in the accompanying table.

LIVERPOOL RECEIPTS OF BOLIVIAN TIN.
(In tons of 2240 lb.)

Year.	Bars.	Ore.		Total Tin.
		Crude Weight.	Metallic Content.	
1900.....................	1,507	5,431	3,530	5,037
1901.....................	1,730	9,086	5,905	7,635
1902.....................	1,685	10,961	6,576	8,261
1903.....................	1,614	10,401	6,240	7,854
1904.....................	1,573	13,824	8,294	9,867
1905.....................	1,386	17,504	10,504	11,888
1906.....................	1,569	20,489	12,293	13,862
1907.....................	1,143	18,532	11,119	12,262

Burma.—A survey of the tin mining industry in South Burma is being carried out by the Indian Geological Survey. The industry has shown considerable expansion during the last two years, the output of tin ore from the mines in the Mergui and Tavoy districts having advanced from 1495 cwt., valued at £9783, in the fiscal year 1905-06 to 1919 cwt., valued at £13,574, in 1906-07. The Mergui district is the center of the industry at present.

China.—The exports of tin in 1906 were 4052 long tons, against 4482 in 1905. This comes chiefly from the Ko-chiu-ch'ang mines, near Mengtze, in Yunnan. It passes through Tonkin, whence it is shipped to Hong-Kong where there are four native refineries which handle it. From Hong-Kong the product is distributed abroad and to other Chinese ports.

A recent British consular report gives the exports of tin from Meng-tze in 1906 as 3985 tons (of 2240 lb.) against 4462 tons in 1905. The decrease was due in part to the drought and also to the fact that the French railway offered as much as $1.20 a day in cash and rice for labor, causing a scarcity of miners. With the idea of applying modern machinery and methods to the tin mines of the Yunnan region a French financier visited the Ko-chiu-ch'ang mines and subsequently contracted for the privilege of working the tailings. It is feared, however that unreasonable jealousy on the part of the Chinese will make the introduction of such improvements difficult.

(By T. T. Read.) Yunnan is the principal producer of tin. The mines are at Ko-chiu-ch'ang, near Meng-tze. The ores are reduced locally and the "base metal" is shipped out through Cochin-China to Hong-Kong, where it is refined. The French railroad from Hanoi, in Cochin-China, now extends as far as Meng-tze, offering greatly increased facilities for transportation. Tin also occurs in Honan, Kuangsi and Fokien. The production in Kuangsi is small, and that of the others is unknown.

Congo (By John R. Farrell).—North of the copper zone of Katanga a tongue of granite intrudes, forming the Bia range of hills and beyond this is found the tin belt, extending for more than 100 miles in a northeast and southwest direction from the Lualaba to the Lufira river and nearly at right angles to the copper zone. It was found and traced by hard, close prospecting, but being at some distance from supply stations and local obstacles being in the way not much work has yet been done upon it. At a number of places, however, cassiterite has been found in alluvial wash consisting of angular fragments of pinkish quartz mixed with tourmaline, tourmaline schist and schorl rock. Boulders of cassiterite are frequent, the largest so far weighing 156 lb. and from that down to shot grains. This wash is found along the lines of strong quartz reefs in which bunches and grains of the tin ore have so far been found irregularly distributed, so these reefs are no doubt the source of the alluvial.

Enough is known to make it quite certain that a new tin field has been discovered which will be of commercial importance when transportation arrives. To date work has almost been confined to the Busanga area in the angle of the junction of the Lualaba and Lufira rivers, where 215 acres have been tested by more than 200 shafts, pits and holes, and found to have a gravel bed averaging 2 ft. 9 in. thick, containing 0.682 per cent. of cassiterite, or practically 6000 tons of metallic tin. The cassiterite contains from 63½ to 65 per cent. tin and it is unmixed with other metals. The Kasonsa area, further to the north, is known to be much more extensive and important than Busanga.

France.—It is said that the tin deposits at Montebras, in the Department of La Creuse, are to be developed by an Anglo-French company, the French Mining Syndicate, capitalized at £250,000. Engineers report three

types of deposits, viz., veins, averaging 4 per cent. tin, one of which extends for 200 m., with an average width of 2 m.; alluvial ground amounting to 7,950,000 cu.m.; and a white, feldspathic rock, averaging 1½ per cent. tin, which is expected to yield, as a by-product, a fine quality of china clay.

Germany.—Some of the tin mines of the Erzgebirge (Saxony and Bohemia) were worked in 1907. A prospecting company, after exploring some ancient workings at Frühbuss, near Neudeck, where tin was mined in the seventeenth century, found several veins at a depth of 23 m. The same company met with satisfactory results at Hirschenstand. Here the abandoned workings were more than 1 km. long, and in a good state of preservation.

Interest in the tin mines of the Erzgebirge of Saxony and Bohemia, has been revived. On the Saxon side there are old mines at Geyer, Schönfeld, Schwarzenberg, Seiffen, Zinnwald and Ehrenfriedersdorf, and on the Bohemian side at Platten, Barringen, Neudeck, and Joachimsthal.

Malaya.—During 1907, J. E. Scrivenor, geologist of the Federated Malay States, published a report covering his investigations from September, 1903, to January, 1907. In his conclusion he says: "It would appear that all are now agreed as to the future of tin mining in the Federated Malay States. The easily worked alluvial deposits are coming to an end, and on deeper-seated and low-grade surface deposits depends the continued prosperity of the industry. In the development of deposits in the bed-rock the miner is aided to a very great extent by the extraordinary depths to which rocks weather in this country, owing no doubt to the dense vegetation and the warm moist climate. That many tin-bearing granite modifications, rich in kaolin, owe their softness, wholly or partially, to something distinct from weathering may be deduced from a comparison with the kaolin deposits and granite masses of Cornwall; but we may safely assume that deposits such as those at Bruseh and Jeher owe a great deal to the tropical conditions."

The decline in the value of tin in the latter part of 1907 had a serious effect upon the mining industry of the Malay States.

The *Straits Times* recently published the views of "a well-known Chinese miner of Johore" on the position of the mining industry in the F. M. S. He classified the mines under three heads: (1) Mines that can not be worked when the price of tin is below \$50 per pikul; (2) under \$65 per pikul; and (3) under \$90 per pikul. Twenty-five per cent. of the mines in the F. M. S. come under the first heading, 35 per cent. under the second, and 40 per cent. under the third, and it is the latter two classes of mines which will soon be compelled to close down, as in many instances they are unable to recoup themselves for the losses already sustained. The result will be that the richer mines will have to satisfy the demands for tin in the F. M. S. as far as possible.

Douglas Osborne of Ipoh, expresses a different view, saying that it is not much of a mine that cannot be worked at a profit with tin at $60 per

ROUGH SKETCH MAP OF THE FEDERATED MALAY STATES.

pikul (equal to £120 per ton), especially since wages have been reduced all around. Where coolies have been discharged from one mine they have

been readily absorbed by others. Most mines can work at the present price of tin, while the reduced wages of the coolie are three times what it actually costs him to live, despite the inordinately high prices of commodities. It has also been brought to notice that a good many tributers, taking advantage of the fall in price, have reduced the number of their coolies in order to be in position to go to the land-owner and tell him that, unless the rate of tribute is reduced, coolies cannot be got to work. This is merely an attempt at a "squeeze," and not a genuine excuse, as is proved by the fact that other tributers working tailings only, and paying from 10 to 15 per cent., are working as usual. "The present price of tin is not such as to call for any alarm, still less to cause the shutting down of any but a few of the poorest mines. Things have been too prosperous, and we have got to learn to manage—as we have before, and can do again—with tin at a reasonable instead of an inordinately inflated price."

TIN PRODUCTION IN THE FEDERATED MALAY STATES. (*a*)
(In pikuls of 133⅓ lb.)

State.	1906.			1907.		
	Tin in bars.	Tin exported in the form of ore. (*b*)	Total.	Tin in bars.	Tin exported in the form of ore. (*b*)	Total.
	Pikuls.	Pikuls.	Pikuls.	Pikuls.	Pikuls.	Pikuls.
Perak	132,870	303,039	435,909	99,245	332,141	431,386
Selangor	116,968	151,655	268,623	82,093	191,807	273,900
Negri Sembilan	47,490	30,276	77,766	40,199	34,956	75,155
Pahang	9,420	25,068	34,488	12,620	20,575	33,195
Total	306,748	510,038	816,786	234,157	579,479	813,636

a As reported by the *Selangor Government Gazette*. *b* Computed at 70 per cent. of the gross weight of the ore.

PRODUCTION OF TIN IN THE FEDERATED MALAY STATES.
(In pikuls of 133⅓ lb.)

	1900	1901	1902	1903	1904	1905	1906	1907
Perak	355,590	385,060	405,870	430,296	443,507	446,781	435,909	431,386
Selangor	269,490	302,570	278,360	284,592	300,413	289,867	268,624	273,900
Negri Sembilan	82,320	75,230	73,520	85,461	84,849	85,133	77,766	75,155
Pahang	15,700	26,310	23,120	25,317	27,469	34,879	34,488	33,195
Total in pikuls	723,100	789,170	780,870	831,666	856,238	856,660	816,787	813,636
Total in metric tons	43,123	47,713	47,211	50,254	51,790	51,793	49,859	

According to Thornwell Haynes, the U. S. Consul general at Singapore, the decreased export may be more imaginary than real, because the output did not fall off to any appreciable extent. Certain it is that about 500 tons of the metal was held in the Federated Malay States at the end of 1907 by buyers awaiting a rise in price. The official returns are unavoidably misleading, in that the government takes cognizance only of exported tin, and when the buyers see fit, owing to low prices, to hold the metal at the

mines until a rise, the official returns do not accurately show the actual output.

There seems to be a consensus of opinion that there is no immediate possibility of the yield of tin rapidly increasing. This is chiefly due to the wasteful methods of the Chinese, but with the introduction of European capital and appliances a revolution may be looked for in the near future, for while the output from the Chinese-controlled mines will undoubtedly fall off, the returns from the European-controlled mines will increase. In fact, scientific methods of extraction are already beginning to have an effect in the returns of the various English-owned companies. The commercial agent for the State of Victoria, Australia, who recently made a tour through the Malay States, says:

"What particularly impressed me was the extent of alluvial tin-bearing country, which has been surface-worked in a primitive manner by Chinese, and its possibilities for reworking in a thorough and systematic manner by means of centrifugal-pump dredging plants, the same as those now in use in working old abandoned alluvial gold fields in Victoria. The ground has to be poor, indeed, which does not pay for reworking with these labor-saving, economical, Victorian dredging plants. Working night and day such plants, with six or eight men per shift, will do the work of 500 men as I saw them under Chinese system working, and what is specially important, will do the work better by saving more tin, and taking practically everything out of the material treated. My own opinion is that there is a great future for the tin fields of Malaya, provided these dredging plants are extensively introduced, and that this systematic and economical system of working would give profitable results to shareholders interested in their operations. This opinion I found shared by several of the leading men connected with tin mining in Malaya and who have either personal knowledge or know from other sources of what is being accomplished by means of these centrifugal-pump dredges."

The Eastern Smelting Company was formed among the principal tin-miners of the Federated Malay States to take over the stock-in-trade and good-will of Chin-Ho & Sons, tin-smelters, of Penang. The capital stock is $1,500,000. The directors consist mostly of Chinese tin-miners. The capacity of the works acquired was about 600 tons of refined tin per month; it was the intention to increase this to 1000 tons per month and introduce improved furnaces, machinery, etc.

Nigeria.—The province of Bauchi, in the western portion of which the tin has been found, averages about 4000 ft. in elevation. The rocks in the mining district are granites, diorite, felspathic gneiss schists, quartz and silicious ironstones (laterite), and concretionary ironstone. The minerals are cassiterite, columbite, zircons, and numerous iron minerals. All the workings are surface alluvial, the cassiterite being usually found to an

average depth of 12 to 15 ft. No lode has yet been discovered. At one workings the gravel, which varies from 10 lb. to the ton within 2 ft. of the surface to 40 lb. to the ton and over at 10 ft. below the surface, is washed twice. The concentrates are then taken to the dressing floors, where they are again washed by hand in large tubs. This gives nearly pure cassiterite. This, when dry, undergoes further cleaning by women, who pour it from a hight of 2 or 3 ft. into trays, thus separating the silica dust from the cassiterite by the wind. A small portion of the output is smelted at the mines. The plant for smelting consists of a cupola, having a capacity of about 6 cwt. per day. The blast is supplied by two Roots' blowers, driven by a small vertical engine. The cassiterite is first smelted by charcoal fuel, with a small quantity of red hematite as a flux, and after a sufficient number of bars have been obtained, they are refined by being again passed through the cupola with green wood as fuel. The total output of black tin (65 to 70 per cent. metal) for 12 months ending Jaunary 31, 1907, was 129 tons.

The progress of the tin mining industry is greatly handicapped by the primitive methods of transport. The black tin and ingots are carried from the mines to Loko, on the River Benue, almost entirely by carriers, who are only able to carry a bag of black tin weighing 62 lb. or an ingot weighing 65 lb. The distance is about 190 miles, and the journey takes 12 days. The Kano-Baro railway will at once give a new and shorter route, and in time no doubt the rail will run to the Bauchi highlands, and so bring the mining district within a couple of days of the Niger river. Until the supply of water in the mining district is conserved by the construction of dams, so as to enable the mining company to carry on operations throughout the year, the mining industry cannot assume large proportions.

Siam.—In this country tin is mined principally in the provinces of Puket and Kedah by two English and one Dutch companies. The annual production is about 5000 metric tons. Vast districts are yet unexplored, though probably their mineral wealth is considerable. In these places the government is constructing roads and improving the existing ones.

According to a British consular report the exportation of tin, including metal and the metallic content of ore (estimating the latter at 65 per cent.) from the monthon of Puket is about 4000 long tons per annum. In 1906 the export from the island of Tongkah alone amounted to 18,476 pikuls (1109 tons) of smelted tin, and 30,635 pikuls (1839 tons) of tin ore. The Straits Trading Company has a branch establishment at Puket, the capital of Junk Ceylon, and headquarters of the monthon of the same name. It exported 21,164 pikuls (1270 tons) of tin ore in 1906 to its smelting works in Penang. It is probable that, owing to smuggling and the want of sufficient revenue officers, 25 per cent. more ore is exported from the mainland than is accounted for in the official returns of the mines depart-

ment. It is estimated that returns for the year ending March 31, 1907, will show a considerable increase over those for the previous year, which amounted to 4231 tons. A contract has been made with Captain Miles, of Tasmania, by the government for dredging the harbor at Tongkah. The bed of the harbor is supposed to be rich in ore and Captain Miles will have the right to the tin on condition of dredging a dock 1200x850 ft. in extent and not less than 20 ft. deep at low water, and of making a channel of the same depth therefrom to the deep-water anchorage.

According to a more recent British consular report, the tin output of Siam in 1907 was, roughly, 2652 tons of tin slabs, valued at £442,954. Most of the tin ore was shipped by the Straits Trading Company's agent at Puket to its smelting works at Penang. The tin slabs were smelted locally in primitive Chinese furnaces. South Kedah produced 430 tons of metallic tin, and the rest of it came from the island of Junk Ceylon and the mainland to the north. Dredging operations have just commenced in the harbor of Puket. The price for the year was on an average £165 per ton in Penang. At the end of the year many small mines were shut down. The monthon of Puket has lately been attracting the attention of European prospectors. A large company has obtained a concession in Renang, and there are several Englishmen prospecting in the neighborhood of Junk Ceylon. Otherwise the mining is entirely in the hands of Chinese, and the methods employed are very primitive compared with those in vogue in the Federated Malay States.

Tin mining in Siam was discussed by K. Van Dort, of Bangkok, in an article in *Eng. and Min. Journ.*, of Oct. 19, 1907. He stated that there seems to be but a very slight appreciation of the potentiality of the Siamese States with regard to the future of the tin-mining industry, and it is only recently that active steps have been taken to investigate this source, with results, generally speaking, that have proved equal to the brightest anticipations. The chief difficulty which besets the Chinese miner at the very outset is want of capital, and consequently, in a country like Siam, where mining laws have never been enforced, they resort to methods which are detrimental to mining interests generally. For instance: A great deal of the tin produced in this country comes from districts where small gangs, or *Kongsis* operate independently, without obtaining any regular license or claim, wherever they choose, and in any manner possible, so long as they do not interfere with the agricultural interests of the natives. This leads to very unsatisfactory results, large areas being worked superficially for the contents of the upper strata only, as affording the quickest returns for a limited expense of labor, and then abandoned entirely for fresh ground. In the course of a few years, with the rapid and luxuriant growth of tropical vegetation, it assumes once more the normal appearance of virgin forest, and many prospectors have been misled in this way, little

suspecting that 15 or 20 ft. beneath the surface were rich leads which in spite of the heavy over-burden would amply pay reworking.

According to a Straits Settlements newspaper the projected railway from Hongkong will be likely to traverse the rich mineral regions of Siamese Malaya. Lang Suan has an important tin supply, while the same may be said of Renang. There are 70 mines in the region of Lang Suan, most of which are worked by natives, but the European concessions in the latter place, as well as in Renang, are exceptionally encouraging and already are giving excellent returns. There is no lack of capital, even the natives making themselves better acquainted with modern machinery and bringing it into use.

South Africa.—The Government *Gazette* of Nov. 29, 1908, contains an account of the State tin workings at Zaaiplaats, by U. P. Swinburne, the district inspector of mines, who states that the occurrence of tin may roughly be divided under three headings: (*a*) More or less isolated ore shoots or chimneys in the red granite, rich in cassiterite, the orebodies varying from a few inches to some feet in size. (*b*) Pegmatites and coarse granites containing large tin crystals covering a more or less extensive area, but with smaller and more irregular mineralization than the shoots containing the finer tinstone. (*c*) Alluvial tin, probably limited to rich patches over a comparatively small area. Five shoots of ore have been opened to date, and some 12 tons of tinstone are lying at grass. Upon these ore chimneys some 140 feet of sinking has been done. The ore varies from mere indications to several feet of pay rock, from a trace of tin to 60 per cent. cassiterite, and contains at present but few impurities.

United Kingdom.—The status of the mining industry in Cornwall in 1907 was excellently summarized in a series of articles in the *Mining Journal* of London. There was no particularly noteworthy change in the production and the decline in the value of the metal at the end of the year was felt keenly.

AVERAGE PRICE PER TON 2240 LB. OF ORE AT THE CORNISH TICKETINGS.

1907								1906							
	£	s.	d.		£	s.	d.		£	s.	d.		£	s.	.d
Jan. 14	110	10	3	July 15	114	10	11	Jan. 15	97	12	6	July 16	97	10	8
Jan. 28	113	18	11	July 29	110	9	7	Jan. 29	96	9	2	July 30	98	12	6
Feb. 11	114	17	5	Aug. 12	104	16	9	Feb. 12	96	15	10	Aug. 13	104	6	6
Feb. 25	117	4	10	Aug. 29	100	14	9	Feb. 26	96	16	1	Aug. 27	104	16	9
March 11	117	3	9	Sept. 9	102	8	6	March 12	97	6	7	Sept. 10	105	11	4
March 25	112	12	11	Sept. 23	102	7	10	March 26	99	3	3	Sept. 24	106	8	0
Apr. 8	114	16	0	Oct. 7	91	14	2	April 9	101	11	7	Oct. 8	113	5	10
Apr. 22	117	19	2	Oct. 21	82	11	4	April 23	104	11	0	Oct. 22	113	17	0
May 6	120	17	2	Nov. 4	86	3	6	May 7	113	7	6	Nov.	114	2	4
May 21	118	13	10	Nov. 18	80	19	8	May 21	107	8	0	Nov. 19	113	0	0
June 3	118	3	10	Dec. 2	80	9	8	June 6	105	9	8	Dec. 3	114	0	4
June 17	117	16	0	Dec. 16	72	4	8	June 18	104	3	10	Dec. 17	113	7	9
July 1	118	0	1	Dec. 30	75	2	7	July 2	102	11	5	Dec. 31	114	0	8

IMPORTS OF TIN ORE INTO THE UNITED KINGDOM.

Country.	1907.		1906.	
	Tons.	£	Tons.	£
Russia, Northern Ports	71	2,911	22	525
Russia, Southern Ports	....	....	14	270
Sweden	....	....	3	366
Germany	1,180	54,276	593	16,478
Netherlands	397	16,081	251	8,650
Belgium	67	1,785	108	2,780
France	521	29,030	622	40,717
Portugal	5	485	17	1,561
Spain	256	1,878	195	12,611
Italy	240	2,374	35	660
Bolivia—Peru	415	28,325	267	18,319
Chile	15,786	1,350,771	17,285	1,307,155
Argentine	254	14,619	74	4,684
U. S. Atlantic Ports	56	3,123	9	651
Canada (Atlantic Ports)	17	1,270	6	346
Cape of Good Hope	119	14,525	133	14,111
Natal	254	15,672	11	705
Nigerian Protectorates	75	7,208	..	..
Portugese E. Africa	1,214	64,217	317	27,109
Bengal	..	..	2	100
Burmah	..	..	1	20
Straits Settlements	13	1,643	86	7,785
West Australia	..	..	2	82
New South Wales	46	4,290	599	59,049
Queensland	11	1,215	..	..
Victoria	28	2,420	20	1,218
South Australia	1	95	..	..
New Zealand	..	105	..	..
Total	20,871	1,635,581	20,672	1,525,926

The Tin Markets in 1907.

New York.—The market for tin during 1907 was characterized by an almost total absence of large available supplies and a hesitancy on the part of the rank and file of consumers to anticipate their requirements. The larger part of the transactions only covered the immediate requirements of buyers; and while in almost every case premiums had to be paid for spot delivery, this did not deter consumers from pursuing their conservative policy in obtaining their supplies.

In sympathy with all other metals and general industrial and financial conditions which developed during 1907, tin prices found a much lower level. Until the end of April, prices remained steady, fluctuating not more than 1 or 2c., the lowest being about 40c., and the highest about 42c. per ℔. At the beginning of May, a large advance in the London market raised prices here to 45c. This advance was greatly helped by the strike among the longshoremen, which not only made the unloading of tin almost an impossibility, but even compelled some of the steamers to take back to Europe the cargo that was destined for our market. With the improvement in the strike situation, and larger arrivals of tin from Europe, the heavy premium which was exacted for spot tin disappeared, and there was a decline of almost 4c. per ℔. toward the middle of June.

When the statistics for June were published, it developed that the supplies had decreased to the extent of 1400 tons, in consequence of which a violent speculative movement took place in the London market, which advanced quotations for spot tin about £8 per ton, this market being quoted in New York at about 43c. per ℔. This was followed quickly by a total collapse in the speculative situation, and at the end of July prices had declined almost £20 per ton from the high level. August witnessed a further serious decline, both at London and in this market, prices touching

here at one time 36½c., but being at the end of the month somewhat steadier at 37¾c. per lb. During September, the market remained around 37c., but it experienced a further sharp decline during October, at the close of which spot tin was selling at about 31c. While at the beginning of November prices hardened somewhat, touching 32c. per lb. this firmer tone could not be maintained, and in sympathy with the continuing sagging market in London, declined to 30c. at the end of that month. The middle of December brought the lowest prices of the year, 26c. being quoted in this market for spot material. The better feeling and greater confidence which developed in all metal markets, did not fail to make itself felt in the tin market, which improved considerably in tone and closed at 27c. per lb.

AVERAGE MONTHLY PRICES OF TIN PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1896	13.02	13.44	13.30	13.34	13.54	13.59	13.63	13.49	13.15	12.94	13.09	12.96	13.29
1897	13.44	13.59	13.43	13.34	13.44	13.77	13.89	13.80	13.98	13.88	13.79	13.71	13.67
1898	13.87	14.08	14.38	14.60	14.52	15.22	15.60	16.23	16.03	17.42	18.20	18.30	15.70
1899	22.48	24.20	23.82	24.98	25.76	25.85	29.63	31.53	32.74	31.99	28.51	25.88	25.12
1900	27.07	30.58	32.90	30.90	29.37	30.50	33.10	31.28	29.42	28.54	28.25	26.94	29.90
1901	26.51	26.68	26.03	25.93	27.12	28.60	27.85	26.78	25.31	26.62	26.67	24.36	26.74
1902	23.54	24.07	26.32	27.77	29.85	29.36	28.38	28.23	26.60	26.07	25.68	25.68	26.79
1903	28.23	29.43	30.15	29.81	29.51	28.34	27.68	28.29	26.77	25.92	25.42	27.41	28.09
1904	28.85	28.09	28.32	28.13	27.72	26.32	26.57	27 01	27.78	28.60	29.18	29.292	27.99
1905	29.325	29.262	29.523	30.525	30.049	30.329	31.760	32.866	32.095	32.481	33.443	35.835	31.358
1906	36.390	36.403	36.662	38.900	43.313	39.260	37.275	40.606	40.516	42.852	42.906	42.750	39.819
1907	41.548	42.102	41.313	40.938	43.149	42.120	41.091	37.667	36.689	32.620	30.833	27.925	38.166

London.—January opened with £193 1s. 2d. for cash warrants and £194 3s. 4d. for three months'. The month closed at £190 3s. 4d. and £190 1s. 2d. respectively. Toward the end of February the demand for prompt delivery increased the backwardation to £2 per ton, and the month closed at £191 3s. 8d. for cash warrants and £190 for three months'. March tin suffered in the severe depression caused by the panic on the Stock Exchange, cash warrants going as low as £180 and three months' to £178, but the month closed with a strong undertone, £184 3s. 4d. being quoted for prompt and early dates and £182 1s. 2d. for three months.

April opened with an advance of 15s., but prices were suddenly depressed in sympathy with the decline in copper. Thereafter there was a steady improvement up to the end of the month, which closed at £195 for cash warrants and £192 for three months'. May opened with a brisk demand, inspired chiefly by statistics showing a reduction of 2810 tons in the visible supply. Urgent orders increased prices, but upon free realizations by sellers, the market drifted downward again, forward metal being freely offered, although immediately available supplies were held firmly. The backwardation gradually widened and stood at one time as much as £5 1s. 2d. The month closed with cash warrants at £190 and three months' at £186 1s. 4d. June opened with a disappointing shrinkage in the American demand and a consequent fall of about £4 in values, but on June

18 a sensational squeeze of the bears lifted prices, and thereafter the bears had an uncomfortable time, and the month closed at £192 1s. 4d. for cash and £182 1s. 2d. for three months'.

July found London stocks under strong control, and bears covering at advancing prices, the backwardation standing at one time as high as £18. The climax was reached on July 2, when £200 was paid for cash warrants, but the arrival of supplies from the Straits relaxed the tension and values rapidly declined. After a series of exciting transactions, the month closed at £182 1s. 4d. for cash warrants and £181 for three months'. August found consumers cautious and the market depressed by heavy offerings from the East, while financial troubles checked improvement. On Aug. 15, £162 was accepted, marking a decline of £10 1s. 2d. in four days. The market remained sensitive, closing at £166 3s. 4d. both for cash and three months'. September opened with a flat market, but an early influx of orders quickly raised prices. However, the advance could not be maintained in the face of heavy offerings induced by the persistent fall in copper. The month closed with cash warrants at £161 and three months' at £157 3s. 4d.

October was a troubled month because of the financial difficulties in America. The lowest point was reached on Oct. 16, when £134 was accepted for three months' warrants. Speculation caused a sharp recovery, but prices relapsed again and the month closed at £146 for spot and £147 1s. 2d. for three months'. November opened under the shadow of heavy realizations, while demand for consumption, particularly from America, was restricted. The month closed at £134 1s. 2d. for spot and £136 for futures. December found the market ready for a severe fall, partly induced by the financial depression and partly by the release of large quantities of Banka tin pressing for sale. Occasional covering of bear sales caused a rally now and then, but the general trend was downward to Dec. 17 when cash warrants touched £115. This low figure was precipitated by rumors of the failure of large Eastern holders, but it attracted some American orders which turned the tide. The year closed with £123 1s. 2d. for spot and £124 1s. 2d. for futures.

AVERAGE MONTHLY PRICE OF TIN IN LONDON. (*a*)
(In pounds sterling per ton of 2240 lb.)

Year	Jan.	Feb.	Mar.	Apr.	May	June
1897..........	60 .5 .1	61 .4 .3	59.18 .9	59.18 .1	60.17.10	61.16 .6
1898..........	63 .1 .7	63.15.11	65 .1 .0	65 .3 .0	66 .6 .0	68.15 .0
1899..........	99.16 .4	108.16 .3	107.16 .8	114 .1 .1	117 .9 .6	117.12 .0
1900..........	118 .9.11	137.18 .4	142 .0 .0	137.15 .0	135 .1 .8	139. 9 .3
1901..........	120 .9.10	122 .6.11	116.15 .6	116 .3 .0	123.13 .0	129.16.11
1902..........	105 .6 .5	114 .4 .9	115.10 .6	125.14 .2	134.13.10	129.12.10
1903..........	127.12 .6	133 .8 .1	137 .0 .6	136.19 .2	133.12 .0	127.11 .0
1904..........	130.10 .4	125.13 .6	126 .9 .8	127 .5 .1	125 .7 .2	119.11 .1
1905..........	131 .5.11	131 .3 .6	134.17 .2	140.11 .8	136.11 .8	138 .3 .6
1906..........	164.11.10	166 .0.10	166. 1 .2	176.14 .5	192 .6 .4	178 .0 .7
1907..........	190 .4 .0	191.18 .9	188.17 .6	187 .1 .2	191 .1.10	187.10.11

Year.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1897.........	62 .5 .7	61.10 .1	61.12 .8	62.11 .9	62.11 .9	62.10 .0	61 .8 .0
1898.........	71 .4 .2	73.10 .1	73.15 .7	78.17.10	82 .8 .6	82.10 .7	71 .4 .1
1899.........	132.13 .1	142 .1 .4	146 .7 .2	144.10 .2	129.16 .0	113. 0 .7	122 .8 .7
1900.........	142.16.10	140.19 .1	132.13 .9	130.14 .3	127 .3 .8	119.14 .9	133.11 .6
1901.........	127.19 .9	116 .1 .7	114.10 .6	113 .1 .5	114 .0 .7	108.17.10	118.12 .8
1902.........	127 .3 .2	126.10 .0	121.10 .7	117.11 .3	115 .2 .3	115.13 .5	120.14 .5
1903.........	125 .1 .7	127.16.10	120 .9 .6	115.17 .1	116.13 .9	125.15 .0	127 .6 .5
1904.........	119.18 .6	122 .5 .9	126 .7 .7	130.11 .6	133 .0 .5	133.15 .6	126.14 .8
1905.........	144 .6 .8	150 .5 .6	146.11 .9	148 .3 .6	152 .5 .3	162.14 .3	143 .1 .8
1906.........	170.12 .5	180.19.11	184.15 .3	195.15.11	195.15.10	195.19 .9	180.12.11
1907.........	188 .0 .2	170 .5 .9	166 .6 .6	146 .7 .7	138 .8 .8	125.10 .4	172.12 .9

(*a*) As reported by Metallgesellschaft, Frankfurt am Main.

A Review of Tin Mining Progress in Bolivia.

By Georg Hohagen.

In Bolivia tin has been extracted from mines and also from streams. Many of the mines from which tin is now taken were formerly worked for silver by the Spaniards, as is the case with the famous Cerro de Potosi mines.

The principal ore of tin in Bolivia is cassiterite, with more or less iron, antimony, sometimes bismuth and also lead. Tin is found as beautiful crystals of cassiterite in the Quimsa Cruz mines, near Oruro. Tin in the alluvial deposits is found frequently in big *rodados*.

Geological Features.—The tin deposits are distributed in Bolivia on the eastern side of the Andean plateau, embracing a zone of about 1500 km. length and 350 km. breadth, from La Paz to Tupiza. The veins are generally found in porphyritic diorite or granite, the latter forming the base of the Potosi mountain, which from 1566 to 1615 yielded the Spanish crown, in taxes only (20 per cent. tax) 3,240,000,000 bolivianos as the revenue from silver mines. Shales generally form a capping to these rocks, as happens at Potosi. The highest mountains in Bolivia are as follows: Sorata, 25,143 ft.; Yllimani, 24,650; Lipez, 19,621; Chorolque, 18,378; Chocaya, 17,056; Tasna, 16,744; San Vicente, 15,037; Potosi, 15,842; Colquechaca, 15,638; these being well known mineral producing mountains. The principal tin mine in Chorolque is at an altitude of 17,400 ft.

The most active mines are situated in La Paz, Potosi, Oruro and Uyuni. On the western slope of the Andes no tin has been found, and in the neighboring countries, Peru and Chile, up to date *no tin has been found.* This remark is emphasized, since some geologists say that those countries are tin producing, which is not the case. In Oruro and Potosi tin is generally associated with silver minerals and iron pyrites, but in many cases tin alone forms the principal portion of the vein. The depth of the tin veins in Bolivia can not yet be said to be great; however, the famous Cerro de Potosi, worked now only for tin, has many veins, which contain

tin, that are profitably worked at a depth of more than 600 m. and probably continue 300 m. more. The width of the veins varies generally from 1 in. to 8 ft., but in Potosi there is a vein 30 ft. wide with 5 per cent. tin. When the veins are very narrow, as a rule they are rich, assaying from 40 to 60 per cent. In this case the mineral is sacked and shipped as *barrilla* to Europe. When the vein is wide the cassiterite presents the form of grains, embedded in argillaceous matter or an irony matrix. The quality of the tin from the different mines is rather variable. Avicaya, Huanuni, Llallagua and Chorolque produce very clean concentrate or *barrilla*. In Potosi, the tin veins are generally found with iron pyrites in the lower levels and very good ore is mined from them. The gangue of all the minerals is always a silicious one.

Mining Methods.—With some exceptions, the methods of mining in Bolivia are very far below the standard of methods followed elsewhere. Generally the ground stands well and little timbering is required, which is fortunate in a country where timber is scarce. The natives and also many foreigners give the management of the mines to lawyers or doctors, sometimes to civil or mechanical engineers and exceptionally they ask a mining engineer to manage their property. It follows from this that no good method in mining is introduced and that many extravagant expenses are incurred, which were met by the high price tin a short time ago.

Water is scarce and when coming from the mines is usually acid. In the rainy season the rainfall is more or less torrential and limited to a few months, from December to April. However, there are some good waterfalls which could be utilized for power, making big savings in fuel; for instance in Potosi charcoal now costs 80 bolivianos per ton. No attention whatever is given to the preparation of maps of the underground workings, or geological sections and plans upon which assays should be plotted, Even companies with a large working capital do not know what an underground map is.

Mining Titles.—According to the law of Bolivia the right of mining belongs to the government. Mining claims can be located by any individual or foreign, paying four bolivianos[1] yearly per *pertenencia*. The *pertenencia* is a square 100x100 m. and is of indefinite depth. Mining machinery is free from any fiscal tax.

Most of the mines in Bolivia have litigations with neighbors, litigations that never end and cause much trouble. Most of these troubles arise because when locating the claims any sort of men are employed for surveying the property, men who seldom know how to use a compass, and consequently if the developments of a mine prove good, some one will try to get also a piece of good land adjacent, occasioning endless lawsuits. Avicaya is the only big property I know that has no lawsuit.

[1] 12.50 bolivianos=£1.

Milling and Smelting Methods.—The treatment of all the tin ores in Bolivia is generally dressing, using for this purpose crushers, stamps and ball and Huntington mills. Spitzlütten and spitzkästen are also used before sending the mineral to the Wilfleys, which are the tables most employed. Round buddles and revolving tables are also used for the slimes. The tailings from the dressing are generally high (7 per cent. when the ore assays 17 per cent.) and are given to contractors to rework them as they can, or are accumulated to be treated in the future.

As an example of prices and production of a small mill in Potosi, working with a ball mill, two jigs and three Wilfleys, I will say that it concentrates 1500 quintals of 18 per cent. tin ore in 30 days, producing 300 quintals of *barrilla* of 60 per cent., costing on an average 13 bolivianos per quintal of *barrilla*, the tailing assaying 7 per cent. and giving a net profit in Potosl of 2000 bolivianos, tin being valued at £130 per ton.

The dressing is a poor one, giving very high tailings (7 per cent.) but the profit is nevertheless enormous. At Potosi, on account of the high freights, ore of 30 to 40 per cent. grade is smelted in small water-jacket furnaces, 1 m. in diameter and 6 m. in hight, with charcoal as fuel and a little lime as flux, producing bars of 92 per cent. tin, the remainder being antimony and lead.

As an example I will give the daily work of such a small water jacket furnace, assuming that the mineral assays 33 per cent. tin, which is a very frequent case. A daily run will be as follows: Ore smelted in 24 hours, 120 quintals; production in tin bars of 92 per cent., 35 quintals; slag assay, 8 per cent. tin; cost of producing one quintal of tin, B. 44.

The daily cost sheet of one of these furnaces at Potosi is as follows: Labor, B. 80; supplies, B. 30; charcoal (100 quintals @ B. 4 per quintal), B. 400; lime (7 quintals @ B. 4), B. 28; coke ($\frac{1}{2}$ quintal @ B. 9), B. 4.50; *yareta* (28 quintals @ B. 0.70), for boilers and motive power for blower, elevator, etc., B. 19.60; wood (5 quintals @ B. 1.20) B. 6; llama dung (100 quintals @ B. 0.60), B. 60; general expenses, B. 93.90; total B. 722. This corresponds to a cost of smelting per 2000 lb. of ore of B. 120. This cost is attained at Potosi but elsewhere it is generally higher, reaching frequently B. 150 per ton (2000 lb) of ore. It must be observed however, that all of the smelting works here are in a dilapidated condition, very awkwardly arranged for the transportation of materials, and the furnaces are run on no rule at all (no assays of the slags being made) by the *practicos*, or men in charge of the furnaces, who are generally very ignorant persons. A small water jacket, like the one above mentioned, very much used in Bolivia, especially at Potosi, costs about B. 30,000 ready to work. And with all these drawbacks of high prices, poor work and scarce labor, still the tin business is a very good one.

Modern Improvements.—The boilers used for motive power are of nearly

all types. The fuel employed for boilers and also for reverberatory furnaces is llama dung, *yareta* and other native fuels, of poor quality and scarce, imported coal being too expensive to be used. Some mining companies have lately been using producer gas engines (Deutz) from Germany with very good results as regards economy, and being easily managed by the men in charge of them, they are becoming popular. When ordering one of these producer gas engines, the altitude has to be taken in consideration (14,000 ft. and often more) because they give in Bolivia only about 60 per cent. of their normal power capacity.

Aerial ropeways are very much used in Bolivia for delivering ore from the mines to the mills, most of the cables being of the English system (only one cable). All the installations in Bolivia have been made by the Ropeways Syndicate of London. The cost of some of these ropeways was as follows: Avicaya (Dante Abelli) 2860 m.; total cost, B. 46,000; cost per meter B. 16; Uncia (J. Minchin) 3150 m.; total cost, B. 60,000, or B. 19.05 per meter; Morococala (Penny & Duncan) 7150 m.; total cost, B. 143,000, or B. 20 per meter; Soux, Potosi, 4000 m.; total cost, B. 150,000, or B. 37.50 per meter.

Magnetic separators are used in Chaca, near Potosi, with very good results. The iron separated still contains from 3 to 5 per cent tin, which is accumulated in heaps; the *barrilla* resulting from the separators is very clean.

Electric drills have been used for drifting in Quimsa Cruz, Uncia, and some other places, but results have been far from satisfactory. Some companies have ordered lately electric air drills, which it is to be hoped will have more success. Some electric hoists are used in the mines with splendid results. In Caracoles, a mine in Potosi of Mr. Soux, a small electric hoist, worked with kerosene, raises ore from the lower levels, giving no trouble at all and has been working perfectly for three years.

Taxes.—The export taxes on *barrilla* and for tin are as follows, the rate being based on the price of tin in the Straits Settlements.

BOLIVIAN EXPORT TAX ON TIN.

Straits. Quotation.	Barilla. Per Quintal	Bar Tin. Per Quintal.	Straits. Quotation.	Barilla. Per Quintal	Bar Tin. Per Quintal
£100–110	B. 1.00	B. 1.60	£150–160	B. 1.75	B. 2.50
110–120	1.15	1.75	160–170	2.00	2.80
120–130	1.30	1.90	170–180	2.25	3.10
130–140	1.45	2.10	180–190	2.60	3.40
140–150	1.60	2.30	190–200	3.00	3.80

When the Straits quotation is less than £100 per ton, the tax on *barrilla* is B. 0.90 per quintal, and on bar tin B. 1.50. When the quotation is £200 or over, the rates are B. 3.50 and B. 4.20 respectively.

Transportation.—Transportation, as a rule, is very difficult in Bolivia, especially as regards transportation from the mines. The only railroads in the country are from Antofagasta, a Chilean seaport, to Oruro, 924 km. distance, covered in three days, and from Mollendo, a Peruvian seaport, to Puno, 523 km., whence a steamboat goes three times a week across Lake Titicaca to Huaqui on the Bolivian side, 237 km. distant. From Huaqui to La Paz the trip is made by railroad in one day, 96 km. A railroad will connect La Paz and Oruro, the second city in Bolivia and the most important mining town in the Republic. But as a rule, mule trails have to be used for reaching the mines, all freight, supplies, laborers and in fact everything being transported generally on mule back.

Supplies and machinery must be divided in packages of from 200 to 300 lb. Llamas, which are much used as pack animals in Bolivia can be loaded only with 50 to 100 lb. Packages of more than 300 lb. are unhandy, and are charged extra freight rates, which are greatly in excess of regular rates. Freight animals, as a rule, are scarce, mules costing about B. 100. The rate from Potosi to Antofagasta is B. 8 per quintal.

Labor Conditions.—Labor in Bolivia is scarce. The building of railroads and the competition between the different mining companies are raising wages. In Potosi the mines are worked by the laborers as partners of the owners of the mine. These men are known as *cajchas*, or contractors. They work as they like in the extraction of ore from the mine, and are paid 50 per cent. of the price of the metal they extract. This stupid system can be destroyed only by a strong company in control of the best properties in the Cerro de Potosi, which might dictate terms. And if to this it be added that they are paid by 36 hours of work, when probably they work only six hours, it can be understood why the ore extraction in Potosi is so dear, costing generally from B. 10.15 per quintal of ore, while in other mining districts in Bolivia, where there are no *cajchas* the extraction of one quintal from the mine costs only B. 1 to 2. The system of working with *cajchas* is maintained only in Potosi, the system of working 36 hours in the mine, being also confined to that place, and probably soon, when a big company may control this rich mountain, no more *cajchas* and no more 36 hours working in the mines will be allowed.

The Future of Tin Mining in Bolivia.—During the 12 months ending with December, 1907, Bolivia produced 27,677,781 kg. of *barrilla*, with an average of 60 per cent. tin, or 16,606,668 kg. of pure tin, the producing *departamentos* (States) of the Republic being as follows: Potosi, 16,314,664.52 kg. of *barrilla;* Oruro, 9,476,287.96; La Paz, 1,810,736.64; Cochabamba, 76,091.82; total, 27,677,780.94 kg. The *departamentos* of Potosi and La Paz increased their production as compared with 1906 as follows: Potosi, 4.08 per cent.; La Paz, 3.50 per cent. The *departamento* of Cochabamba is a new tin producer. It began producing with 800

quintals in 1906, and in 1907 had twice the production of 1906. The Bolivian government had a revenue of B. 1,403,571.23 during 1907 from the export tax on tin concentrates and bars.

To what extent mining can be carried on in depth in Bolivia is yet a problem. There are no factors operating against deep mining if the geological character of the veins is not adversely changed. The existence of payable ore in Potosi, according to what I have seen in the mines, may be calculated at a depth of 800 m. vertically. The future looks bright for tin in Bolivia. Railroads are being constructed, the government is stable and improvements in the management of the mines and consequently in economic conditions are being effected. Water is scarce in the plateau and fuel is dear, but the climate is cold and healthful, and though the mines are at high altitudes and generally with poor trails, with good management they give handsome dividends.

Tin Mining in Tasmania.

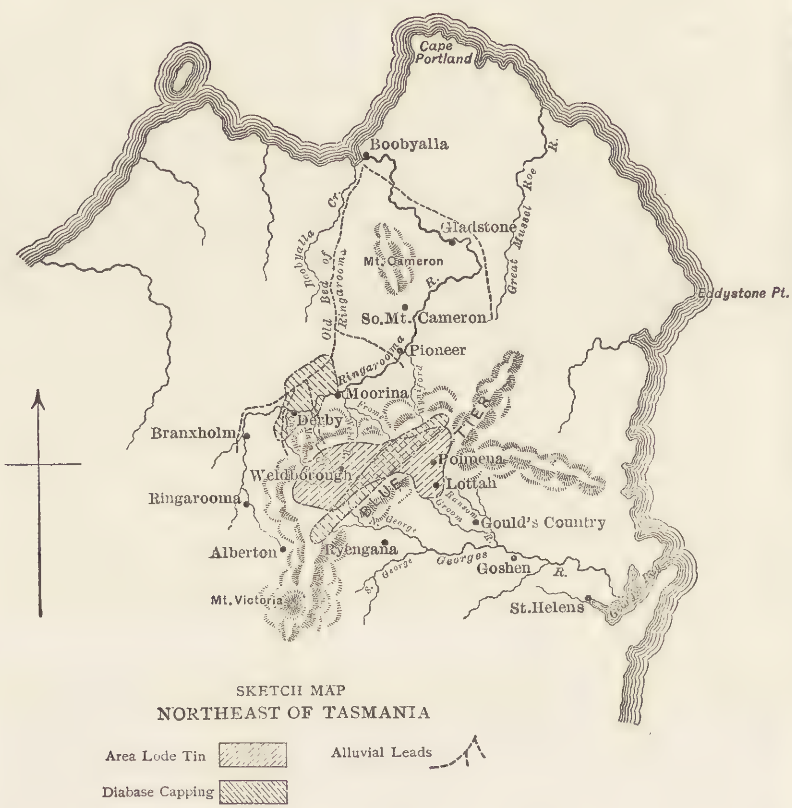
By James B. Lewis.

Tin mining in Tasmania has become an important industry. Its center is in the northeastern district of the State, which is shown in the accompanying sketch map. This map indicates in outline the general formation of the district; it will be seen that all the alluvial leads are on one fall of the mountains; while practically all the large lode deposits worked by the Anchor and other companies and prospected by the Mt. Lyell company are on the other slope. This suggests on the one hand that the lode deposits on the one fall have been denuded to form the deep leads, and that on the other slope denudation has only just exposed them. A circumstance that seems to support this is that the drifts in the Briseis and New Brothers' Home mines, and in some other alluvial mines, average the same as the lode material worked at the Anchor mine. As the latter has now crushed about 750,000 tons some comparison is possible. There are much poorer drifts than the above worked, but they appear to be, partly at least, the result of the denudation of the overlying granite, which usually contains occasional seams carrying coarse tin.

In the following notes the present condition of the different districts is considered in detail.

Mount Bischoff.—The Mount Bischoff mine, comparatively an old mine, has still some surprises. The deposit so long worked occurred in slate country; adjoining it is quartzite country, and in this are fissure veins, which were worked intermittently 20 years ago by other companies. Litigation, the apparently limited extent of the deposits and the low price of tin caused their collapse, and the leases fell into the hands of

the senior company which pigeonholed them as matters that might be considered at some future date. That time has arrived with the practical exhaustion of the richer deposits so long worked, which may indeed have been the breaking down of the lodes—Wheal Bischoff and North Valley—now being explored. An adjoining mine, the Bischoff



SKETCH MAP
NORTHEAST OF TASMANIA

Extended, has been working for some time with more or less success on one of these; mostly less, on account of the difficulties of position, power and capital. By an adit level it has recently struck what is supposed to be Wheal Bischoff vein at a vertical depth of 1000 ft. below the present highest outcrop. These veins are not of great width and

vary considerably in extent and value, but contain rich patches, and worked in with the large low-grade porphyry deposits still existing, may extend the profitable life of the company many years. These porphyry deposits contain tin in the shallower levels, but at no great depth from the surface nodules appear with cores of iron pyrites, which gradually increases until there is much pyrite and no tin.

A few miles out of Waratah are the Cleveland lode deposits that are now attracting some attention, and great things are prophesied for them. However, men who held the same blocks some years since, and abandoned them, report that the tin is of no great extent either in area or depth.

Heemskirk.—A little interest is being taken in Heemskirk tin properties, further south and nearer Zeehan. A number of companies erected plants and essayed to work these over 20 years ago, but all their efforts ended in disaster. Whether these lodes are continuous in depth and value is a doubtful question and one that requires to be settled before capital is invested to any extent in them. They may resemble the east coast deposits presently to be referred to.

Brookstead.—To the east of the main railway line from Launceston to Hobart and toward the flank of Ben Lomond is the Brookstead mine, lode and alluvial, on freehold land, lately acquired by Adelaide capitalists. Others who had options over this had abandoned them, but the lodes are reported to be not only extensive but continuous and fairly rich, and at one time they were worked with some promise of success.

Avoca.—The mines near Avoca, and on the slopes of Ben Lomond, appear to have been created for the benefit of the mine promoter, for they consist mainly of good surface indications, and preliminary work generally suggests richness and permanence. If this be followed by the erection of machinery the bottom immediately falls out and developments seldom get so far as a first crushing.

Blue Tier Deposits.—These are on the northeast coast and appear to be unique. A casual acquaintance gives an impression of great extent, permanency and payable value, which is not borne out by experience. The country is very rough and heavily timbered and consequently difficult to prospect. In the earlier days shallow and rich alluvial deposits produced many hundred tons of tin, and for nearly 30 years attempts have been made to work the lode formation with only trifling success on the part of one company, the Anchor. The Mount Lyell Mining and Railway Company lately took up an area of nearly 2000 acres and spent upward of £10,000 prospecting by diamond drill, trenches and hand bores, and came to the conclusion that the deposits were unworkable. The ground is patchy, the patches being comparatively rich, but shallow, and the ground between them is practically barren on the Mt. Lyell company's

blocks, but averaging nearly 3 ℔. per ton on those of the Anchor company. These lode deposits crop up through the country granite, and although they vary in composition from mica with a little quartz to feldspar with a little mica, they are always easily recognized. They are generally separated from the country granite, which frequently consists of quartz and feldspar alone, by pegmatite seams, and they appear to run under the country granite and give the impression that denudation has exposed them only at isolated spots. The facts make other conclusions untenable, for if there had been the denudation of the lode formations from the higher outcrop to the lower, there would be between the Blue Tier and the sea on the east deep alluvial deposits containing many thousands of tons of tin. There are these deep deposits, but the only tin-bearing ones are shallow and limited in extent and the tin appears to be of purely local origin. On the western slopes of the Tier these deposits appear to have been practically denuded and redeposited in the large deep leads worked by the Briseis, Pioneer and other companies. There are, it is true, small fissure veins, but speaking generally, these, though fairly rich, are insignificant in extent, and usually do not last to any depth. At any rate, none has been proved to any great depth, nor to any great extent horizontally.

Briseis Mine.—With regard to the alluvial deposits, that of most importance at present is the Brothers' Home, or Cascade lead, on which the Briseis and New Brothers' Home mines are working, the former company working the latter mine along with its own under a tribute agreement. A fair presumption from existing information is that the Briseis company will get the present value of its shares back with a fair profit besides from its mine. In addition, it has secured blocks previously worked by several other companies, large portions of which are not yet explored. It is possible that in these blocks there may be longer life for the mine. In addition, the Briseis company now owns all the water power of any value in the district, and can command any new ground that is likely to be opened, so that it is in a fairly comfortable position. It is at present the largest tin producer of the island, the following being the returns for the principal mines for the year ending with June, 1907: Briseis, 1025 tons; Bischoff, 793; New Brothers' Home, 372; Pioneer, 388; Anchor, 224.

Pioneer.—The preceding statement does not give a fair idea of the position of the Pioneer mine, for much lost time was caused by the disastrous floods of February, when 10 in. of rain fell in an hour or two. Without the floods the total return probably would have been nearly 600 tons instead of 388 as shown. This mine, being largely below the level of the river, the drifts are broken down by water under high pressure through nozzles, and subsequently pumped up by centrifugal pumps,

on barges worked by steam, into the dressing races. A scheme is in hand to develop electricity from water power and use this instead of steam.

Dredging Claims.—Down the Ringarooma there are at work now several bucket dredges, which have not yet achieved the success expected of them; expectations based on the success of similar dredges on gold drifts in other places. Tin ore, being of less specific gravity, is harder to save, and in the bed of the Ringarooma there is a good deal of timber and also heavy boulders; there are large bodies of tailings also from the other mines that would have to go through the dredges. Consequently there has been much disappointment. As each dredge, however, has demonstrated its qualities and weaknesses, its successor has benefited, and the last—the Dorset dredge—is a large and powerful machine. Besides the bucket dredges, there are a number with centrifugal pumps, copies of the Pioneer dredges, which are working with more or less success, but the output of any one is not considerable.

Gladstone District.—In this district there are large bodies of drift, but mostly of a low grade, a good deal practically barren. The tin deposits occur apparently as parts of no definite lead, but usually exist as isolated patches, the deeper ground not necessarily the richer. A company with a substantial capital, the Cybele, took up a large area of this, portions of which had previously been worked with more or less success. The promoters put down a large number of bores and on these the mine was floated. A fine up-to-date pumping plant was procured—water-tube boilers, high-speed engines and centrifugal pumps in series. While this was being erected the manager worked a portion of the ground with a centrifugal dredging plant, and re-bored a large portion of the other ground. The dredging, though payable, was disappointing. The fresh boring showed the ground to be of less value than the previous bores indicated, and the manager chose a time of crisis in the company's affairs, when the working capital was exhausted and the plant not quite complete, to announce publicly his sudden loss of faith in the mine, his belief that the earlier bores had been tampered with, and his resignation.

This introduces the question of the reliability of boring in tin drifts, for here is a mine in which the shareholders might be expected to be practically certain as to results; as it was bored three times over, but they are left in greater uncertainty than if it had never been bored at all. Some of the most experienced managers in the district put no faith whatever in bores. On the other hand the Pioneer mine has borne out the results of its boring in a satisfactory manner, and the bucket dredging companies are also quite satisfied with the accuracy of their bores. It would seem that success depends not only on the skill and honesty of the man in charge, but also on the nature of the drift bored, and the quantity of water encountered in it.

Other Districts.—An effort has been made to refer to the more important enterprises, but in addition there are districts not mentioned from which a little tin comes, and there have been several undertakings to work alluvial flats in which the wash consists largely of shingle and boulders; but in only one case and under specially favorable circumstances has the material more than cleared working expenses. It is difficult in sampling to make a proper allowance for the space taken up by the boulders. The wash does not improve in depth, but the best results are just as likely as not to be found in the upper part of the wash. It is generally held back by bars in the river bed, and is usually waterlogged, and requires a jet elevator or pump to lift the water and drift. There is no expeditious method of working, as every boulder and stone has to be handled to free it from the tin-carrying drift, so that on the whole it is a class of investment to avoid.

The northeastern district lately experienced a promoters' boom and a great many wild cats were ushered into existence. Some of them are included in the mines referred to above, but most of them had only a brief and stormy existence, and after answering the promoters' purposes, expired while still young kittens.

TUNGSTEN.

The demand for tungsten has shown a steady increase during recent years. In the United States, tungsten mining has reached the greatest importance in Colorado, which State furnishes the bulk of the output. During 1907, California and Idaho also produced small quantities, and exploration was active in Nevada, South Dakota, Arizona and Montana. Deposits of tungsten ore are also known in Washington, New Mexico and Connecticut. As may be noted in the accompanying table, the production for 1907 showed a great increase over that of 1906.

PRODUCTION OF TUNGSTEN CONCENTRATE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.	Value.	Average per Ton.	Year.	Production.	Value.	Average per Ton.
1900	46	$11,040	$240	1904	740	$184,000	$249
1901	179	27,720	155	1905	834	257,463	308
1902	184	33,112	180	1906	1,096	443,150	404
1903	292	43,639	149	1907	1,468	715,031	487

(a) Statistics reported by the U. S. Geological Survey, except for 1905, 1906 and 1907.

Markets and Prices.—A large part of the tungsten ore produced in this country is smelted by one or two domestic concerns, but a greater or less amount is exported to German smelters. The latter buy the bulk of the output of Cornwall, and other countries, and send tungsten metal to the United States. Owing to the unsettled condition in the steel industry during the latter part of 1907, American buyers could not make use of all the metal they had bought on forward contracts, with the result that they were sending it back to Great Britain and Germany at less than the prices of the German makers. For a time this brought the price of tungsten ore down to a comparatively low level. Early in 1907, tungsten concentrates containing 60 per cent. or more of tungsten trioxide brought in the neighborhood of $6.50 per unit; by the middle of the year the price had advanced to $11; in December there was a recession and sales were made at prices ranging from $5.50 to $7. By a unit is meant 20 lb., or 1/100 of a ton; for a 60 per cent. concentrate the value is therefore (taking $7.50 per unit for example) $7.50x1.00x0.60, or $450 per ton. De Golia & Atkins, of San

Francisco, Cal., give the price of wolframite in Germany in January, 1907, as $6.70@8.30; May, $11.55 (high point); December, $5@5.65.

In former years, wolframite was considered more desirable than scheelite, on account of the difficulty of treating the latter by the old crucible method; but for the modern method of reduction in the electric furnace, scheelite is said to be fully as desirable as wolframite.

Tungsten Mining in the United States.

Arizona (By Wm. P. Blake).—Wolframite and hübnerite are both found in commercial quantity in Arizona, but the latter is the more abundant of the two. The two chief localities are in the southwestern part of the Territory. That which has been the largest producer is in Cochise county, a few miles north of Dragoon station, on the Southern Pacific Railway; the other is in Santa Cruz county, about 70 miles southwest of Tucson, at Gigas, near Arivaca. In this region the mineral accompanies most of the quartz veins along the whole gold-bearing belt southward into Sonora. It penetrates the quartz gangue in small blade-like crystals, or in tabular masses, or bunches, of irregular form and can be easily separated from the gangue by the ordinary processes of concentration.

The best known claims at Gigas were purchased some years ago by the late Mr. Olcott, of New York, but have since passed into other hands and, although hundreds of tons of high-grade ore have been mined and piled up for nearly a decade, no effort to utilize it by concentration has been made. Other claims, unworked, are represented by Bent & Sampson, of Tucson.

The veins near Dragoon have been the chief producers. The ore occurs in quartz in veins traversing granitic gneissoid rocks. These veins are vertical but are irregular in width and the hübnerite is unevenly distributed. This has discouraged deep working and most of the production has been secured by stripping the outcrops to a slight depth, or by washing the gravels of the gulches traversing the outcrops. There is no recent production to record. The mineral has been reported as found in Mohave county, but no satisfactory information regarding the deposts has been obtained.

Scheelite is found in the Maudena vein, at the north end of the Santa Catalina mountains, about a mile north of the Southern Belle gold mine. It occurs both massive and granular. Some sample shipments have been made but the mine is not now worked. Tungstic ocher occurs sparingly upon the outcrops of hübnerite at Gigas but not in quantity sufficient to be commercially important. Tungstate of copper has been found, but not in quantity, in the southern part of Pima county.

California.—The bulk of the tungsten produced in this State during

1907 came from San Bernardino county. However, deposits are known to exist at other points, and development work was actively pushed in Madera and Kern counties. In the latter county several important discoveries were reported in 1907.

Colorado.—In 1907 the production of tungsten concentrates, including a few tons of high-grade ore, was 1170 short tons, valued at \$569,905; all of this came from Boulder county. During the first part of 1907 the industry was greatly stimulated by high prices. Most of the actual mining was done by leasers, this system being best adapted to the pockety nature of the enriched portions of the veins. However, several prominent steel interests have properties which they operate.

Toward the close of 1907, owing to declining prices and a lack of orders for ore, practically every mine and mill of importance was closed down. The early months of 1908, however, brought a slight change for the better and it is expected that the industry will slowly regain its former position.

Idaho (By Herbert S. Auerbach).—Tungsten is known in several properties and occurs in considerable quantities in the Golden Chest mine, which produced 15 tons of scheelite in 1907. The ore assayed from 61 to 77 per cent. tungsten trioxide. At the Golden Winnie tungsten is found, generally as a concentrating ore, with occasional pieces the size of an apple. Several tons of concentrating ore are now on the dump at this mine.

In early days a German prospector is reported to have struck a 2-ft. vein of tungsten in Pony gulch. On account of the great weight of the mineral, he had it assayed for lead, and getting no return for that metal abandoned the vein, which since then has often been sought but never found. Scheelite was frequently found in the sluice boxes while placer operations were carried on in Trail and Pony gulches. In Eagle gulch, Dunlap & Smith find scheelite in their boxes and one small vein has been uncovered near the Columbus group.

Montana.—Tungsten ores occur in limited quantities at many points in Park county, but no production was reported from this district during 1907. Pockets of scheelite have been found in the Kimberly-Montana mine at Jardine, and some of this has been marketed. Hübnerite has been found sparingly in some of the mines in the Butte district. In only one case, that of the Birdie mine, in the hills to the east of Butte, has the occurrence of this mineral shown any persistence. Small quantities of the ore have been mined from time to time and shipments made.

Nevada.—During 1907, the discovery of pockets of tungsten ore was reported from Round Mountain. The ore is said to be of good quality, and several tons were mined. Development work was prosecuted in White Pine county and a considerable tonnage of ore was piled on the dumps.

South Dakota.—Tungsten deposits are known in Custer and Pennington counties. In the latter county, the American Tungsten Mining Company

operates a mine about five miles east of Hill City. The shaft at this property has been sunk over 100 ft. and drifts have been extended. A 50-ton concentrating mill and a modern saw mill are included in the equipment.

Tungsten Mining in Foreign Countries.

Australia.—The production of wolframite in Queensland during 1907 is reported at 87 tons, valued at £10,220, as compared with 236 tons, valued at £22,368, in 1906. Tasmania produced 41 tons of wolframite, valued at £4,411. During 1907, New South Wales produced scheelite to the value of £23,781, and wolframite to the value of £26,235. Early in the year the industry in the neighborhood of Hillgrove was greatly stimulated by high prices and a strong demand, but toward the end of the year prices fell off considerably and activity slackened proportionately.

India.—Wolframite has been discovered at Agargaon.

New Zealand.—There is a scheelite mine of considerable importance at the head of Lake Wakatipu. It is reported that work was prosecuted on three levels during 1907, and that three shifts daily were worked. A concentrating mill, which is said to turn out a clean product, has been erected. Scheelite mining on Marcus Flat, Otago, has made rapid progress in the last few years. An increasing number of miners are being employed, and the industry promises well for the future.

South Africa.—Scheelite has been discovered in Rhodesia in promising quantities. Although no commercial importance is attached to the discovery so far, it is stated that prospectors have opened up a vein carrying quartz and scheelite in alternate bands over a width of 15 ft., and that the outcrop has been traced for a distance of 1200 ft. along the surface. The deposit is six miles distant from a railway.

United Kingdom.—The preliminary official statement of the mineral production for 1907 gives the output of tungsten ore as 312 long tons, as compared with 263 tons in 1906. This ore comes from what is called the high-level platforms of Bodmin Moor, in Cornwall. For many years little mining was done in this district because of the difficulty of separating the wolframite from the tin; however, this difficulty has now been overcome. The veins from which the wolframite is derived have been found close to the points where the wash is enriched by their denudation. The success of mining depends to some extent on the slope of the granite floor on which the detritus rests; where the dip is only slight, the deposit becomes water logged, and the method of separation adopted is extremely expensive to carry out.

URANIUM.

The principal known deposits of uranium ores in the United States are in Colorado, where carnotite and uraninite (pitch-blende) occur. A reduction plant near Cedar, in San Miguel county, is reported to be operating on carnotite ores carrying 2 per cent. or less of uranium and having also a vanadium content. Uraninite is mined in the Kirk, Wood and German mines in Gilpin county, Colo. Small quantities of uranium have also been found in the Black Hills in the form of uranium phosphate (autunite). Other small deposits, which are of mineralogic interest only, have been found in Connecticut, North Carolina, Texas and California. Pitch-blende, which is by far the most important ore of uranium, is found at Joachimsthal, in Germany, and in certain parts of Hungary, Turkey and Canada.

So far but few uses have been found for uranium. It is used as acetate as an indicator in various determinations in organic chemistry, and other salts are used in iridescent glass and pottery glazes. Experiments to determine the suitability of uranium as a steel hardening metal indicate that it does not impart any special properties which are not supplied by other and cheaper metals.

The value of the imports of the oxide and other salts of uranium into the United States for a period of years, the figures being for the fiscal year ending June 30, have been as follows: 1903, $13,024; 1904, $10,151; 1905, $8574; 1906, $7093; 1907, $14,656.

VANADIUM.

BY EDWARD K. JUDD.

Interest in the use of vanadium for the preparation of special steels has become more general with the finding of new and important deposits which can be depended upon to supply large and regular amounts of the ores of the metal. The recovery of vanadium from its ores, the methods of its utilization, and its effects upon iron and steel are now so well known that several of the largest steel makers have begun the manufacture of vanadium steel on a serious basis.

Ores of Vanadium.—Among the minerals which can be considered as possible ores of vanadium may be mentioned those in the accompanying table.

VANADIUM-BEARING MINERALS.

Name.	Composition.	Per Cent. V_2O_5	Principal Localities.
Carnotite	$K_2O,2U_2O_3,V_2O_5,3H_2O$	15–18	Montrose San Miquel and Rio Blanco counties, Colo.
Descloizite	$(Pb.Zn.)(PbOH),VO_4$	22.70	Associated with vanadinite
Mottramite	Vanadate of lead and copper	18.85	Cheshire, England
Patronite	Vanadium sulphide	15	Minasragra, Peru
Pucherite	Bi_2O_3,V_2O_5	28.2	Saxony
Roscoelite	$H_8K(Mg,Fe)(Al,V)_4(SiO_3)_{12}$	24	Southwestern Colorado
Vanadinite	$Pb_5Cl(VO_4)_3$	19.40	Spain, Arizona, New Mexico
Volborthite	$(Cu,Ca,Ba)_3(OH)_3VO_4+6H_2O$	19.60	Urals

In addition to the above, iron ores not infrequently carry some vanadium which is reduced like phosphoric acid in the blast furnace, and goes into the pig iron. Vanadium is also found in a coal from San Raphael, Province of Mendoza, Argentina. The ash of this coal is said to carry as much as 38.22 per cent. vanadium oxide.

At present the consumption of crude vanadiniferous ore in the United States is supplied from the roscoelite-bearing sandstone in southwestern Colorado, and the recently discovered patronite deposit in the Cerro de Pasco district of Peru. The Vanadium Alloys Company, in the former locality, has a reduction and mining plant at Newmire, Colo., while the American Vanadium Company imports crude material from Peru, and makes it into ferro-vanadium in its plant at Pittsburg. The Vanadium Alloys Company does not make ferro-vanadium, but supplies its product,

a vanadate of iron containing about 34 per cent. vanadium, to other manufacturers. Besides the American Vanadium Company, the other producers of ferro-vanadium are the Primos Chemical Company, Primos, Pa., and the Electro-Metallurgical Company of America, at Niagara Falls.

Prices.—The Vanadium Alloys Company sells its vanadate of iron for $2 per lb. of metallic vanadium contained in it. The present price for ferro-vanadium, free from carbon, is $5 per lb. of metallic vanadium contained. The price for electrolytic alloy, containing 1 to 4 per cent. carbon, is much less.

According to a recent decision of the board of appraisers, vanadium ore is admitted free of duty.

Vanadium in the United States.

Arizona (By Wm. P. Blake).—Vanadate of lead occurs at many places in association with the ores of lead, and also with the compounds of molybdenum. It is much more widely distributed in Arizona than in California or Nevada, and some of the localities have yielded choice crystals for the mineralogical collections of the world. It has not yet been worked commercially and there is no production to report.

The chief localities in which the mineral has been found are: The Mammoth gold mine at Shultz, Pinal county; the old Yuma mine in Pima county, about 20 miles northwest of Tucson; near the Silver Belt and Silver King mines in Yavapai county; and at the Castle Dome mines in Yuma county. In the last named locality the vanadinite occurs at the water-level of the Railroad claim, in close association with yellow wulfenite and crystalline lead carbonate. It is also found in brilliant red crystals at the Hamburg claim in Silver district, about 20 miles north of Castle Dome landing.

Colorado.—The principal operations in this State are those of the Vanadium Alloys Company. Their reduction plant is at Newmire, San Miguel county, on the Rio Grande Southern, 14 miles west of Telluride. The mines are on Big Bear creek, two miles south of the mill, and comprise quarries and short tunnels into a 9-ft. bed of sandstone through which is disseminated fine grains of roscoelite, giving the stone a greenish color. The rock averages 4 per cent. vanadium oxide. Carnotite is almost absent at this locality. At Placerville, seven miles northwest of Newmire, is a similar deposit, containing a little more carnotite, though roscoelite is the principal mineral. Elsewhere in the same part of Colorado, deposits are found in which carnotite is the prevailing mineral. Such a deposit has been worked at Cedar, San Miguel county, on the Dolores river, by the Dolores Refining Company; and others have been discovered on La Sal creek, Montrose county, and on Coal creek, Rio

Blanco county, 14 miles northeast of Meeker. The same rocks that carry the vanadium and uranium ores in Colorado extend over into Utah.

New Mexico.—Near the town of Magdalena, Dr. J. H. Young recently found a deposit which proved to contain vanadium. Four shafts have been sunk in the orebody, which besides vanadium contains galena, silver and copper. At present two companies are working groups of claims, and about 150 locations have been made. A test on ore from the Silver Line claim showed 7.8 per cent. vanadium oxide. Edgar D. Stone records the finding of similar ore on the dump of the Torrence mine, at Socorro.

Vanadium in Foreign Countries.

Argentina.—It is rumored that a European concern is to develop the unique vanadium deposits of Argentina. Certain coals in the province of Mendoza and in the Neuquen territory contain vanadium. Analysis of the ashes of a coal from San Raphael showed the following composition: Vanadic oxide, 38.22; phosphoric acid, 0.71; sulphur trioxide, 12.06; lime, magnesia and potash, 12.30; oxides of iron and aluminum, 18.76; silica, 13.70 per cent. The coal is said to be low in ash.

Peru.—The discovery of large deposits of vanadium sulphide, patronite, has been described by several writers.[1] The following is by J. Kent Smith.[2] The original ore has the following composition: Vanadium sulphide, 39.84; molybdenum sulphide, 1.57; nickel sulphide, 1.49; iron sulphide, 4.07; free sulphur, 30.57; silica, 13.60; alumina, 2.46; combined water and carbon dioxide, 5.00; alkalies, 1.40 per cent. The ore burns freely, losing 45 per cent. of its weight, after which it contains: Vanadic oxide, 58.08; iron oxide, 4.98; molybdenum oxide, 2.62; nickel oxide, 2.24; silica, 25.00; alumina, 4.52; alkalies, 2.56; sulphur, 0.23 per cent. Mining conditions are easy, and direct railroad communication exists nearly all the way to Callao. The claims have an area of 3.5 by 1.5 miles, and a large proportion of the area contains vanadium. Five distinct veins have been proved. The one now being worked has a width of 16 ft. and has been exposed along its outcrop for a distance of 200 ft. It dips at an angle of 65 deg., and has been opened at a depth of 140 ft. by an adit lower down on the hill.

Technology of Vanadium.

Extraction Processes.—The method used by the Vanadium Alloys Company, at Newmire, is as follows: The ore is hauled up an incline to

---

[1] Foster Hewett, *Eng. and Min. Journ.*, Sept. 1, 1906.
[2] *Bull.* Am. Inst. Min. Eng., Sept., 1907.

the bins by a self-dumping car. It passes through a Blake crusher which reduces it to 1-in. size, then through coarse rolls, to 4-mesh, and finally through fine rolls to 20-mesh size. Circular, impact, revolving screens, with 4- and 20-mesh screens are used, the coarse going back to the rolls, while the fine passes to the head end of a reverberatory furnace. Salt is added to the ore during the crushing process. The mixture is rabbled and worked down the hearth by hand, and when roasted is dumped to a cooling floor. The roasting forms sodium vanadate and sets free chlorine. The cooled ore is put into tanks with water and is agitated with compressed air. This dissolves the sodium vanadate and any vanadium chlorides that may have been formed. The leached residue contains little or no vanadium. The solution is led to other tanks and ferrous sulphate is added, which precipitates the vanadium as ferric vanadate. The precipitate is collected in filter-presses, and dried slowly to a lumpy, olive-green powder, which is then sold to manufacturers of ferro-vanadium. The capacity of the plant is 12 tons per day.

The Haynes process for the treatment of carnotite ore, in which uranium is contained, consists in agitating the crushed ore in a revolving barrel for 1 hr. with a boiling solution of sodium carbonate. Both vanadium and uranium are extracted by this treatment. To the filtered solution is then added sodium hydroxide, which precipitates the uranium as sodium uranate; this is filtered out as the uranium product. To the filtrate is added slacked lime, which precipitates calcium vanadate, but contaminated with carbonate of lime. This is the vanadium product. The final filtrate, containing mainly sodium hydroxide, is then recarbonated by contact with furnace gases.

According to a recently devised process (U. S. patent No. 880,645, Mar. 3, 1908) particularly applicable to carnotite ore, the pulverized ore is agitated with hot sulphuric acid, of 15 to 20 per cent. concentration, about 400 lb. of 65-deg. (B.) acid, diluted to the above strength, being sufficient to treat 1 ton of ore. The filtered solution is then brought in contact with fresh ore, whereby the solution will be neutralized and part of the dissolved metals will be precipitated upon the fresh ore. The process is repeated until a sufficient quantity of solution is secured, when it is treated with sulphur dioxide to reduce any iron present so as to prevent excessive precipitation of iron in the next step. To the cleared solution is then added pulverized limestone up to the point at which the vanadium and uranium just begin to precipitate, calcium sulphate being formed. The calicum sulphate is filtered off, and the metals in the filtrate are precipitated by boiling with more limestone. The resulting product is marketable, or the two metals may be separated by dissolving in sulphurous acid, boiling to precipitate basic uranium sulphite, while the vanadium remains in solution and may then be precipitated with lime.

ZINC.

By W. R. Ingalls.

The production of spelter in the United States in 1907, together with the corresponding figures for previous years, is given in Table I, which is based on reports from all of the producers. The nature of the zinc-smelting industry is such that the statistics of production can be collected and summarized with great accuracy. Table I gives the production only of virgin spelter. There is in addition a small annual production of spelter from the dross of galvanizing works and other waste products. It will be observed that Oklahoma appears for the first time as a producer of spelter. The production in that State will no doubt increase greatly, inasmuch as two works which were not completed until December, 1907, will figure prominently in the statistics of 1908.

I. PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1899.	1900.	1901.	1902.	1903.	1904.	1905.	1906.	1907.
Colorado	....	....	....	....	877	4,906	6,599	6,260	5,200
Illinois (a)	49,290	37,558	44,896	49,672	49,526	47,607	45,357	48,238	56,103
Kansas	55,872	57,276	74,270	87,321	87,406	103,721	114,948	129,741	133,561
Missouri	15,710	20,138	13,083	10,548	9,894	12,056	11,800	11,088	11,594
Oklahoma	....	....	....	....	....	....	....	....	5,094
South & East (b)	8,803	8,259	8,603	10,698	10,799	13,513	23,044	30,167	38,060
Total tons of 2000 lb	129,675	123,321	140,822	158,239	158,502	181,803	201,748	225,494	249,612
Total tons of 2240 lb	115,781	110,028	125,734	141,283	141,520	162,324	180,132	201,343	222,868
Total metric tons	117,644	111,794	127,751	143,552	143,792	164,921	183,014	204,548	226,398

(a) Up to 1903, inclusive, includes also the production of Indiana. (b) New Jersey, Pennsylvania and Virginia, and (since 1903) West Virginia.

World's Production.—It appears that the world's production of spelter in 1907 was 736,500 metric tons, the remarkable annual increase which has been going on uninterruptedly since 1899 having thus been maintained in 1907. A noteworthy feature in the statistics is the fact that the increased production in 1907 came chiefly from the United States, the European countries showing only trifling gains. It is most noteworthy, however, that in 1907 the United States for the first time in history took the premier position among the world's producers of spelter, its production having been 226,398 metric tons, against 208,195 tons as the combined production of the two German districts, viz., Silesia and the Rhine. Giving the United States credit for the zinc content of its production of oxide made directly

from ores, it has held the first place for a long time. Now it occupies the first place on the basis of spelter production alone.

II. PRODUCTION OF ZINC IN EUROPE AND AMERICA. (*a*)
(In metric tons.)

Year	Austria	Belgium.	France.	Germany.	Holland.	Italy.	Russia.	Spain.	United Kingdom.	United States.	Totals.
1896...	6,888	113,361	45,585	153,082	4,770	*Nil.*	6,257	6 133	25,278	70,432	421,786
1897...	6,236	116,067	38,067	150,739	6,600	250	5,868	6,244	23,805	91,070	444,946
1898...	7,302	119,067	37,155	154,867	6,700	250	5,664	6,031	28,387	103,514	468,937
1899...	7,192	122,843	39,274	153,155	6,235	251	6,331	6,184	32,322	117,644	491,331
1900...	6,742	119,315	36,305	155,799	6,845	547	5,963	5,611	30,207	111,794	465,438
1901...	7,558	127,170	37,600	166,283	7,855	511	6,090	5,354	29,877	127,751	516,049
1902...	8,309	124,780	36,300	174,927	9,910	485	8,280	5,569	40,244	143,552	552,356
1903...	8,949	131,740	37,416	182,548	11,515	126	9,901	5,134	44,110	143,792	569,971
1904...	9,159	137,323	41,600	193,058	12,895	189	10,607	5,887	46,218	164,921	621,857
1905...	9,204	142 555	43,200	198,208	13,550	5	7,520	6 184	50,125	183,014	653,565
1906...	10,711	148.035	46,536	205,691	14,650	69	9,610	6,209	52,587	204,548	698,646
1907...	11,359	154,492	(*c*)49,733	208,195	14,990	(*b*)	9,738	(*c*)6,000	55,595	226,398	736,500

(*a*) From the official statistics of the various Governments, except 1906 and 1907, for which years the figures reported by Henry R. Merton & Co. have been used where the official statistics were unavailable. In addition to the production reported in this table, Australia produced 286 long tons in 1903, 299 in 1904, 544 in 1905, 1008 in 1906, and 980 in 1907. (*b*) Included in Austria. (*c*) An approximate separation of the total which is reported for "France and Spain."

General Conditions.—During the early part of 1907, the margin on ores was quite satisfactory to the smelters, so much so that the Collinsville Zinc Company contemplated putting its old plant at Collinsville, Ill., into operation again. The remarkable set-back in the zinc industry in the latter half of the year reduced the margin to a narrow figure, and the tendency was then strongly to put furnaces out of commission rather than to fire up old ones. Nevertheless, the old coal smelters at Rich Hill and Nevada, Mo., and two of the old works at Pittsburg, Kan., continued in operation on a small scale. The new life of these old works is based (1) on the increased cost of gas to many of the gas smelters, reducing materially the difference in cost between smelting with the two fuels; (2) on careful attention to the metallurgical work and the commercial side of the business; and (3) on the reduction of general expense to a very small figure by the direct attention of the proprietors.

With respect to the cost of smelting in general, it is now well known that the cost of natural gas to many of the smelters in Kansas is so much that a coal smelter in Illinois is at least on equal terms, and perhaps on superior, if sulphuric acid be recovered as a by-product. There are still, however, some smelters, especially those at Caney, Deering, and Bartlesville, who are able to obtain natural gas at low figures, and smelt a good class of ore at as low as $8 per ton, which is not very much above the cost in the halcyon days at Iola. A smeltery in the natural-gas field has the great attraction that its cost per ton of capacity is much less than that of a modern smeltery to use coal as fuel, which is an important consideration even when the gas supply is comparatively costly. However, the general development of the industry in the United States is taking place precisely

on the line that I predicted several years ago, namely, there is the installation of new plants at new points in the gas fields where the builders are willing to risk short life with the expectation of reaping quick and large profits; while on the other hand there is an increasing tendency to build more costly and more substantial plants in the coalfields of Illinois, where a permanent business may be expected.

As to the situation at Iola, it may be summed up in the statement that the smelters still have an adequate supply of gas. It has been predicted during each of the last four years that the next year would be the end, but the smelters still have sufficient gas to run the 42 blocks of furnaces of the district. Nevertheless, the opinions that were expressed four years ago were not incorrect. It is the fact that the old Iola pool, which was the original source of supply, has been exhausted, but new pools have been discovered to the north and west, from which these smelters have been obtaining their gas. In 1907 some of the smelters united in putting down a deep, experimental well, in East Iola, to prospect for gas at a lower horizon. By the middle of November this had attained a depth of 2500 ft. and there was not much hope that the prospecting would result successfully.

Smelting Capacity.—In 1907 there was a large increase in the zinc-smelting capacity of the United States. The National Zinc Company, Bartlesville Zinc Company, and Lanyon-Starr Smelting Company each completed new works at Bartlesville, Oklahoma. The American Zinc, Lead and Smelting Company completed a new plant at Deering, Kansas. The United Zinc and Chemical Company completed a small plant at Springfield, Ill. Several of the works above mentioned did not begin operation until December, therefore they had no material effect upon the production of 1907. The works of the Mineral Point Zinc Company, at Depue, Ill., under construction in 1906, were in operation with three furnaces during the first half of 1907. The works of Hegeler Brothers at Danville, Ill., were completed in 1907, but did not go into operation. James Latourette built a small plant at Clarksburg, W. Va., to smelt galvanizers' dross.

Among the older works there were also many increases in capacity in 1907. At Palmerton, Penn., the New Jersey Zinc Company added one furnace with 200 retorts. The Granby Mining and Smelting Company added one new furnace with 620 retorts to its works at Neodesha, Kan. The Grasselli Chemical Company added a new furnace with 576 retorts to its works at Clarksburg, W. Va. The Cockerill Zinc Company added a new furnace with 740 retorts to its works at Altoona, Kan.

At the end of 1907, the list of American zinc smelters, together with their number of furnaces and number of retorts, was as shown in Table III, which is substantially correct, although there may be slight errors

in the case of a few companies; however, these will not materially affect the total. It will be seen from this table that at present there is a great surplus in smelting capacity. The total number of retorts is 87,640, and of muffles is 1920. These give capacity for smelting annually about 625,000 tons of roasted ore (or calamine), which would correspond to about 750,000 tons on the basis of blende. The ore purchased by the smelters in 1907, excluding the Pennsylvania and Virginia works of the New Jersey Zinc Company, amounted to 644,170 tons, but of this about 93,000 tons was used by the Western oxide works. The combined production of the Joplin and Wisconsin districts, together with that of Arkansas and Oklahoma, was 357,475 tons, leaving only 193,695 tons of ore for making spelter as coming from other districts, chiefly those west of the Rocky mountains and in Mexico. The Joplin district does not appear to be able to make any large increase in production; the output of the Wisconsin district is steadily increasing, but it does not yet play a very large part in the domestic zinc industry. At the end of 1907 there was a lull in smelter construction, due partly to the industrial setback and partly to the realization that the smelting capacity is now sufficient for the needs in the near future. At present, so far as I am

III. ZINC SMELTING CAPACITY OF THE UNITED STATES.

Name.	Location.	Furnaces.	Retorts.
Grasselli Chemical Co.	Clarksburg, W. Va.	10	5,760
Matthiessen & Hegeler Zinc Co.	Lasalle, Ill.	5	4,320
Illinois Zinc Co.	Peru, Ill.	7	4,800
Sandoval Zinc Co.	Sandoval, Ill.	4	896
Mineral Point Zinc Co.	Depue, Ill.	6	4,800
Hegeler Bros.	Danville, Ill.	2	1,700
Edgar Zinc Co.	St. Louis, Mo.	9	2,016
Edgar Zinc Co.	Cherryvale, Kan.	24	4,800
Lanyon Zinc Co.	Iola, Kan.	5	3,000
Lanyon Zinc Co.	Iola, Kan.	5	3,000
Lanyon Zinc Co.	Iola, Kan.	5	3,000
United Zinc and Chemical Co.	Iola, Kan.	4	2,304
United Zinc and Chemical Co.	Iola, Kan.	2	(a)480
United Zinc and Chemical Co.	Springfield, Ill.	2	640
Cockerill Zinc Co.	Iola, Kan.	4	2,520
Cockerill Zinc Co.	Iola, Kan.	3	1,856
Cockerill Zinc Co.	Altoona, Kan.	6	3,840
Cockerill Zinc Co.	Pittsburg, Kan.	6	1,344
Cockerill Zinc Co.	Nevada, Mo.	3	672
Cockerill Zinc Co.	Rich Hill, Mo.	3	672
Granby Mining and Smelting Co.	Neodesha, Kan.	6	3,840
United States Zinc Co.	Pueblo, Colo.	6	(a)1,440
American Zinc, Lead and Smelting Co.	Caney, Kan.	6	3,720
American Zinc Lead and Smelting Co.	Deering, Kan.	6	3,840
New Jersey Zinc Co.	Bethelehem, Penn.	3	672
New Jersey Zinc Co.	Palmerton, Penn.	12	2,400
Bertha Mineral Co.	Pulaski, Va.	10	1,400
Prime Western Spelter Co.	Iola, Kan.	9	5,344
Prime Western Spelter Co.	Iola, Kan.	5	3,220
Bartlesville Zinc Co.	Bartlesville, Okla.	6	3,456
Pittsburg Zinc Co.	Pittsburg, Kan.	4	896
Lanyon-Starr Smelting Co.	Bartlesville, Okla.	5	2,880
National Zinc Co.	Bartlesville, Okla.	4	2,432
Chanute Zinc Co.	Chanute, Kan.	8	1,600
Totals		205	89,560

(a) Rhenish furnaces, with muffles.

aware, there is no large plant under construction or in definite contemplation, and the existing smelters are making no further additions to their plants.

Consumption.—The production of virgin spelter in 1907 was 249,612 tons. The stock on hand at the beginning of the year was 4550 tons. The imports were 1778 tons. The total supply was consequently 255,940 tons. The exports amounted to 563 tons, and the stock on hand at the end of the year was 32,883 tons. The computed consumption, consequently, was 222,494 tons. In addition to the consumption of virgin spelter there is a considerable consumption of spelter produced by resmelting of galvanizers' dross and other waste products. Estimating this supply at 3000 tons, the total computed consumption in the United States in 1907 was 225,494 tons.

As in previous years I have sought to check the computed consumption and separate it according to uses by direct reports from the consumers. Most of the large consumers (manufacturers) promptly and willingly report their figures, but there are many small users of spelter from whom it is next to impossible to secure reports. Fortunately, these are chiefly users of spelter for brass-making and for what is classed as "other purposes." All of the manufacturers of sheet zinc made reports for 1907, as did also practically all of the galvanizers. The consumption of spelter for the desilverization of lead is readily computed from the amount of lead desilverized. Consequently there is a sound basis for estimating the consumption of spelter for brass-making and for other purposes by difference. A very large part of the consumption for 1907 was actually accounted for by direct reports from the manufacturing consumers. The classification of the consumption is given in Table VIII.

Table VIII is highly interesting in its showing. The consumption for galvanizing increased largely, as, of course, was known in a general way to the smelters during the year, their orders from the galvanizers being unusually large up to the third quarter. The consumption for the manufacture of sheet zinc fell off largely. In this business the industrial setback was clearly manifest as early as June, and by July the manufacturers were reporting a serious decrease in orders. The huge falling off in the consumption of spelter for brass-making indicates that adverse conditions must have developed equally early in the brass industry. Indeed, we know from the history of the copper industry that such must have been so, the depression in business dating back at least to April, 1907. I estimated that in 1906 the domestic consumption of copper for brass-making was about 114,000 tons. On the same basis the consumption in 1907 was only 80,000 tons. The apparent falling off in the consumption of zinc for "other purposes" in 1906 is explained by a more complete itemization of the consumption in 1906; in other words, there was probably more

spelter used for brass-making in 1905 than the table shows. The great increase in the proportion used for galvanizing during the last three years is a particularly noteworthy feature of the industry.

IV. EXPORTS OF ZINC ORE AND ZINC OXIDE FROM THE UNITED STATES. (a)

Year.	Ore.			Oxide.		
	Short tons.	Value.	Value per ton.	Short tons.	Value.	Value per ton.
1896	(b) 2,324	$47,408	$20.40	(c)	......	......
1897	9,251	211,350	22.85	1,859	$104,140	$56.02
1898	11,782	299,970	25.50	3,925	252,194	64.25
1899	28,221	725,944	25.90	5,343	366,598	68.61
1900	42,062	1,134,663	26.98	5,656	496,380	87.76
1901	44,146	1,167,684	26.45	4,561	393,259	86.22
1902	55,733	1,449,104	26.00	5,358	433,722	80.93
1903	39,411	987,000	25.04	7,215	578,215	80.14
1904	35,911	905,782	25.22	8,157	628,494	77.05
1905	30,946	848,451	27.41	11,280	810,203	71.83
1906	27,720	733,300	26.45	15,578	1,149,297	73.78
1907	20,352	579,490	28.47	13,256	1,069,924	80.71

(a) In addition to the exports of ore, 9,953 short tons of zinc dross (galvanizers' waste) were exported in 1907, against 15,887 tons in 1906. (b) Includes oxide. (c) Included in ore.

V. EXPORTS OF DOMESTIC SPELTER FROM THE UNITED STATES. (a)

Year	Plates, Sheets.	Pigs and Bars.	Wares.	Total Value.
	Short Tons.	Value.	Value.	
1896	10,150	$1,013,620	$51,001	$1,112,029
1897	14,245	1,356,538	71,021	1,743,049
1898	10,499	1,033,959	138,165	1,724,188
1899	6,755	742,521	143,232	1,978,295
1900	22,411	2,217,963	99,288	2,317,251
1901	3,390	228,906	82,046	310,952
1902	3,237	300,557	114,197	414,754
1903	1,521	163,379	71,354	234,733
1904	10,073	1,094,490	117,957	1,212,447
1905	5,516	682,254	159,995	842,249
1906	4,670	583,526	204,269	787,795
1907	563	75,526	186,283	261,397

(a) There is also a comparatively insignificant re-export of foreign-made spelter and zinc wares.

VI. IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES.
(In pounds.)

Year.	Sheets. Blocks, Pigs and Old.		Manufactures	Total Value.	Oxide.	
	Amount.	Value.			Dry.	In Oil.
1896	856,044	$25,904	$15,728	$41,632	4,572,781	311,023
1897	2,557,341	95,883	19,431	115,314	5,564,753	502,357
1898	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899	2,985,463	151,956	14,800	166,756	3,012,709	41,699
1900	2 013,196	97,772	36,836	134,608	2,618,808	38,706
1901	775,881	30,920	42,643	73,563	3,199,778	128,198
1902	1,238,091	46,713	37,191	83,904	3,271,385	163,081
1903	728,614	30,900	18,938	49,838	3,487,042	166,034
1904	933,474	44,326	11,918	56,244	2,585,661	224,244
1905	1,042,081	51,052	12,390	63,442	3,436,367	342,944
1906	4,407,481	253,310	17,385	270,695	4,191,476	292,538
1907	3,555,890	210,322	16,282	226,604	5,311,318	362,418

VII. CONSUMPTION OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

	1905	1906	1907
Stock, Jan. 1	6,500	4,000	4,550
Production	201,748	225,494	249,612
Imports	521	2,203	1,778
Total supply	208,769	231,697	255,940
Exports	5,515	4,670	563
Stock, Dec. 31	4,000	4,550	32,883
Consumption	199,254	222,477	222,494

VIII. USES OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

Purpose.	1905	1906	1907	1905	1906	1907
Galvanizing	100,000	124,000	149,000	50%	55%	66%
Brass-making	52,000	57,000	40,000	26	25¼	17¾
Sheet zinc	34,000	36,000	30,000	17	16	13¼
Lead desilverization	2,400	2,500	2,600	1¼	1	1¼
Other purposes (a)	10,854	6,000	4,000	5¾	2¾	1¾
Total	199,254	225,500	225,600	100	100	100

(a) The apparent falling off in the consumption of zinc for "other purposes" in 1906 is explained by a more complete itemization of the consumption in 1906; in other words, there was probably more spelter used for brass-making in 1905 than the above table shows.

Zinc Oxide.—In 1907 four companies were engaged in the manufacture of zinc oxide and zinc-lead pigment direct from ores. Their production, together with the corresponding figures for 1906, is summarized in Table IX. It is to be remarked that this table no longer represents the production of zinc oxide proper, inasmuch as certain of the companies make both the oxide and zinc-lead pigment, the latter being produced by the treatment of mixed ores in substantially the same way that zinc ore is treated alone. Inasmuch as the two products are consumed for the same purpose, namely, as pigment, they are properly lumped together in the statistics. It is impossible to report them separately without disclosing the business of individual producers.

IX. PRODUCTION OF ZINC OXIDE IN THE UNITED STATES. (a)

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1896	15,863	14,391	$1,189,725	$75.00	1902	52,730	46,929	$4,023,299	$76.30
1897	26,262	23,285	1,686,020	64.26	1903	59,562	54,034	5,005,394	83.69
1898	32,747	29,708	2,226,796	68.00	1904	59,613	54,081	4,523,414	75.88
1899	39,663	35,982	3,331,692	84.00	1905	72,603	65,859	5,808,240	80.00
1900	47,151	42,775	3,772,080	80.00	1906	77,800	70,573	6,257,361	80.43
1901	46,500	42,266	3,720,000	80.00	1907	85,390	77,449	7,731,100	73.28

(a) The figures for 1905 and 1906 include zinc-lead pigment, which was not included in the statistics for previous years.

Production of Zinc Ore.

The production of zinc ore in the United States, Canada and Mexico in 1906 and 1907 is given in Table X. The figures in this table are based on reports received from the various smelters, showing their purchases of ore, classified according to States, Territories or country of origin. Such reports were received from all of the smelters with two exceptions, whose statistics were compiled on the basis of reports received as to the shipments of ore to them. For this reason the total may be a little too low, especially in the case of certain of the minor States, including Arkansas.

Although the total for the United States showed a slight falling off as compared with 1906, it is to be noted that this was due to the decrease in the State of New Jersey, the figures for which represent the crude ore produced by the Franklin mine of the New Jersey Zinc Company. The large decrease in the production of New Mexico was due to the smaller output of the mines at Magdalena, which were worked on a comparatively small scale pending the installation of works for the separation of mixed products. With these exceptions, most of the important zinc-producing States showed increases in 1907.

It is worthy of remark that the reports of the smelters as to ore received from the Joplin and Wisconsin districts show in each case slightly larger amounts than reported by Jesse A. Zook and J. E. Kennedy, the local statisticians, who are also correspondents of the *Engineering and Mining Journal.* Deducting the shipments from Oklahoma, Mr. Zook's figure for the Joplin district is 283,433 tons, while the smelter figure is 295,626 tons. Mr. Kennedy's report for Wisconsin was 49,778 tons, while the smelter figure is 53,011 tons. These differences may be accounted for by ore in transit, and also probably by various small omissions, but both for the Joplin district and the Wisconsin district the results are so close that great credit is reflected on the local statisticians and is strong testimony as to the reliability of their work.

All of the Western smelters in 1907 purchased ore from the Joplin district to more or less extent, with two exceptions, both of the latter using Colorado ore almost exclusively. The output of the Wisconsin district was purchased by six smelters. Mexican ore was imported by 15 smelters. Nearly all of the Western smelters used ore from Mexico and from the States and Territories west of the Rocky mountains to more or less extent, the important exceptions being the companies at Lasalle and Peru, Ill., whose supply was derived chiefly from the Joplin and Wisconsin districts. The major portion of the output of the Joplin district was purchased by six concerns, each of these obtaining therefrom supplies of ore in excess of 23,000 tons.

In spite of the steady decline in the values of spelter and ore during the

second half of 1907, and the financial catastrophe in October, the production of ore in the Joplin district and in the Wisconsin field showed a considerable increase. What might have happened if it had not been for the set-back is manifested by the statistics for the first six months of 1907, which show shipments of 158,000 tons from Joplin and 25,000 tons from Wisconsin. On the other hand, many of the districts west of the Rocky mountains made smaller outputs than in 1906, indicating that the productive capacity of their mines has reached a maximum. The importation of ore from Mexico was checked when the price for spelter fell to 5c., St. Louis, in September, but nevertheless the total for the year was largely in excess of that of 1906. But little ore was brought in from British Columbia at any time, the supply in that province being blende. During 1907 all importations of zinc ore were made with payment of duty under protest, the decision of the Board of Appraisers having been appealed to the courts by the Government. A decision from the U. S. Circuit Court for southern Texas, upholding the previous decision of the Board of General Appraisers in favor of the importers, was rendered in April, 1908. The experience of 1907 seems to show that these foreign ores are not serious competitors of the American, duty or no duty.

X. PRODUCTION OF ZINC ORE IN THE UNITED STATES.
(In tons of 2000 lb.)

State.	1904	1905	1906	1907
	Tons.	Tons.	Tons.	Tons.
Arkansas	(e) 1,900	2,200	4,200	4,088
Colorado	(a) 94,000	105,500	114,000	142,510
Idaho	*Nil.*	1,700	2,150	11,847
Kentucky	(d) 958	(d) 414	975	1,005
Miss.-Kan.	(b)273,238	(b)258,500	(b)280,260	297,126
Montana	*Nil.*	2,000	4,900	1,218
Nevada	*Nil.*	*Nil.*	7,080	4,593
New Mexico	(e) 21,000	17,800	30,000	4,281
New Jersey	(d)280,029	(d)361,829	404,690	368,710
Oklahoma	....	....	....	3,240
Utah	*Nil.*	9,265	10,700	9,043
Wisconsin	(c) 19,300	32,690	42,130	53,011
Others	(a) 2,600	(f) 3,800	(h) 850	(h) 2,241
Totals	693,025	795,698	905,175	902,923

(a) Estimated. (b) Production of Joplin district, plus output of southeastern Missouri, the latter as reported by the State mine inspector. (c) According to H. F. Bain, "Contributions to Economic Geology," 1904. (d) Report of State Geologist; crude ore. (e) Partly estimated. (f) Arizona, Nevada, Illinois, Iowa, Tennessee and Virginia. (h) Tennessee, Arizona and California.

XI. IMPORTS OF ZINC ORE INTO THE UNITED STATES.
(In tons of 2000 lb.)

Source.	1904	1905	1906	1907
British Columbia	2,100	8,561	600	1,157
Mexico	?	(a)32,164	(a)88,900	(a)108,800
Totals	?	40,725	89,500	109,957

(a) The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore, which for statistical purposes has been reduced to the zinc ore equivalent. This table is based on reports from the smelters of the ore received by them from these countries.

Margin on Joplin Ore.—During the early part of 1907 the margin on Joplin ore was strongly in favor of the smelters and they realized handsome profits, but in July, when the price for spelter began to fall off sharply, the margin diminished to less than the smelters should normally have in order to earn a fair return on their investment, and in subsequent months the margin fell to the point where there was no profit at all. The monthly figures are given in the following table, whereof the first column gives the value of 1020 ℔. of spelter at St. Louis (85 per cent. of 1200 ℔.), the second column the value of blende containing 60 per cent. zinc at Joplin, and the third column the difference, or margin.

XII. MARGIN ON JOPLIN ORE.

Month.	Spelter.	Ore.	Margin.	Month.	Spelter.	Ore.	Margin.
January	$67.14	$46.90	$20.24	July	$60.40	$46.80	$13.60
February	67.97	48.30	19.67	August	56.62	44.56	12.06
March	68.21	49.75	18.46	September	51.88	41.00	10.88
April	66.66	49.25	17.41	October	53.86	41.75	12.11
May	64.17	46.90	17.27	November	48.71	38.60	10.11
June	63.94	47.00	16.94	December	41.86	31.50	10.36

Conditions in the Joplin District.—The decline in the value of ore was felt keenly both in the Joplin district and in Wisconsin, where the market price fell below the actual cost of production in so far as a large proportion of the output is concerned. The difficulties of the situation were enhanced by the simultaneous decline in the value of lead ore. An attempt to check the decline by restriction of production by agreement among the Joplin operators was made in September, but although this had a temporary effect, it was of course ineffectual in combating the general industrial depression that was impending. In November, when it became a question with many operators either to close down or to reduce the cost of production, a reduction in wages was made, drill-runners, who had previously been receiving $3 per eight hours, being reduced to $2.50, while back-hands, who had been receiving $2.25@2.50, were reduced to $1.75@2, and muckers and trammers, who had been receiving 6@7c. per can (1000 ℔.), were reduced to 4@5c. In all cases these reductions were received by the men in good humor, and indeed, in many cases the men themselves proposed that the reductions should be made.

It is impossible to generalize the cost of production at Joplin, so many different factors entering into the consideration. The most important is, of course, the yield of the ore, after which are the factors of cost of mining and milling and the amount of royalty to be paid. Assuming that blende containing 60 per cent. zinc is worth $40 per ton, and lead ore $42 per ton, and that one-sixth of a ton of lead ore is obtained with each ton of zinc ore, the gross value of the two concentrates is $47 per ton. If

a royalty of 20 per cent. is to be paid, the net value to the operator is \$37.60 per ton. If it be necessary to mine 33 tons of ore to obtain one ton of blende concentrate, i.e., to obtain a yield of about 3 per cent., and the cost of mining and milling be \$1.10 per ton, the total cost of production is \$36.30, leaving a profit of only \$1.30 for the mining and milling of 33 tons of ore, or only 4c. per ton of ore hoisted from the mine.

A good deal of ore was produced in 1907 under precisely the above conditions. On the other hand there was more or less concentrate produced from ore of a better grade, even up to a yield of 5 per cent., or at a lower royalty, or perhaps even at a little lower cost for mining and milling, which could be produced profitably at less than \$40 per ton for blende concentrate, and the revival in the Joplin district during 1908 indicates that it has been possible to readjust conditions so as to make a large production with zinc ore commanding less than \$40 per ton.

It is obvious that a great drawback to mining in the Joplin district is the excessively high royalty that must be paid to the owners of the land. The methods of mining and milling in the district have been raised to a high degree of efficiency, especially the mining, and in respect to the amount of work done per man there are few mining districts in the world which surpass the Joplin district. In the matter of milling there is considerable room for further improvement, particularly in the reduction of losses. It is believed that the actual extraction from the ore of the sheet ground, which now furnishes nearly a half of the Joplin output, is about 65 per cent. on the average, which may be increased to 70 per cent. or possibly more, by only a small addition to the cost of the milling process. In this particular and in the readjustment of royalties lie the future prospects of the Joplin district if the price for ore continues low.

Production in the Rocky Mountains.—The cost of producing zinc ore in the districts to the west of the Rocky mountains is variable and uncertain. In general the cost of mining is much higher than at Joplin, but also the crude ore is much higher in grade, although the component minerals are not so easily separated. Leadville, which is the largest single source of supply among these districts, began the production of zinc ore as a by-product for which \$5 per ton f.o.b. cars was a welcome price, but the zinc-lead ore originally mined, from which the blende was so obtained, has been largely exhausted and the recent production has been made chiefly from blende-pyrites from which a commercial grade of zinc ore cannot be cheaply produced. In 1907 a good deal of Leadville zinc ore was sold on the formula of T—8 less a returning charge of \$15 per ton, basis Iola. However, other ores were brought at much lower terms, and indeed during the latter half of the year the only margin of profit to the smelters was on their purchases of Western ore. When the price of spelter fell to the neighborhood of 4c., St. Louis, a good many of the Leadville

producers found that there was no more profit in the business and suspended operations.

The great drawback to a large production of zinc ore in the Rocky mountains is the costliness of the processes required to bring the zinc concentrate up to good commercial grade. The first cost of a magnetic separating plant is high, and the operating cost also is high. The principal plants of this character are those of the Empire Zinc Company, at Cañon City, Colo., and of the American Zinc Extraction Company, at Leadville, Colo.; other plants in regular operation are those of the Colorado Zinc Company, at Denver, Colo., and the United States Zinc Company, at Pueblo, Colo. Outside of these plants, however, neither magnetic separation nor electrostatic separation has yet proved to be a great new source of ore supply. Of course this does not refer to Wisconsin.

Some interesting new installations in this line were made in 1907. Among these was the completion of the mill at the Tiro General mine, at Charcos, in Mexico, where the Sutton-Steele pneumatic tables and dielectric separator are used. Magnetic separation was conducted on a large scale in Wisconsin, but in some cases it was found that the roasting-separating process was too expensive, and improvement is hoped from the electrostatic process, which is to be given another trial in Wisconsin. Danger of litigation over this process was eliminated in 1907 by the combination of the interests controlling the Blake and the Huff separators, which are now in the hands of parties connected with the American Zinc, Lead and Smelting Company.

Review of Zinc Mining by States.

California.—The Western Zinc Company made a small production from its mine at Silverado, Orange county. The company is reported to be planning the installation of a metallurgical plant at San Francisco.

Colorado.—Leadville continues to be the chief source of zinc production in this State, although shipments were made in 1907 from numerous other camps. A large tonnage of zinc silicate was shipped from the Madonna mine at Monarch, Colorado. This mine has large bodies of low-grade ore, assaying 15 per cent. zinc, 5 to 10 per cent. lead (as carbonate) and the remainder iron (oxidized). The ore contains 4 to 5 oz. silver per ton, and about 0.05 oz. of gold. Some efforts have been made to work this ore, but the minerals are intimately mixed, and the problem is a difficult one.

The American Zinc and Chemical Company, of Denver, which has been

exploiting the Dewey zinc extraction process, decided to wind up its affairs. The company failed to develop the process on a paying basis. The process consisted of sulphate roasting, leaching of zinc sulphate, evaporating to dryness and calcining to zinc oxide.

Idaho (By Robert N. Bell).—In 1907 Idaho produced zinc ore containing 4596 tons of zinc, of which about two-thirds came from the Success mine, about four miles north of Wallace, and one-third from the Frisco mine, on Canyon creek, both in the Coeur d'Alene. The Success mine was operated steadily until Dec. 10, when it was shut down because of the low price of zinc; its 100-ton mill has been doubled in capacity, the addition having been completed in November. The Success mine has developed above its lowest adit about 150,000 tons of sulphide ore averaging 20 per cent. zinc, 2 to 8 per cent. lead, and 4 to 8 per cent. iron.

Illinois (By H. Foster Bain).—Illinois shared in the prosperity of the lead and zinc industry in the early part of 1907. In the northwestern part of the State, Galena continued to be the center of active prospecting and development. The Northwestern Lead and Zinc Company continued shipments. The Vinegar Hill, Dinsdale and Drill Hole mines began producing, the old Black Jack mine was reopened and more or less work was done on a number of other properties. Local statisticians credit Galena with shipments of 7,035,580 lb. of zinc ore in 1907, but these figures possibly include some reshipment of ore owing to the presence of the Joplin Separating Company, which reconcentrates and cleans low grade blende-marcasite ores purchased throughout the upper Mississippi Valley. The really significant feature of 1907 as regards this district would seem to be the fact that shafts sunk as a result of prospecting with the drill have shown bodies of ore as good or better than were anticipated. This confirmation of results lends security to a very cheap and readily applied method of prospecting in this area of shallow depths in flat-lying rocks. It is to be noted, however, that these satisfactory results only followed painstaking work by skilful men in the direction of the drilling and the study of its results. On the whole the tendency of 1908 was decidedly toward the use of higher grade machinery and higher grade of talent throughout the field. The forced shut down at the close of the year seems likely to prove a blessing in disguise in the introduction of economies in production and a lower wage scale. Late in January several of the mines resumed work.

Outside the northwestern corner of the State there were no developments of lead and zinc to record. The lead found in the southwestern part of the State occurs so far as is yet known only in the form of scattered masses of galena in residual clay. In the southern counties in connection with the mining of fluorspar a small amount of galena, notable for a silver content, is regularly shipped to St. Louis. The old Eureka mine, now

being reopened, formerly produced a rather heavily leaded ore and similar ore is said to remain in the mine. Zinc occurs, but, except where reconcentrated as the carbonate in surface deposits, has not so far proved of commercial importance owing to its intimate combination with the fluorspar. The surface deposits so far found north of the Ohio have proved small.

Iowa (By James H. Lee).—The year 1906 was the first one for some time in which any zinc ore had been marketed from the Dubuque field, and thus the production during 1907 of over 300 tons of zinc concentrates indicates the rapidity of the rejuvenescence of this industry. Also a number of carloads of dry bone were shipped. Several mines were opened and equipped with up-to-date mining machinery, and three mills were constructed. The Avenue Top mine is making a fine showing and is being developed as rapidly as possible. Its mill was put in operation during 1907. The Dubuque & Lake Superior Mining Company has let the contract for a 150-ton mill. A recent assay of the ore from its property showed about 25 per cent. zinc with 0.5 per cent. copper.

Kentucky.—Little or nothing was done in the Marion zinc district during 1907. The inability to market zinc blende mixed with fluorspar has largely stopped mining for the present.

Missouri.—The general conditions in the Joplin district have been summarized in previous paragraphs of this article, and are further discussed in the following notes by Dr. Buckley and Mr. Zook.

XIII. AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN, MO.

(Dollars per 2000 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1900.....	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901.....	23.72	23.96	23.70	24.58	24.38	24.22	24.08	23.88	21.63	21.63	26.15	28.24	24.21
1902.....	26.75	27.00	28.00	28.85	29.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903.....	31.50	32.50	35.75	37.75	36.60	36.50	36.00	36.00	34.40	34.40	30.75	30.00	34.44
1904.....	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905.....	51.94	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40
1906....	49.33	49.25	45.60	44.00	41.50	44.20	43.88	44.38	43.20	42.50	44.43	45.55	44.82
1907....	46.90	48.30	49.75	49.25	46.90	47.00	46.80	44.56	41.00	41.75	38.60	31.50	44.36

(By E. R. Buckley.) Although Jasper, Newton and Lawrence counties in the southwestern part of the State are the chief producers of zinc ore, yet the wide distribution of this ore throughout the Ozark region has naturally resulted in the discovery and development of small deposits in many other counties. Likewise in the case of the lead ores, while

St. François, Madison, Newton, Jasper and Lawrence counties are the chief producers, yet other counties have added considerably to the total output of the State. While Jasper, Newton and Lawrence counties are large producers of lead as well as zinc ore, Madison county produces no zinc and St. François county very little. About one square mile in Sections 17 and 18, T. 38 N., R. 5 E., in St. François county, has been producing zinc ore for 38 years at the rate of 1200 to 4000 short tons of concentrates annually. This is what is known as the Valle Mines area. There is also considerable blende—often from 3 to 4 per cent.—associated with the disseminated galena in the mines near Leadwood in St. François county but no attempt is made to recover it. There are a few small areas in the disseminated lead belt that are known to contain blende in sufficient quantity to be workable, but as long as the disseminated lead deposits hold out the zinc will probably remain undeveloped.

In the southwestern part of Missouri the productive area is being gradually extended in all directions. Prospecting has been especially active in the vicinity of Carthage, Wentworth, Sarcoxie and Pierce City to the east; in the vicinity of Neosho, Spurgeon and Newtonia to the south; and in the vicinity of Carl Junction, Alba, Neck and Purcell to the north. There does not appear to be any likelihood that the production from this district will lessen materially for a great many years.

Attention is being directed to the enormous waste constantly going on as a result of the leasing system and milling methods of this district. It is conservatively estimated that the quantity of zinc and lead minerals left in the ground, beyond the possibility of future recovery, through imperfect mining methods and the exaction of high royalties, and the amount lost through the present milling system will aggregate fully 60 per cent. of the total mineral occuring in the orebodies. It is difficult to see how the leasing system can ever be replaced by any other, but the attention now being given to securing a better extraction in the mills will certainly result in a decrease in the waste from this direction.

In St. François and Madison counties, which constitute the disseminated lead belt, there have been some new orebodies discovered and considerable additions have been made to the known orebodies. There are about 40 operating shafts in the district and several that will probably be put into operation during 1908. There are 12 mills in operation and one in process of construction, with a total daily milling capacity of 12,650 short tons. The mining properties are all controlled by eight companies, three of which are operating in Madison county and five in St. François county.

During 1907 several interesting occurrences of lead and zinc have been brought to my attention. One of these is in Wayne county and consists of a dike of diabase in granite through which is disseminated

blende and galena. The ore contains about 10 per cent. of these minerals. Near Munger in Iron county a vein of galena in granite has been uncovered, but as far as I know it has not been developed. Attempts to develop similar veins in granite years ago did not prove successful.

(By Jesse A. Zook.) The year 1907 opened auspiciously upon a scene of satisfactory conditions in zinc-lead mining in the Joplin district, with prices of both minerals at a high point, and this condition continued so evenly until the end of June that the expectation that 1907 would so far exceed 1906 as to eclipse all previous records of increase was fully justified. During July and August prices dropped slightly, but no realization of a 50 per cent. reduction in the price of lead concentrate and 40 per cent. reduction in the price of zinc concentrate could be conjectured from what was believed at that time to be only the lowering of prices on account of the large reserve stock of concentrate then in the bins.

The accompanying tabulation shows the shipments for 1906 and 1907. The first section of the table embraces the "sheet ground" area, extending from Oronogo on the north to Duenweg on the south. The second section embraces all territory west and south of Oronogo-Duenweg sheet ground to the State line. The third section is from the sheet ground to the east line of Jasper county. The fourth section embraces the Lawrence, Green and Morgan county production. The fifth section is the producing area of Cherokee county, Kansas. The sixth section is the producing lands of Ottawa county, Oklahoma.

The Oronogo-Duenweg "sheet ground" section increased its production of zinc ore by 20,648 tons, while increases in all other sections of the district totaled only 7761 tons, to offset which all other points give an aggregate decrease of 21,882 tons.

Joplin's large decrease of zinc ore production indicates the working out of the older levels in the Chitwood group. Its increase in lead ore came from all sides of the camp. The Webb City-Carterville increase came wholly from the new developments north of those towns, a large proportion of which is in the corporate limits of Oronogo. Oronogo's increase came from deeper levels and improved machinery. The Prosperity increase came from new mines south and east toward the Porto Rico group in the Richland valley, and in the further development of the divide from Prosperity to Duenweg.

The larger tonnage of increased production coming from the "sheet ground" mines of Oronogo, Webb City, Carterville Prosperity and Duenweg, substantiates a statement made last year that this production could not be maintained under an average of $45 per ton for zinc ore. It may be added this year that a price of $60 per ton for lead is an additional necessity. This is not a rule applicable individually to the sheet ground mines, but including all of them in an average. These prices for zinc and lead ore

will not put all of the sheet ground mines opened during 1907 on a paying basis.

Prospecting during 1907 indicates that the new production of 1908 will come from the sheet ground discoveries on the divide between Turkey and Short creeks, at the head of Chitwood and Leadville hollows, west of Joplin, from all the camps showing a large percentage of increase for 1907, from points along Spring river between Alba and Carthage, and northeast from Alba. The further development of lower levels at Granby promises to restore that camp to the ranks of those showing an increased zinc production.

The total shipment of zinc concentrate for 1907 was 286,589 tons, at a value of $12,521,423; the total shipment of lead concentrate was 41,742 tons at a value of $2,898,404; the total shipment of both zinc and lead concentrate was 328,331 tons at a value of $15,419,827.

XIV. SHIPMENTS OF ORE FROM THE JOPLIN DISTRICT.

(In tons of 2000 lb.)

Year.	Zinc Ore.	Lead Ore.	Year.	Zinc Ore.	Lead Ore.
1894	147,310	32,190	1901	258,306	35,177
1895	144,487	31,294	1902	262,545	31,625
1896	155,333	27,721	1903	234,873	28,656
1897	177,976	30,105	1904	267,240	34,362
1898	234,455	26,687	1905	252,435	31,679
1899	255,088	23,888	1906	278,930	39,189
1900	248,446	29,132	1907	286,589	41,742

Zinc concentrate sold as high as $50 per ton during the first week of 1907, advancing 50c. each of the following two weeks and closing the month at $51. February opened with $52.50 and closed at $53.50, holding at this until the last week in March when it closed at $52.50. Back to $53.50 at the opening of April it dropped to $53, and then to $52 in the last fortnight. May started at $51 and ended at $49. Through June the price was $51, but in July it dropped to $50, then resting until the second week in August when it went down to $49, then to $48 and $47, closing the month at $46. During the first two weeks of September it was $45 and the last two weeks $43. October marked a reaction to $44, $45.50, $46.50, closing at $46. November opened at $43.50, dropping to $42.50, $41.50, $40, $38, and in December to $36.

The highest price paid for first-grade zinc concentrate was $53.50 in the last two weeks in February, all of March and one week in April. The base price for 60 per cent. zinc at this time was $48@51. The lowest base price was in December, $28@34 per ton. The average base price of the year ranged approximately from $40@42.

XV. PRODUCTION OF THE JOPLIN DISTRICT

(In tons of 2000 lb.)

	Zinc Ore.				Lead Ore.			
	1907	1906	Inc.	Dec.	1907	1906	Inc.	Dec.
Oronogo	10,537	8,768	1,769	....	578	359	219	....
Webb City-Carterville	78,491	64,172	14,319	....	18,864	17,262	1,602	....
Prosperity	8,010	4,498	3,512	....	3,026	2,754	272	....
Duenweg	19,032	17,984	1,048	....	3,555	3,963	....	408
Totals	116,070	95,422	20,648	....	26,023	24,338	2,093	408
			20,648				1,685	
Carl Junction	997	46	951	....	114	....	114	....
Sherwood	1,156	3,206	....	2,050	20	339	....	319
Zincite	1,876	1,592	284	....	80	43	37	....
Cave Springs	1,030	1,151	....	121	17	....	17	....
Joplin	59,336	68,744	....	9,408	7,478	6,813	665	....
Spurgeon	6,898	5,691	1,207	....	1,575	870	705	....
Diamond	38	44	....	6	....	....	....	....
Granby	13,826	10,895	2,931	....	1,343	985	358	....
Totals	85,157	91,369	5,373	11,585	10,627	9,050	1,896	319
				6,212			1,577	
Alba-Neck	22,924	25,270	....	2,346	157	365	....	208
Carthage	2,438	3,575	....	1,137	9	62	....	53
Reeds	403	87	316	....	....	....	....	....
Sarcoxie	1,473	229	1,244	....	....	5	....	5
Wentworth	569	368	201	....	....	....	....	....
Totals	27,807	29,529	1,761	3,483	166	432	....	266
				1,722				266
Stott City	918	1,161	....	243	....	20	....	20
Aurora	13,300	15,147	....	1,847	342	345	....	3
Ash Grove	....	....	....	....	33	....	33	....
Springfield	193	324	....	131	255	44	211	....
Morgan County	....	135	....	135	....	4	....	4
Totals	14,411	16,767	....	2,356	630	413	244	27
				2,356			217	
Galena	27,762	29,553	....	1,791	3,572	4,190	....	618
Playter	298	80	218	....	17	4	13	....
Badger	11,900	14,100	....	2,200	46	43	3	....
Lawton	28	....	28	....	....	....	....	....
	39,988	43,733	246	3,991	3,635	4,237	16	618
				3,745				602
Quapaw-Baxter	2,775	3,242	....	467	661	669	....	8
Peoria	321	....	321	....	....	....	....	....
Miami	60	....	60	....	....	....	....	....
Totals	3,156	3,242	381	467	661	669	....	8
				86				8
District	286,589	280,062	28,409	21,882	41,742	39,139	4,249	1,646
			6,527				2,603	

XVI. PRODUCTION BY COUNTIES AND STATES.

(In tons of 2000 lb.)

	Zinc Ore.				Lead Ore.			
	1907	1906	Inc.	Dec.	1907	1906	Inc.	Dec.
Jasper County	207,703	199,322	8,381	....	33,898	31,965	1,933	....
Newton County	21,331	16,998	4,333	....	2,918	1,855	1,063	....
Lawrence County	14,218	16,308	....	2,090	342	365	....	23
Green County	193	324	....	131	288	44	244	....
Morgan County	....	135	....	135	....	4	....	4
Missouri	243,545	233,087	12,714	2,356	37,446	34,233	3,240	27
Kansas	39,988	43,733	....	3,745	3,635	4,237	....	602
Oklahoma	3,156	3,242	....	86	661	669	....	8
District	286,589	280,062	6,527	....	41,742	39,139	2,603	....

Montana.—The zinc production of this State in 1907 was derived chiefly from Butte, where the La France company, a Heinze interest, has installed a mill of 100 tons daily capacity, using Sutton-Steele pneumatic tables and electrostatic separators. At the end of the year this mill was reported to be working successfully.

Nevada.—The production of zinc ore in this State was obtained chiefly from the mine at Potosi. The Monte Cristo mine, in the Good Springs district, Nye county, also made shipments of zinc ore.

New Mexico.—The Tri-Bullion company of Magdalena finally decided to build its smelter at Cañon City, Colo., instead of at Albuquerque, N. M., as first planned. Cañon City has moderate-priced fuel, is classed as a Colorado common point both for zinc ore and spelter, and is near the colony of skilled smeltermen who are employed in the works of the United States Zinc Company at Pueblo.

(By R. V. Smith.) The production of zinc ore was curtailed heavily in 1908 owing to the action of the larger mines at Magdalena. The shipments from one of the largest producers was continued until within the last few weeks of the year, but the marketing of ore from the property of the Tribullion company was stopped as soon as the lease expired and the property came back into the hands of the owner. This was on Jan. 1, 1907, and the year was spent in installing machinery and erecting a mill in order to extract more cheaply and prepare the ores for shipment to the new smelter of the company at Cañon City, Colo. Several companies within the Territory are, however, shipping zinc to Eastern points for treatment, and the character and grade of the ore is such that it is not at all likely that the shipments from Cook's Peak, and Hanover, as well as from Magdalena, will be interfered with by the lower prices. The mines show reserve orebodies larger than in 1906 and development work is being systematically carried on.

(By Chas. R. Keyes.) Two years ago the carbonate ores formed the chief shipping ores; now the sulphides are the more important. Magdalena continues to be the principal producing camp. Among other localities which are becoming important producers are Hanover and Pinos Altos, in Grant county, Tres Hermanos, in Luna county, and the Organ district in Dona Ana county. In a number of mountain ranges in southwestern New Mexico considerable deposits of zinc ore have been lately developed. The great need now is some means of handling these sulphide ores such as local mills centrally located. At the present time most of the New Mexican zinc ores are sent to Mineral Point, Wis., Iola, Gas, and Coffeyville, Kan., Pueblo, Colo., and to Belgium.

New York.—Nothing was done in 1907 at the zinc mine at Gouverneur, which was discovered two years ago. The property is still in litigation.

Tennessee.—The increased production of zinc ore in this State in 1907 is accounted for by the continued developments of the two principal companies operating in the Holston River district and the company operating at Straight Creek.

Utah.—The major part of the production of zinc ore in Utah in 1907 was contributed by the Scranton Mining and Milling Company of the North Tintic district, about 12 to 15 miles due north from Eureka. Its ore is a carbonate, averaging about 33 per cent. zinc. The Daly-Judge Mining Company entered into a contract with the Grasselli Chemical Company, whereby the latter is to install a plant for the separation of the zinc ore of the former.

Wisconsin (By J. E. Kennedy).—The production of zinc ore in Wisconsin-Illinois-Iowa in the first six months of 1907 was 24,954 tons, exceeding that of the corresponding period of 1906 by 8137 tons. The 20 new concentrating mills which went into commission during the early summer, together with those previously operating, should have resulted in a large gain for the second half of the year, but the slump in the ore market caused such a curtailment of output that the total shipments for the year were only 49,778 tons; however, this was 7649 tons in excess of the shipment in 1906. Fifty concentrating mills were erected during 1907, about one-half of them being single-jig 50-ton plants, and the others mostly 75- and 100-ton plants, with rougher, cleaner and sand jigs. Sludge tables are coming into more general use, the Wilfley being the favorite. Five roasting and magnetic separating plants were built during 1907, which makes 25 now in the district; of this number 20 are of the slow-roast, Galena type, which so far has been the only successful one in this field. The American Zinc, Lead and Smelting Company of Boston, controlling the Huff electrostatic separator, made plans to give that apparatus a trial in the Wisconsin district.

The Present Position of the Zinc Industry.

During the last few years, especially during 1906 and 1907, conditions have been developing in the zinc industry of the world, which are of supreme importance. This relates particularly to the enormous increase in the supply of ore offered to the smelters. New South Wales and Mexico, the former gradually and the latter suddenly, have developed into producers of zinc ore of the first magnitude. The exportation of zinc ore in New South Wales in 1907 amounted to 237,218 long tons, against 103,665 long tons in 1906 and 103,000 tons in 1905, but going behind the face of the returns, it appears that there was a considerable increase in 1906 because of the better grade of the ore produced, the estimated yield of spelter from the ore exported having been 33,427 tons in 1906 against 30,637 tons in 1905. American smelters received from Mexico 108,800 short tons of ore in 1907 against 88,900 in 1906, 32,164 in 1905, and practically none in 1904. Some Mexican ore also was exported to Europe. Besides these large new supplies, European smelters have been recently obtaining a good deal of ore from other new countries, such as Japan (where zinc smelting works are now being erected), China, and Turkey; while in 1906, a considerable supply of calamine ore of high grade was for the first time received from Rhodesia, where there appears to be large deposits, which will afford a steadily increasing output.

This plethora of raw material has already had an important effect upon the world's market for spelter. For one thing, it has cut off all hope that the United States will soon become an exporter of spelter. On the contrary, it is to be feared that, more frequently than usual, the price of spelter in the United States will have to be reduced in order to prevent importations, in spite of the protective tariff of 1.5c per lb. During the last three or four years the prices for spelter at London and New York have been showing an increasing disparity. This is easily explained by the position of the European smelters, who in the abundance of their ore supply are able on the one hand to offer spelter at lower and lower prices, and on the other hand bid lower and lower prices to the miners for ore, preserving a large margin for smelting all the time. During 1906 the smelters' margin was so large that the business was unusually profitable. The natural result of this, together with the refusal of the smelters to accept certain supplies of ore, was to lead several of the mining companies, together with other metallurgical companies, to go into the smelting business on their own account. During 1906-07 there was something like an epidemic in the construction of new zinc-smelting works in Europe. In Silesia and the west of Germany several large smelting works were under construction; one or two new smelters are being built in France; one in Spain; a large works has just gone into operation in Great

Britain, and another one is contemplated. A new works was lately completed in Australia, making two now in that Commonwealth, and a works is to be built in Japan.

Some of the new works that are being erected in Germany are so situated as to have much better facilities for the transportation of ore by canal and river from Rotterdam and Antwerp than any other works in Germany at present possess. Consequently, there is no doubt that some of these new works will capture a good deal of the business of smelting foreign ore. Naturally, the situation is viewed with alarm by the existing concerns. The increased competition will doubtless reduce smelting prices to a normal level, and the hope of great profits from the new enterprises, based on previously existing conditions, probably will fail of fulfilment for that reason. However, the effect on the American market will be obvious. With large supplies of cheap ore, and competition among the European smelters that materially reduces their margin for smelting, there is little likelihood that American smelters will have any opportunity to export spelter; in other words the conditions in Europe will tend to keep the London price steadily below the price in America. As regards the situation in Europe, looking toward the more distant future, the hope will be that the new works will produce no more than to satisfy the annual increase of the demand, while the old works will maintain their present production. This is the most optimistic view that can be taken.

The situation in the zinc industry outlined above was undoubtedly the inspiration for the convention among the German smelters recently organized to control production and regulate the price for the metal. According to the terms of this convention the product of German spelter is to be sold through the three great metal houses of Germany, namely, the Metallgesellschaft of Frankfurt am Main, Beer, Sondheimer & Co., and Aron Hirsch & Sohn. All of the zinc smelters of Germany entered this convention with the exception of Giesche's Erben, which remained outside as a matter of principle, although it is understood that its management will operate in harmony with the other smelters. It is planned to extend this convention so as to include all of the smelters of Belgium, France and Great Britain.

Zinc Mining in Foreign Countries.

Australia.—The Broken Hill district in 1907 produced 984 long tons of spelter against 1008 tons in 1906 and 544 tons in 1905, this being the output of the Sulphide Corporation at Cockle Creek. In addition thereto, the mines of Broken Hill exported 237,218 tons of concentrates against 103,665 tons in 1906, the Proprietary Company alone producing 66,595 tons of concentrates in 1907. Comparatively little progress was made

in 1907 by the Broken Hill Proprietary Company at its zinc smeltery at Port Pirie. In the course of the first trial the furnace proved unsatisfactory and the retorts were found to be defective. Steps have been taken to remedy the troubles and it is expected that the manufacture of spelter will be begun by the middle of 1908.

XVII. PRODUCTION OF ZINC IN NEW SOUTH WALES.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907
Spelter	286	299	544	1,008	984
Zinc in ore exported	14,625	22,318	30,637	33,427	76,645

Austria.—The Löbbeckeschen management added a new zinc smeltery, with modern furnaces and arrangements, to its plant at Niedzieliska, in Galicia, during 1907. To supply it with ore the calamine mines at Dlugoszyn were reopened.

Belgium.—The production of the Vieille Montagne Company in 1907 was 92,290 long tons of spelter. In 1906 its production was 86,880 metric tons of crude spelter, 6160 tons of special spelter and 3649 tons of manufactured zinc, a total of 96,689 tons. The rolling mills of the company made 67,253 tons of sheet zinc, while the oxide works made 4422 tons of zinc oxide. The work of the company at Baelen and Viviez produced 71,525 tons of sulphuric acid. These figures illustrate the magnitude of the business of this company, which is the largest zinc-smelting concern of the world.

Canada.—The production of zinc ore in this country in 1907 was small, owing to the inability of the producers to ship ore into the United States, because of the disinclination of American smelters to buy blende while the tariff question was still in the courts. The Canadian Metal Company made no attempt to operate the smeltery at Frank, Alberta, and probably never will do so. The Canada Zinc Company was engaged in the construction of a small smeltery at Nelson, B. C., to try the Snyder electrothermic smelting process in a more or less experimental way. It was reported that a smeltery would be erected at Kingston, Ont., to treat ore mined in Frontenac county, in the northern portion whereof several veins of zinc ore are said to have been discovered.

China.—Shipments of zinc ore from the port of Hankow amounted to 7248 tons in 1906 against 6600 in 1905. Most of this ore comes from the province of Hunan.

France.—The production of the zinc smelteries of France in 1907 was as follows: Viviez, 15,000 tons; Auby, 15,000; Mortagne, 8500; Noyelles-Godault, 7500; St. Amand, 3000; total, 49,000. The Mortagne works,

the newest in France, have six furnaces (240 retorts per furnace) and is said to be developing very successfully.

Germany.—The mania for new zinc smelteries attained the maximum in Germany. The works of the Metallhütte A. G., of Duisburg, controlled by the Metallgesellschaft Frankfurt am Main, went into operation early in 1907. That of the International Zinc Company, at Hamburg, began work in November. The new works near Blexersande, in the Grand Duchy of Oldenburg, will be completed in 1908.

The production of spelter in Upper Silesia in 1907 was 137,736 metric tons, against 135,970 metric tons in 1906. The same works produced in 1907 zinc dust to the amount of 3668 tons, lead 1190 tons and cadmium 32,949 kg., against 3277, 1293 and 27,561 respectively in 1906. The production of spelter in Rheinland and Westphälia was 69,160 tons in 1907, against 67,615 tons in 1906.

XVIII. GERMAN IMPORTS AND EXPORTS.
(In centners of 100 kg.)

	Imports.			Exports.		
	1905	1906	1907	1905	1906	1907
Spelter	268,406	370,359	284,591	623,233	633,947	622,379
Zinc sheets	544	808	1,191	189,817	172,979	214,759
Broken zinc	27,425	22,777	10,264	53,515	57,007	66,686
Zinc ore	1,265,773	1,790,360	1,847,026	389,727	426,055	348,632
Zinc dust (poussiere)	....	6,033	....	....	40,443	....
Oxide of zinc	....	52,310	70,492	....	141,057	187,633
Lithophone	9,073	15,104	22,080	77,467	79,947	94,951

XIX. AVERAGE PRICE OF SPELTER PER 1000 KG. IN SILESIA.

	1904	1905	1906	1907
First quarter	416 Mk.	468 Mk.	509 Mk.	511 Mk.
Second quarter	421	454	516	487
Third quarter	427	481	523	427
Fourth quarter	464	547	538	401
Average	432.00 Mk.	487.50 Mk.	521.50 Mk.	456.50 Mk.

Mexico.—In spite of the litigation respecting the importation of zinc ore into the United States, Mexico made greatly increased shipments in 1907, chiefly from Monterey and Chihuahua, but the industry received a severe setback in the fall when the price for ore declined to a low point. The ore receipts of American smelters from Mexico were 108,800 tons in 1907 against 88,900 tons in 1906.

Norway.—The owners of the Ranen mines have concluded an agreement with the Société Anonyme Métallurgique Procédés de Laval to work Dr. de Laval's patents for making zinc from low grade ore. A "Cyclon" furnace is to be erected at Mo in the Ranen district, and also

an electric furnace at a place where sufficient electric power (2500 to 3000 h.p.) is obtainable. At first about 50 tons of ore, expected to yield about eight tons of zinc and about one ton of lead will be treated per day.

Russia (By I. I. Rogovin).—Zinc is produced only in Poland, where the situation of the industry was not flourishing during 1907. A considerable part of the calamine deposits is worked out and the ore contains only a small percentage of metal, which along with the increase in operating costs and the scarcity of money has raised the cost of the metal. The Polish zinc works could only in a slight degree avail themselves of the recent favorable condition of the metal market, and now, owing to the decline therein, their situation is critical and it is only with the greatest effort that they can carry out their agreements with the government.

The amount of calamine extracted in 1907 was 3,441,037 poods against 3,806,685 in 1906. The stock of calamine on Jan. 1, 1908, was 847,407 poods. The average number of workmen was 1111; the average wage was one ruble. The three zinc smelteries of Poland produced 593,896 poods of spelter in 1907 against 586,205 poods in 1906. The stock on hand Jan. 1, 1908, amounted to 91,869 poods.

South Africa.—In the neighborhood of Zeerust, a new venture, the African Zinc and Galena Mines, Ltd., is opening deposits of galena and blende; samples and assays are reported to show that these deposits are rich.

Spain.—As heretofore, the Compagnie Royale Asturienne, at Arnao, was the only producer of spelter in 1907, but a new work is being built at Peñarroya (Cordoba) by the Sociedad Minera y Metalurgica de Peñarroya which will be completed before the end of 1908. Blende from the San Quintin mines will be treated along with that from other mines in the district. The works will have capacity for 5000 tons of blende per year, with a production of 1600 to 1700 tons of zinc.

Tunis.—While there has been no very notable increase in the shipments, the high price for zinc ore during the first nine months of 1907 yielded handsome profits to the mine owners such as Djebel Ressas, Khanguet, Ben Amar, and Royal Asturienne. The Ressas company purchased the mine of Koudiat Sidi, near Kef. The mine of Guern Alfaya, a very fine calamine mine, has passed into the hands of a Paris group. The lead and zinc mines of Ain Allega, purchased two years ago by the Mokta company, are said to be developing a large quantity of ore, which requires. however, special treatment.

Turkey.—The German Mining Company Speidel, which in 1903 began to prospect for zinc ore on the Isle of Thasos, was very soon in a position to obtain satisfactory results. The exports increased from 19,000 tons in 1905 to 21,000 tons in 1906. In 1907 work was started with an up-to-date ore-washing plant. Unfortunately, the output of the mines during

1907 was not as favorable as it had been up to then. The mine at Volgaro, having proved to be unpayable, had to be closed down, while the yield of the Castros and Sotiros mines considerably decreased. New prospecting works have lately been carried out at Morica and Theologos, and the results are reported to be favorable.

United Kingdom.—The new works of the Central Zinc Company at West Hartlepool, England, were well advanced in 1907 but did not go into operation until the spring of 1908. The concentrates from the Central mine, at Broken Hill, belonging to the Sulphide Corporation, are to be smelted there. Plant capable of treating 10,000 tons of zinc concentrate is being erected. The output of zinc should be about 80 tons a week. The land purchased for the works extends over 52 acres and cost £100 an acre. It is situated on the north bank of the mouth of the river Tees. The North Eastern Railway Company has built a branch line 10 miles long to connect the works with its railway system.

The Spelter Markets in 1907.

New York.—The year 1907 opened under very favorable auspices for the spelter industry. Smelters throughout the West operated at full capacity. Prices were at a level which permitted the mines in this country, as well as in Mexico and British Columbia, to furnish a full quota of ore at prices satisfactory alike to the miner and smelter. Nothing but the ordinary stock of metal existed and the spelter was shipped by the smelting works as fast as produced. Prices at the beginning of the year stood around 6½c., St. Louis. A heavy demand from galvanizers and brass manufacturers gradually forced the market higher until in the early part of March it reached about 6.75c., St. Louis. The available supplies by that time had become so scarce that spelter for immediate delivery commanded a premium. This was particularly the case with the higher grades of spelter used by the brass manufacturers, whose requirements had broken all previous records. During this wave of exceptional prosperity a number of the small coal smelters in the Kansas and Missouri districts, which had been idle for years past, were started up.

The March disturbances in the stock market took the edge off the brisk movement in the spelter market. The requirements of brass manufacturers, owing to the peculiar conditions which developed in the copper market, began to show a rapid falling off. As a result, larger quantities of spelter became available for galvanizing purposes, so that the pressure to sell gradually increased. Prices began to decline, at first slowly, but as the season advanced, the movement accumulated force and practically without reaction 4.90c. was reached about the middle of September.

High prices having ruled for such a long time, they had become more

or less a habit, and as an outgrowth of it, it was generally assumed that the Joplin field could not furnish ore at a parity of a spelter price lower than 5c., St. Louis. As a seeming confirmation of this, production fell off rapidly when the smelters tried to buy ores at the equivalent of the lower spelter price. Consumers of spelter, recognizing this situation, drew the conclusion that bottom had been reached, and a very large buying movement developed on the part of galvanizers throughout the country, fostered by the unabated activity in the iron and steel business, which quickly carried the market back to 5.40c., St. Louis, which price was reached toward the middle of October. This advance proved to be short-lived, and when the October panic upset the iron and steel business and the demand from that quarter became almost nothing, spelter again started on its downward course, and the decline did not stop until the price had fallen to 3.97½c., St. Louis, during the last few days of the year.

XX. AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900.....	4.65	4.64	4.60	4.73	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901.....	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.07
1902.....	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903.....	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904.....	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905.....	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882
1906.....	6.487	6.075	6.209	6.078	5.997	6.096	6.006	6.027	6.216	6.222	6.375	6.593	6.198
1907....	6.732	6.814	6.837	6.685	6.441	6.419	6.072	5.701	5.236	5.430	4.925	4.254	5.962

XXI. AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN ST. LOUIS.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1903.....	4.688	4.681	5.174	5.375	5.469	5.537	5.507	5.550	5.514	5.350	4.886	4.556	5.191
1904.....	4.673	4.717	4.841	5.038	4.853	4.596	4.723	4.716	4.896	5.033	5.363	5.720	4.931
1905.....	6.032	5.989	5.917	5.667	5.284	5.040	5.247	5.556	5.737	5.934	5.984	6.374	5.730
1906.....	6.337	5.924	6.056	5.931	5.846	5.948	5.856	5.878	6.056	6.070	6.225	6.443	6.048
1907....	6.582	6.664	6.687	6.535	6.291	6.269	5.922	5.551	5.086	5.280	4.775	4.104	5.812

London.—Opening at £28 2s. 6d. per ton for ordinary brands ex ship in the Thames, the price quickly fell to £27 15s., a few bear sales helping the depression. Consumption, however, remained on a large scale, with every prospect of continuing so. Moreover it was known that Continental producers had already sold at relatively high prices for delivery in the second quarter of the year; but these favorable indications did not serve to arrest further decline down to £26 12s. 6d. on Jan. 18. At this point a rally ensued and the price rapidly rose to £27 7s. 6d. after which it fell back to £26 17s. 6d. for ordinary brands, and £27 7s. 6d. for Silesian specials. In February speculative realizations gradually forced the price

of ordinary brands down to £25 15s. Closing prices were £26 1s. 3d. for ordinary brands, and £26 2s. 6d. to £26 7s. 6d. for specials. March opened with moderate but increasing demand, particularly for export, and with prices still ruling higher on the Continent than in London. About Feb. 26 prices relapsed seriously, on account of forced sales for future delivery, and fell to £25 17s. 6d.; special brands stood at the relatively high price of £26 10s.

April opened with a dull market which, however, soon revived. On April 10 ordinary brands commanded from £26 2s. 6d.@£26 5s. Closing values were £26@£26 2s. 6d. for ordinary brands, and £26 10s. for specials. May opened with active business. Galvanizers bought largely down to the middle of the month, but were freely met by dealers. When this demand was satisfied, a persistent decline finished at £24 17s. 6d. for ordinary brands, and £25 17s. 6d. for specials. June showed no improvement.

The close was at £24 5s.@£24 10s. for ordinary brands, and £25 12s. 6d. @£25 17s. 6d. for specials.

July showed a persistent decline, due to abstention of consumers and eagerness of holders to sell. Closing prices were £23 2s. 6d. for ordinary brands, and £24 5s. for specials. August was dull and the market was demoralized by the apathy of consumers, no less than by pressure of sales on the London Exchange by parties operating for the fall. The middle of the month saw a temporary rally, but this was of short duration, and the decline thereafter continued to the end of the month when ordinary brands were quoted £21 12s. 6d.@£21 15s., and specials £22 5s.@£22 12s. 6d. September opened with general uneasiness and depression. Consumers bought reluctantly and only for early delivery. The month closed with ordinary brands at £20 17s. 6d., and specials at £21 10s.@£22.

October opened with a steadier market, which induced an improved volume of business. Values improved until Oct. 15, when the general depression in commerce prompted some bear selling for forward delivery and reduced values by 5s. It was evident, however, that prompt supplies were scarce, and producers well sold and holding for full prices. Zinc rollers were busy, and an improvement in the galvanized iron trade stimulated an advance to £22 2s. 6d.; special brands, £22 12s. 6d. In November, spelter was less affected than other metals. Advancing rate of interest on money in Germany induced sellers there to part with their holdings. The result was a decline in price. Prices at the close were £21 5s. for early delivery, and £20 15s. for forward; special brands being quoted at £22 and £21 10s. December opened with a dull market, the consuming industries being sufficiently supplied with metal, and the prevailing depression in commerce and finance being sufficient to check enterprise. Prices varied, with fully £1 margin between spot and forward

metal. The close was at £19 10s. for good ordinaries, and £19 15s. for special brands.

XXII. AVERAGE MONTHLY PRICE OF SPELTER IN LONDON.
(Pounds sterling per ton of 2240 lb. of good ordinary brands.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£	£	£	£	£	£	£	£	£	£	£	£	£
1905.....	25.063	24.594	23.825	23.813	23.594	23.875	23.938	24.675	26.375	28.225	28.500	28.719	25.433
1906.....	28.225	25.844	24.563	25.781	27.000	27.728	26.800	26.938	27.563	28.075	27.787	27.938	27.020
1907....	27.125	25.938	26.094	25.900	25.563	25.469	23.850	21.969	21.050	21.781	21.438	20.075	23.771

Progress in the Metallurgy of Zinc.

A good deal of experimental work was done in 1907 in connection with new processes for the separation of mixed sulphide ores and for smelting, without, however, indicating the probability of any radical innovation over existing processes. Undoubtedly the most practical importance is to be attributed to the development of the new methods for separating mixed sulphide ores, since it is through them that the great additions to the supply of zinc ore in the future are to come. Already they have reached the stage at which they are having a weighty effect upon the market. This refers especially to the Australian ore, the separation of mixed sulphides having so far attained its greatest practical development at Broken Hill.

The Broken Hill Proprietary Company assigned to Potter's Sulphide Ore Treatment Company all of the patents it held in the name of F. M. Dickenson and G. D. Delprat, including those relating to the Delprat process. In future the Proprietary Company will work under license from Potter's Sulphide Ore Treatment Company, but will pay no royalty, the consideration for the right to use the process being £10,000 paid at the settlement of the litigation.

In the operation of the flotation process by the Broken Hill Proprietary Company it has been found that superior results are obtained from the use of a deep vat, and consequently the old shallow vats have been replaced by new ones 16½ ft. deep, not lead lined. The Cattermole, Ballot, and the older magnetic separating processes are used at Broken Hill to a more or less extent. The Zinc Corporation after a trial of the Potter process, and subsequently the Cattermole process, from neither of which were satisfactory results obtained, adopted the Elmore vacuum process. The plant for the application of the Elmore process went into operation early in 1908, it is reported with strong promise of success. In the United States the Elmore vacuum process was tested experimentally at the works of the Empire Zinc Company at Cañon City, Colorado, and also at other places.

Further work was done during 1908 with the Sanders flotation process,

in connection with which several interesting new features were developed. Experiments were made with the ore produced at Straight Creek, Tennessee, which was found to give much difficulty, owing to the intimate mixture of dolomite with the blende; this proved to be an obstacle to flotation apparently impossible to overcome. On the other hand, very successful results were achieved in the separation of the mixed sulphide ore of the Kelly mine at Magdalena, N. M., and the Tri-Bullion company was engaged in the installation of a large plant for the application of the process. This plant will go into operation about July 1, 1908. An interesting new flotation process was introduced by A. P. S. Macquisten at Golconda, Nevada, which was described in the *Engineering and Mining Journal* of Oct. 26, 1907. At Golconda, this process is employed for the concentration of a copper ore, but experiments show the principle also to be applicable to the separation of blende.

In zinc smelting, much interest was displayed in the development of electric furnaces, the leading work in this direction being done by Snyder, Johnson and De Laval. So far De Laval is the only one who is actually producing zinc by electrothermic smelting; he has the advantage of exceptionally cheap power. However, we are still a long way off from the substitution of the electric furnace for the ordinary furnace. Further experimental work with the Imbert process of decomposing zinc sulphide by means of metallic iron or copper was done in Europe.

The following paragraphs are not intended to be a thorough review of the metallurgy of zinc in 1907, but record simply some of my personal observations during the year.

The new works of 1907, especially two of those at Bartlesville, show some interesting modifications of previous practice. In most cases the distillation furnaces are Iola furnaces, with only four rows of retorts. The merits of the furnace with only four rows of retorts as against the older furnaces of five retorts in hight now seem to be well established. The National Zinc Company has four furnaces, each of 608 retorts. The Bartlesville Zinc Company has six furnaces, each of 576 retorts. For roasting the sulphide ore both works have Zellweger furnaces, of which the National Zinc Company has two and the Bartlesville Zinc Company three. This furnace is now well established as the favorite for blende roasting in Kansas and Oklahoma, in spite of its enormous consumption of fuel.

Bartlesville Zinc Company.—This company has a well-equipped crushing and sampling mill, with a large cylindrical drier of good design and construction, which is commendably provided with a dust chamber. The crushed ore is received in elevated storage bins of 4000 tons capacity in the aggregate. From these bins the ore goes to the roasting furnaces. From the latter a sunken tramway brings the roasted ore to a platform elevator, by which the cars are raised to be discharged into a series of

elevated steel bins. From the latter the roasted ore is drawn into cars which convey it to the mixing house where the mixture with reduction material is effected by a Vapart mill. The Vapart mill discharges upon a conveyer which delivers into large cars to go to the distillation furnaces. A tramway extends in front of the inner ends of the furnace houses, and connection is made with the tracks running down in front of the furnaces by means of the ordinary transfer car. The charge cars are of wood, and extraordinarily large, there being only two per furnace, one for each side, which makes them rather unwieldy. Nothing is gained by so large a charge car.

The pottery is decidedly superior to the ordinary constructions in Kansas and Oklahoma. In the mill, the clay is crushed by means of a dry-pan. The clay is mixed in a taper-tub pug-mill and the retorts are formed by a Mehler hydraulic press. The condensers are made by a Vanatta & Stafford machine, which is now generally in use in Kansas and Oklahoma. The store rooms for the retorts are comprised in a two-story building of standard slow-burning mill construction, except for the roof. There are separate rooms, heated by steam, for the retorts, with an aggregate capacity for 10,000.

The pottery and crushing and sampling mills are driven directly by shafting from the engine-room. The latter contains electric generators, which supply electric power for use elsewhere in the works. The air for the distillation furnaces is supplied by two large fans, with a main leading to the furnaces in the usual manner. The draft for the roasting furnaces is provided by two Custodis chimneys, 120 ft. in hight. The general layout of the works is good and shows a marked advance over that of many of the older Western smelteries; if any criticism is to be made, it might be said that the plant is rather too compact.

National Zinc Company.—The works of the National Zinc Company, which has been designed by Otto Rissmann, is decidedly original in many features. The storage for ore and coal, the crushing mill, roasting furnaces, and mixing department are all contained in one large shed, of light steel construction. The crushing machinery is of the usual Kansas style. For drying the ore, there are two circular, plate driers, heated from below, upon which the ore is moved by a revolving stirrer, these driers being identical with those at the works of the United Zinc and Chemical Company at Iola. There are two Zellweger roasting furnaces. The hot ore delivered by the latter is elevated into two brick bins, one at the end of each furnace. Crushed calamine and reduction material are received in wooden bins nearby. The components of the charge are drawn from these bins and dumped into a Ransome mixer, which stands in a pit below the floor level. The mixed charge is delivered into a tub which is elevated from the pit by a traveling electric hoist, and by the latter is conveyed upon an overhead

runway, extending by the ends of the distillation furnace houses, to the charge cars which are pushed out under this runway.

The charge is dropped into the charge cars, and the latter when filled are ready to be pushed down the tracks in front of the furnaces. This is a very simple system, which eliminates the construction of heavy trestles, the use of a transfer car, curves and switches, and other annoyances. It is a straight run from the mixer to the last furnace, and except for the overhead track, the only tramways in the works are the short pieces of track in front of each furnace, the track from the pottery, and the track for taking away the spelter; on the last a gasolene locomotive is to be used.

Another novel feature of these works is the arrangement of the blowers for the distillation furnaces. Instead of a large fan, with a long main from the engine room to the furnaces, each furnace has its own blower, or rather its pair of blowers because the installation is made in duplicate, one to run by day and the other by night, placed in a shed at the end of each furnace house. These blowers, which are of Sturtevant manufacture, are driven by direct-connected electric motors. This system of forced draft is decidedly superior to the older system; if forced draft is to be used at all, this is the way it should be done; as to whether there is any advantage in forced draft over natural draft is another matter.

A further novelty in the works of the National Zinc Company is the design of the pottery. This building is entirely upon one floor, in fact the brick for the latter is laid directly on the ground. In the press-room it is necessary to have a pit for the press to stand in, but this pit is small and not particularly objectionable. The hydraulic pump and all the other apparatus are on the same floor with the top of the press. From the press, the molded retorts are taken to the drying rooms, which are a series of brick vaults with arched roofs. Each vault is 16 ft. wide and 65 ft. long in the clear, and holds 1000 retorts. The installation comprises 12 vaults. A wooden grating is laid on the main floor, the steam pipes extending under this grating. Down the center of each vault there is a track. The retorts are conveyed down an alley in front of the vaults on a transfer car, from which they are readily trammed into any of the vaults as required. This arrangement requires a good deal more ground space than the ordinary store house of two or three stories, but it is convenient, and if properly designed it is cheaper to carry a large weight per square foot, such as occurs in this kind of storage, on the ground than upon elevated floors.

Smelting Practice at Iola.—At Iola several interesting experiments in details of the practice have been made. The United Zinc and Chemical Company is using prolongs on the condensers up to the first draw of metal and finds the additional recovery of metal effected thereby to be satisfactory. Prolongs were used in Kansas many years ago, but were abandoned as unprofitable. It is to be remarked, however, that now in smelt-

ing the Western ores, high in iron, it is found to be the best practice to bring the retorts quickly to a high temperature instead of slowly as in the case of the more docile Joplin ore. This, of course, is a good reason why it may be worth while to employ prolongs now where formerly it was not. At certain works, Leadville ore assaying 40 per cent. zinc, 17 per cent. iron and 4 per cent. lead is received, and after roasting is smelted without admixture of any other ore. Of course, this means about 20 per cent. of iron in the roasted ore. In some cases mixtures of ores are made to reduce the iron content of the average, but in other cases ores are smelted as they come, and we find certain works with every furnace on a different charge. An interesting experience was noted in an attempt to smelt a peculiar ore from Aspen. This ore contained 10 per cent. of lime, 5 per cent. of baryta, and 2 per cent. of magnesia. All attempts to work it failed. It was found that the zinc content of the ore would reduce satisfactorily, but the vapor burned away as oxide, which was perhaps the result of too much carbon dioxide in the gas and too strong a development of gas.

Smelting Western Ores.—In smelting Western ores containing about 12 per cent. of iron, the daily loss of retorts is about 4 per cent. Ore assaying 40 per cent. zinc yields 75 to 80 per cent. of its metal. Nearly all the works now have hydraulic presses for the manufacture of retorts, and condensers are generally made with the aid of the Vanatta & Stafford machine, which is simple, efficient and economical. This machine costs $1000 including royalty. With it one man and helper make 1000 to 1100 condensers per day at a contract price of $0.60 per 100, against the former price of $1 per 100 by hand work.

The retorts used at a certain works are $\frac{15}{16}$ to $1\frac{1}{8}$ in. in thickness of wall and 8 to $8\frac{1}{2}$ in. in internal diameter, the larger figures in each case arising from the gradual wear of the die of the press. The Queneau composite retorts are still being tried at one of the works of the Prime Western Spelter Company. They are said to show a little longer life than the ordinary retorts, but to be of doubtful advantage. It seems to be impossible by this system of manufacture to give the retorts an even lining, and difference in the coefficient of expansion of the lining and the outer portion of the retort tends to crack them. At the best the benefit of their increased durability appears to be insufficient to outweigh their increased cost, at least not to any important extent.

Experiments with the Greenawalt porous-hearth roaster at the Prime Western works proved a failure and the furnace has been remodeled so as to run as an ordinary straight-line mechanical reverberatory. The difficulty was in maintaining an even passage of air through the hearth, the latter becoming clogged by ore that had any particular tendency to sinter.

Several of the works at Iola which use Western ore pay a good deal of attention to the residues, which are shipped back to Colorado to the lead

smelters and constitute an important by-product. At one works the surplus of reduction material is simply screened out. At another works the residues are run over a Hancock jig, which takes out the surplus of reduction material. The latter is used again in making up the furnace charge, for which purpose it is found to be sufficiently clean. As a reduction material Spadra semi-anthracite is commonly employed. The proportion is about 50 to 55 per cent. of the weight of the ore, but in one case is as low as 40 per cent.

Consumption of Natural Gas.—With the increased cost of gas, more attention is paid to economy in its use. In several cases the consumption has been metered for test. The figures for an ordinary block range from 630,000 to 750,000 cu.ft. per day. In round numbers this is 40,000 to 50,000 cu.ft. per ton of raw sulphide ore, or 48,000 to 60,000 cu.ft. per ton of roasted ore or calamine. I do not know of any measurements in the case of the roasting furnaces commonly used. In the case of a well designed roasting furnace the consumption was about 4500 cu.ft. per ton of raw ore, which is a reasonable figure, but the Zellweger furnaces are very extravagant of gas.[1] Probably they consume 30,000 to 40,000 cu.ft. per ton of raw ore.

This is an important matter now that gas is becoming more and more costly. According to the report of the U. S. Geological Survey, the average value of the gas consumed by zinc smelters in Kansas, in 1906, was 1.8c. per 1000 cu.ft., the cost ranging from 1 to 3c. per 1000 cu.ft. These figures appear to be nearly correct. They are none too high.

Smelters in Illinois.—Among the Eastern works, those of the Illinois Zinc Company, at Peru, Ill., have recently been reconstructed to a large extent, especially with a view to affording more economical handling of material, better furnace design, and increased safety against fire. All of the distillation furnaces at these works are now of the Neureuther-Siemens design. The gas is furnished by Swindell producers. The old Taylor producers have been modified into approximately the Swindell form. With the badly clinkering coal of this district, the producer with a sloping grate, which gives access to the bed of fuel from below, affords better results than the water-sealed producer without a grate. The roasting furnaces are

---

[1] My attention has been called to this statement, which appeared originally in an article by me previously published in the *Engineering and Mining Journal*. A. B. Cockerill wrote to me under date of June 21, 1908, as follows: "It has until recently been the impression of everyone in the zinc business, including myself, that the Zellweger furnace consumes a great deal more gas than is consumed by the Ropp or Brown furnaces, but this is a mistake. C. H. Armstrong, who was for years connected with the Lanyon Zinc Company and metered the gas at its works, informed me that the Ropp furnace used approximately 500,000 cu.ft. of gas every 24 hours. This led me to meter the gas consumption of the Zellweger furnace at Altoona and I found that the quantity used during 20 days was from 491,000 to 513,000 cu.ft. per day, the average being a little less than 500,000 cu.ft. The quantity of roasted ore drawn during this period ran from 37,000 to 43,000 ℔s., the average being about 40,000 ℔s." I have only to call attention to the fact that I made no comparison between the Zellweger and the Ropp or Brown furnaces. As commonly installed in Kansas and Oklahoma, I have no doubt that the three furnaces are substantially on an equal footing, as stated by Mr. Cockerill, but nevertheless all are extravagant of gas. This is because of the enormous excess of air that is allowed to pass through the furnaces. The figures given by Mr. Cockerill for the Zellweger furnace at Altoona correspond to about 21,000 cu.ft. of gas per ton of raw ore, which is less than I estimated. However, I fancy that the gas consumption of these furnaces varies largely among different works. W. R. I.

Hegelers with regenerative gas firing. Mr. Noon has done excellent work in remodeling this old plant, which ranks now among the most efficient of American zinc-smelting works.

At the works of Matthiessen & Hegeler, at Lasalle, there has been but little change during several years. The furnaces which were formerly five rows of retorts in hight have been increased to six rows in hight. Consequently the tendency at these works and at the various works in Kansas has been directly opposite. The large furnace previously noted, which had 1008 retorts arranged in 21 sections, has been cut down to 18 sections, or 864 retorts. It is said that beyond the latter number the limit of capacity of the furnace crew was exceeded. The furnace equipped with a hot-gas blower between the battery of producers and the furnace is sometimes worked with the blower, sometimes without it; it is said that there is no particular advantage in the blower, the furnace running equally well when the latter is cut out for repairs, etc. The producers are blown with air only.

The works of the Mineral Point Zinc Company, at Depue, Ill., have six Neureuther-Siemens furnaces, arranged end to end, three in a house. Each furnace has 800 retorts. The working floor of the furnaces is elevated above the ground in the modern way, but in this case the hight is extraordinary. The gas is furnished by Duff producers, which stand in a house between the two furnace houses. The roasting furnaces are Hegelers, two in number, both gas-fired, one of them regeneratively. Sulphuric acid is made from the roast-gas by the Grillo-Schroeder contact process. The Depue plant is one of magnificent distances, in this respect surpassing even Palmerton.

The three works at Lasalle, Peru, and Depue, are fine object lessons in respect to the sulphur smoke problem. Each of them converts all of its sulphurous gas into acid, while the use of gas-fired furnaces largely eliminates the nuisance of black coal smoke. In close proximity to the Matthiessen & Hegeler works, particularly, there are fine lawns and gardens, and magnificent shade trees, which flourish as if there were no smelting within 100 miles.

The Valuation of Western Zinc Ore.

By W. G. Martin.

In view of the fact that the production of zinc ore has become in recent years so important to mine operators in the western States, it is instructive to compare different schedules used in the payment for both low-grade and high-grade zinc ores.

For ores of a smelting grade, I have given three schedules, Nos. 1, 2 and 3, and have plotted them so as to show how the value of the ore varies

as the price of spelter varies, and also to show how the value varies for different grades of ore. These schedules are for ore delivered at the smelting point (Kansas common points) and may be accepted as prices actually in use. It is understood that these prices may be cut should an ore be objectionable because of the presence of lime, a high percentage of iron or silica, or should its physical condition be such as to cause an abnormal loss in smelting. On the other hand a smelting company may pay slightly more for a very desirable ore.

Referring to Fig. 1 and the line representing schedule No. 1, it will be seen that there is a regular variation in the value of ore of different percentages of zinc as the value of spelter varies. This is due to the variation of 6c. per ton of ore of all grades for each variation of 1c. per 100 ℔. in the price of spelter. It will be observed that this regularity is not true in regard to schedule No. 2. In this quotation, for 45 per cent. zinc ore, the variation per ton of ore amounts to 6c. for each variation of 1c. in spelter price and therefore coincides with schedule No. 3 for all values of spelter. On 40 per cent. zinc ore, however, the variation amounts to only 5c. for each change of 1c. in spelter price, and for 35 per cent. zinc ore to only 4c. per ton of ore. Considering schedule No. 1, which is of the form used abroad for the settlement of zinc ore, we find the variation per ton of ore amounts to slightly more than 7c. for each variation of 1c. in spelter for 45 per cent. zinc ore, slightly over 6c. for 40 per cent. zinc and slightly over 5c. for 35 per cent. zinc. Therefore for 40 per cent. zinc ore schedules Nos. 1 and 3 very nearly coincide and are plotted as being identical.

It will be readily seen from Fig. 1 that when spelter is below $6 per 100 ℔., ore carrying up to 45 per cent. zinc will net the mine a better price on schedule No. 2 than on Nos. 1 or 3, while with spelter above $6, schedule No. 3 gives higher returns. Also, when the zinc tenor is above 45 per cent. the mine will receive a higher price if schedule No. 3 is used as a basis of settlement.

To find the value of ore of other percentages of zinc than shown in the diagram between the limits of 30 and 45 per cent. zinc for any specific quotation of spelter, the difference may be interpolated from the two nearest percentages of zinc shown to within 10 or 15c. per ton of the correct value. On quotations of a similar form but of a different base price from the above, the diagram may be used by determining the net value of the ore and adding or subtracting the difference in the base price.

In the sale of low-grade Western ores for milling purposes the schedules are quite different. This is natural, for while the milling companies produce a zinc concentrate that usually meets with a ready sale at prices approximately as shown in the diagram, the loss in milling is heavy.

VALUE OF ZINC ORES.

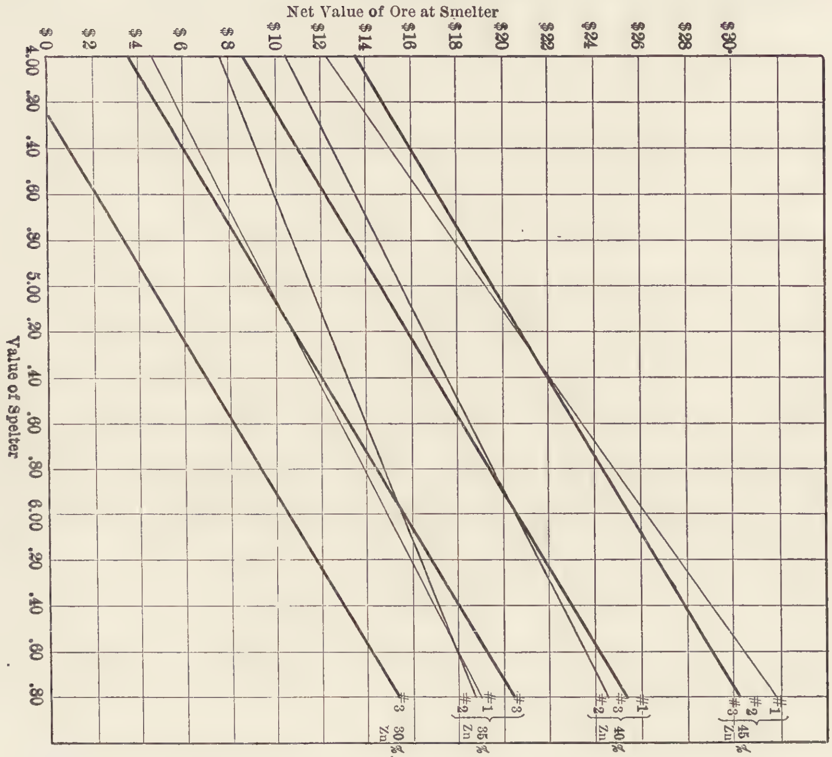


Fig. 1.

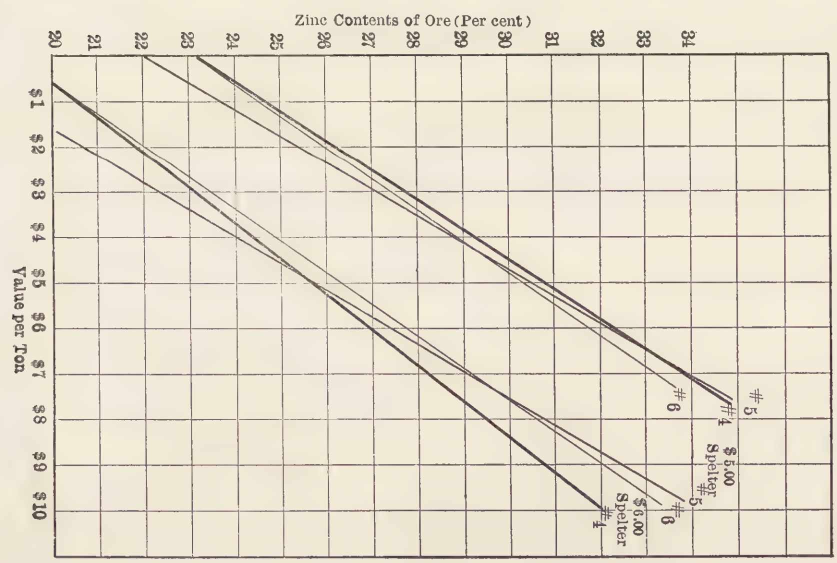


Fig. 2.

Schedules Nos. 4, 5 and 6 are used in the sale of low-grade milling ores and are plotted on Fig. 2, showing how the price varies as the grade of ore varies, as the price of spelter varies, and, also, how the different schedules differ on ores of the same zinc tenor.

It will be seen that ore sold on schedule No. 5, basis of \$6 per 100 ℔. for spelter, will bring a better price up to 25.5 per cent. zinc than schedule No. 4, and up to 30 per cent. zinc will bring a better price than schedule No. 6, beyond which points schedules No. 4 and 6, respectively, net higher returns. However on \$5 spelter, schedule No. 4 gives higher returns than No. 5 only after the grade of the ore reaches 33 per cent. Schedule No. 5 gives increased returns over schedule No. 6 on ore containing above 30 per cent. zinc and smaller returns on ore carrying less than 30 per cent. zinc regardless of the price of spelter.

Schedules for Calculating the Value of Zinc Ores.

No. 1.—Value of ore per ton equals $[0.95\ P(T—8) \div 100]—R$, in which P is the value of spelter per ton; T is the per cent. of zinc in the ore; R is the returning charge per ton, here used as \$15.98 for all grades of ore carrying from 35 to 45 per cent. zinc.

No. 2.—Value of ore per ton is \$20.50 for 40 per cent. zinc, basis of \$6 spelter, \$1 per unit of zinc up or down, variations as follows as price of spelter per ton varies: For ore with 35-36 per cent. zinc, add or deduct 20 per cent. of the variation in value of spelter; for ore of other percentages add or deduct the variation given in brackets, as follows: 36-37 (21); 37-38 (22); 38-39 (23); 39-40 (24); 40-41 (25); 41-42 (26); 42-43 (27); 43-44 (28); 44-45 (29); 45-46 (30).

No. 3.—Value per ton of ore, \$15.50 for 35 per cent. zinc, basis \$6 spelter with variation of 6c. per ton up or down for each variation of 1c. market quotation per 100 ℔. spelter, and \$1 per unit up or down for each variation of one per cent. in zinc contents.

No. 4.—Value of ore per ton equals $(13 \times T \times P)—R$, in which T is the units zinc in ore; P is the St. Louis quotation for spelter per pound; R is the returning charge per ton, here used as \$15 for all grades.

No. 5.—Value per ton, \$7.60, basis \$6 spelter for 30 per cent. zinc with variations of 60c. per unit zinc up or down, also variations as follows as price of spelter ton varies: For ore with 26-27 per cent. zinc, add or deduct 10 per cent. of the variation in value of spelter; for other percentages add or deduct the variation given in brackets as follows: 27-28 (11); 28-29 (12); 29-30 (13); 30-31 (14); 31-32 (15); 32-33 (16).

No. 6.—Value per ton, the same as No. 5, except that the ore is paid for with variations of 70c. per unit zinc up or down from a base of 30 per cent., instead of 60 cents.

A REVIEW OF THE LITERATURE ON ORE DEPOSITS IN 1907.

By J. F. Kemp.

Of all the metals, copper has been in later years the most suggestive in connection with the reactions of vein formation and deposition. More than is true of any other, copper depends upon secondary enrichment to bring its mines within the margin of profit, and to an extent scarcely equalled by the rest, its orginal sulphides are subject to solution and reprecipitation. The ordinary reactions seem now to be fairly well understood, but during the year 1907 several additional points of interest have been developed by the chemists.

To establish the connection it will be necessary to refer to earlier work. Thus in 1903, in the *Zeits. f. prak. Geol.*, p. 49, Dr. Ernst Kohler developed the idea of "adsorption." Adsorption was a property believed to be possessed by clay, whereby it was able to extract copper from solutions filtering through it, and to retain the compounds of this metal, not by chemical combination but by adhesion, thus practically absorbing them. W. H. Weed brought the idea especially before American readers, by commenting upon its apparent applicability to Arizona cases;[1] and at his suggestion experiments were begun in the laboratories of the United States Geological Survey in Washington. Several papers of prime importance have resulted. In Bulletin 312 of the Survey, issued in 1907 and entitled, "The Interaction between Minerals and Water Solutions, with special reference to Geologic Phenomena," Dr. E. C. Sullivan reviews the entire subject. Compounds of the alkalies such as are of great importance in the chemistry of the soils are necessarily considered, as well as the characteristic metals. Obviously the colloid conditions of matter, dialysis, mass action and similar phenomena must be also discussed, but in the end, after a long series of very careful quantitative experiments, and especially as affecting copper, the following conclusions are reached, p. 64: "The natural silicates precipitate the metals from solutions of salts, while at the same time the bases of the silicates are dissolved in quantities nearly equivalent to the precipitated metals. The bases most commonly replacing the metals in the processes are potassium, sodium, magnesium and calcium. Where exact equivalence is wanting,

[1] "Adsorption in Ore Deposition," *Eng. and Min. Journ.*, Feb. 23, 1905, p. 364.

it is attributable either to solubility of the mineral in pure water or to the precipitation of basic salts. The metals are precipitated as hydroxides or basic salts (in the case of cupric sulphide, for instance, as a basic cupric sulphate similar to brochantite or langite[1]) with more or less metal silicate."

Dr. Sullivan exposed cupric sulphate solutions to albite, amphibole, augite, biotite, enstatite, garnet, clay-gouge, kaolin, microcline, muscovite, olivine, orthoclase, prehnite, shale, talc, tourmaline and vesuvianite. He gave also particular attention to the behavior of orthoclase when treated with solutions containing salts of sodium, potassium, magnesium, calcium, strontium, barium, manganese, iron, nickel, copper, zinc, silver, gold and lead. The effect of kaolin on solutions of zinc and iron; that of glass, fluorite and pyrite on cupric sulphate, and that of carbonic and sulphuric acids on orthoclase, were all investigated.

Adsorption is doubtful, but precipitation by still undecomposed feldspar, mixed in with the kaolin, is certain. As a precipitant of copper, orthoclase does not differ materially from pyrites. Orthoclase, and presumably other alkaline silicates as well, accelerate the oxidation of ferrous sulphate by the atmospheric oxygen. Strangely enough when copper is precipitated in the presence of ferrous silicates, or of iron-bearing minerals in a clay gouge, their iron does not go into solution in consequence.

Dr. Sullivan's work clears up our thinking along not a few lines in the complicated processes of replacement. Somewhat against our natural disposition we see in orthoclase and other feldspars precipitants, while the ferrous silicates do not behave exactly as we had suspected. Indeed no observer of impregnation phenomena, of secondary enrichment and of the replacement of silicates should fail to interpret what he sees in Nature in the light of these important researches.

When the precipitation of copper is under discussion, it is impossible to keep from the background of one's mind the native metals of Keweenaw Point and the elusive problem of its precipitation. In earlier years both electrolytic reactions and the reducing effect produced by the oxidation of ferrous compounds to the ferric state have been appealed to, but have never seemed entirely satisfactory. The recent investigations of Dr. A. C. Lane in determining the composition of the waters found in occasional pockets in the deep mines, have shown that they are quite concentrated solutions of calcium and sodium chlorides. This fact raised the question as to the precipitating effect of ferrous chloride upon copper chloride. H. N. Stokes had previously shown that ferrous sulphate possessed this ability. At Dr. Lane's suggestion, an investigation of the reactions of the mixed chlorides was undertaken by Gustave Fernekes in the Michigan College of Mines, and it was found that with the chlorides

[1] Brochantite varies from $CuSO_4$, $\frac{5}{2}$ CuO, $3H_2O$, to $CuSO_4$, $\frac{7}{2}$ CuO aq. Langite is $CuSO_4$, $3\,Cu(OH)_2+H_2O$.

alone, no precipitation ensued. Mr. Fernekes very acutely reasoned, however, that in the reaction free hydrochloric acid necessarily formed, and being a solvent of metallic copper, prevented its precipitation. He was therefore led to conduct the reaction in the presence of various calcium-bearing minerals, characteristic of the orebodies, such as labradorite, prehnite, laumontite, datolite and pectolite and also with the sodium-silicate, analcite. In the cases of prehnite and datolite copper was precipitated in shining particles, but with the others nothing decisive resulted. In the orebodies, datolite is closely associated with the deposition of the copper, but prehnite preceded it. Had copper chloride encountered ferrous chloride in the presence of prehnite, precipitation of copper would have ensued.[1]

The precipitation of the minerals in orebodies has also been attacked from the point of view of physical conditions. Some years ago Professor F. Becke,[2] of Vienna, developed the very interesting and important idea that those minerals that show an increase in density over the specific gravities of their component molecules, or in other words a diminution in molecular volume, are characteristic of the deep-seated zones of the earth; whereas those that exhibit a decrease in density, or an increase in molecular volume, belong in the upper zones. Thus, in the metamorphic rocks we have three minerals, andalusite, cyanite and sillimanite, all having the same composition, $Al_2O_3.SiO_2$, or a combination of the corundum and quartz molecules. The specific gravities of the three silicates are respectively: Andalusite, 3.16; sillimanite, 3.24; and cyanite, 3.66. When compared with those of corundum, 4.0, and quartz 2.65, we find that the two lighter ones expand in molecular volume while the heaviest one contracts, or is a more characteristically deep-seated mineral.

U. Grubenmann,[3] a former associate of Professor Becke's, has used this idea in a scheme of classification and description of the metamorphic rocks in an extremely suggestive and valuable little book. Grubenmann concludes that minerals have their characteristic "critical levels," below which they resist alteration, but above which they fall an easy prey to change. Some are "persistent" and endure despite contrasted physical conditions. In the great treatise on "Metamorphism," President Van Hise also develops the relations of minerals to depth, although in a different way from the one just outlined.

With reference to ore deposits, W. Lindgren[4] has applied these conceptions in a very important and suggestive contribution in which he classifies the minerals, found in ore and gangue, into groups, characteristic of the zones in which they form. In this grouping it is essential first to

---

[1] G. Fernekes. "Precipitation of Copper from Chloride Solutions by Means of Ferrous Chloride." *Economic Geology*, II, 580, 1907.
[2] "Ueber Mineralbestand und Structur der krist. Schiefer." *Sitzungsber.* Wiener Akad., May 7, 1903.
[3] "Die krystallinen Schiefer." Berlin, 1904.
[4] "The Relation of Ore-deposition to Physical Conditions." *Economic Geology*, II, 105, 1907.

establish a division of "Persistent Minerals," which, although formed in the depths, may still hold their own when brought by erosion or uplift into higher or even surface levels. We may then have the following:

A. Minerals of Contact Metamorphic Deposits.
B. Minerals of Deeper Vein Zones.
C. Minerals of Middle or Upper Vein Zones.
D. Minerals unstable in all Vein Zones.
E. Minerals formed in the Lower Ground-water Zone of Ore deposits (Zone of Secondary Sulphide Enrichment.)
F. Minerals formed in Oxidizing Ground-water Zones of Ore deposits.

Lists of minerals are given under each which invite study and corroboration or modification by observers in the field. The paper is one which cannot fail to start thinking along a new line and prove stimulating to all who read it. It marks an endeavor to bring into more definite expression phenomena about which we have all reasoned in a more general and hazy manner.

Almost at the same time with the publication of Mr. Lindgren's paper, which, however, was read at the International Geological Congress the previous summer, somewhat the same line of attack was independently developed by Dr. P. Krusch, of Charlottenburg.[1] Dr. Krusch treats the minerals of ore deposits very much as a paleontologist does the remains of life which he calls "index fossils," or *Leitfossilien*, and by which the position of a stratum in the geological column may be determined. Thus there are oxidation products and cementation or secondary ores, which may result in the superficial horizons from originals, which themselves are characteristic of lower depths and which can only form at these individual distances from the upper world. The originals are therefore index ores, *Leiterze*, of the depths at which the vein was filled. The point of molecular volumes is not however used, although the several ores are passed in review.

The subject of underground waters has received additional attention the past year and with very interesting results. The point brought out by me at the Richmond meeting of the American Institute of Mining Engineers in February, 1901, that the ordinary ground waters were of comparatively shallow depth so far as experience furnished evidence, has received much support, although at the time it was mentioned, the tide of expressed opinion was all setting in the opposite direction. M. L. Fuller, until recently in the hydrographic branch of the United States Geological Survey, has made some additional observations and calculations.[2] If the estimated amount of the ground water be expressed in the depth of a sheet which it would furnish over the surface of the earth, the results are respectively according to Delesse, 1861, 7500 ft.; Slichter, 1902, 3000 to 3500 ft.; Van Hise, 1904, 226 ft.; Chamberlin and Salisbury, 1904, 1600 ft.; Fuller, 1906, 96 ft. These figures of course greatly

[1] "Die Einteilung der Erze mit besonderer Berücksichtigung der Leiterze sekundärer und primärer Teufen." *Zeits. f. prak. Geol.*, May, 1907, 129.
[2] Water Supply Bulletin No. 160, pp. 59-60, and 72.

reduce the possibilities of the ground waters, so far as derived from the rainfall, and as agents in the production of veins. In so far as we used to place our undivided confidence in them ten to twenty-five years ago, we seem, if we may adopt the famous phrase of the British statesman, to have put our money on the wrong horse.

The topic of deep-seated underground waters, their sources, periods of activity and effects, is also discussed in a very suggestive way by G. S. Bancroft in an extremely thoughtful paper on "The Formation and Enrichment of Ore-bearing veins."[1] With many illustrations drawn from a wide personal experience, the following general conclusions are supported: 1. The majority of veins are the products of expiring vulcanism. 2. Most of these veins were primarily mineralized by comparatively rich solutions in comparatively brief periods. 3. The solutions derived their metal values from a comparatively rich source. 4. There is a barysphere containing large amounts of the useful metals. 5. Eruptions spring from various depths and bring various kinds of magma toward the surface. 6. Only those eruptions which disturb the barysphere and bring a magma rich in metals sufficiently near the surface to be leached by vein-making solutions are productive of valuable ore deposits, other eruptions producing barren veins.

A mineralogical associate of gold quartz not hitherto recognized as of importance has been found in alunite by F. L. Ransome, in a careful piece of work upon the Goldfield, Nev., veins.[2] Alunite has the formula K_2O, Al_2O_3, $4SO_3$, $6H_2O$, and is obviously a close relative of the ordinary alums. One would anticipate its formation as the result of the action of sulphuric acid upon potassium feldspar, probably in association with other reagents. By a close study of the mineralogical associations, Mr. Ransome arrives at some very interesting and well-grounded conclusions. We find at Goldfield, in the midst of very extensive and once open-textured pyritiferous quartz-reefs, localized bodies of extremely rich gold ore, containing the native metal, pyrite, bismuthinite and some variety of tetrahedrite. With these are both alunite and kaolin. It is impossible in a brief summary to touch upon all the points of interest in Mr. Ransome's paper, but it appears that both alunite and kaolin are best referred to waters from a consolidating eruptive mass and not to descending surface waters. The ores were deposited at moderate depths, apparently by acid solutions or vapors containing gold, copper, bismuth, antimony, a little arsenic, and tellurium, hydrogen sulphide, and probably sulphurous and sulphuric acids. Since these solutions are acid, and we have previously thought of alkaline ones as productive of many gold quartz veins, the latter were probably deposited further from the source and after waters

[1] *Bulletin*, A. I. M. E., May, 1907, p. 499.
[2] "The Association of Alunite with Gold in the Goldfield District, Nevada. *Economic Geology*, II, 667. See also Bulletin 303, U. S. Geological Survey."

orginally acid had become alkaline by passage through masses of rock. The propylitic type of alteration with its abundant carbonates is perhaps due to later emissions of carbonic acid. It apparently occurs as an outer zone around the kaolin and alunite.

Magmatic emanations have now proved of so much importance in connection with ore deposition, that two years ago it seemed to be of interest to summarize what we actually knew regarding them. Accordingly F. C. Lincoln went carefully through the literature and compiled and discussed all the attainable analyses, thus placing in the hands of those dealing with this subject a series of actual facts instead of surmises.[1]

In previous reviews mention has been made of the valuable work of Dr. A. L. Barlow, of the Canadian Geological Survey, upon the ore deposits at Sudbury, and the equally careful and extended studies of Prof. A. P. Coleman, under the auspices of the Ontario Bureau of Mines. Prof. Coleman has recently summarized the whole question in a very readable paper.[2] The eruptive, carrying the ore in its bottom, is shown to shade from a norite below to a granite above. While it was yet molten the sulphides of iron, copper and nickel are believed to have settled down to the bottom and there to have crystallized, Subsequently the great sheet was folded into a synclinal trough by the collapse of the basin, and thus the edges were upturned to the dips now seen.

As apparently opposed in some respects to this view, and favoring solution-phenomena which were especially advocated by Dr. C. W. Dickson a few years back, Dr. Wm. Campbell and C. W. Knight show by their study of polished slabs, by metallographic methods, that the minerals exhibit a characteristic and invariable order of succession, viz.: 1. Magnetite; 2, silicate; 3, pyrrhotite; 4, pentlandite; 5, chalcopyrite.[3] It is to be hoped that some way may be found of reconciling these two views, which may not be after all more than apparently contradictory.

As a still different view for a pyrrhotite ore body in the Black Forest of Baden, Prof. E. Weinschenk, of Munich, has advanced the following explanation.[4] He finds the ores along the contact of the medium or basic rocks, which actually contain them, and intrusive granites. The nickel favors the neighborhood of aplitic varieties of granite. Hence the conclusion that the ores are due to the granite and are contact metamorphic deposits. Dr. Weinschenk attempts the somewhat hazardous thesis of showing the connection of nickel enrichment with intrusive granite in other districts, even in Sudbury. To this Prof. Coleman replies in a subsequent communication.[5]

---

[1] F. C. Lincoln. "Magmatic Emanations." *Economic Geology*, II, 258.
[2] "The Sudbury Laccolithic Sheet." *Journal of Geology*, XV, 759, 1907.
[3] "Microstructure of Nickeliferous Pyrrhotites." *Economic Geology*, II, 350 .
[4] "Die Nickelmagnetkieslagerstäten im Bezirk St. Blasien in südlichen Schwarzwald." *Zeits. f. prak. Geol.*, March, 1907, p. 73.
[5] *Ibid*, June-July, p. 221.

The puzzling zinc deposits in southwest Missouri have received renewed attention since the abstracts of Dr. Buckley's report and of Mr. Siebenthal's paper were given a year ago. The Joplin District folio has been issued by the United States Geological Survey, coming from the pens of W. S. Tangier Smith and C. E. Siebenthal. The former gave especial attention to the ore-deposits, the latter to the areal geology, but the recent discovery of facts apparently opposed to faulting in one important locality by Mr. Siebenthal led to his revising somewhat the original manuscript on the ores. The folio covers two ordinary quadrangles on the scale of a mile to the inch. It is beautifully printed and illustrated and the letter press goes over the ground with great thoroughness of description. The forms and distribution of the orebodies are described and mapped in greater detail than in any previous publication and the geological history and structure of the region are carefully elaborated.

Mr. Smith refers the metals to the pre-Pennsylvanian limestones which constitute the Ozark plateau. By the general subterranean flow outlined by Van Hise and Bain, the meteoric waters are believed to have effected a primary precipitation, but subsequent enrichments have operated to bring the deposits up to the present grade. That the doctors may easily disagree over this question is shown by the fact that the State Geologist, Dr. Buckley, referred the metals, just a year before, to the overlying Pennsylvanian once present in the region; and there are doubtless some, not negligible, observers still surviving, who think a deep-seated source the only defensible one from which to derive these great amounts of zinc and lead.

For the Mississippi Valley at large, and with the purpose of showing the connection of the ores with the physiographic development, H. F. Bain has discussed this phase of its geological history in a comprehensive way.[1] The author treats more particularly the Wisconsin region, but employs the subject also as the foundation for a general statement of the importance of weathering, of solution in descending waters, of precipitation and concentration, and of other distinctively superficial processes in the formation of orëbodies. The two opposing theoretical views are then contrasted under the descriptive headings of sedigenetic for the deposits due to meteoric waters, and igneogenetic for those due to magmatic.

A contribution, which will be warmly welcomed by all who read it, is the first instalment of the later work upon Leadville.[2] No monograph of the United States Geological Survey has gained greater respect than the early one upon this district and it is the sincere desire of all interested in mining geology that Mr. Emmons' name should be associated with

[1] "Some Relations of Palaeo-geography to Ore Deposition in the Mississippi Valley." *Economic Geology*, II, p. 128.
[2] S. F. Emmons and J. D. Irving. "The Downtown District of Leadville, Colorado." Bulletin 320, U.S. Geol. Survey.

its later elaboration. Mr. Emmons has been faithfully assisted by J. D. Irving. The bulletin presents the results of late mining in the downtown section of the camp, which 25 years ago had not been developed. The faults are traced and described and such new light as can be thrown on matters of genesis is indicated. The primary precipitation by uprising and presumably magmatic waters has come so much to the fore in later years, that the authors are influenced by its importance. This phase of the subject is reviewed in the closing chapter, by Mr. Emmons himself, and is contrasted with the views in the original monograph. The reader cannot help feeling that Mr. Emmons takes the later discussions and modifications a little too seriously; since upon the obscure questions of genesis all our views are confessedly working hypotheses. No master of the craft, however high his ideals, can expect to turn out a flawless work, nor one which later observers and thinkers may not modify.

Two interesting papers have appeared in 1907 relating to the gold veins of Hungary, the classic home of the tellurides. At the International Geological Congress in 1906, Mr. Bela von Inkey, who has already contributed important monographs upon the region, presented a discussion of the method of formation of the ores and the paper has since been issued.[1] The author shows the universal and intimate association of the orebodies with those andesitic erputives which have undergone the propylitic change, that is, have had their dark silicates, especially the hornblende, altered to chlorite and carbonates. Believing that the gold and its associated metals were once bases in the complex silicates, hornblende being the most important, Mr. von Inkey concludes that they were freed during this process and introduced into the veins, as one would infer, by processes akin to lateral secretion.

On the other hand M. von Pálfy[2] states that his observations show the crevices, in and along which the tellurides are deposited, to be cooling cracks in the eruptives. The ores, however, only appear where these cracks are either tangent or secant to volcanic chimneys, which in his view have yielded the nourishing vapors or solutions during the period of thermal activity. The ores are independent of the special kind of wall-rock and solely due to the relations with the old tubes of outbreak.

On the part of two writers upon ore deposits the disposition has been shown to emphasize the existence of provinces in which the ores show relationships similar to those furnished by eruptive rocks and already described among petrographers following J. P. Iddings' suggestion, by the term consanguinity. J. E. Spurr has suggested for the areas of related ore deposits the name "metallographic province," but this is

[1] "De la relation entre l'etat propylitique des roches andésitiques et leur filons minéraux." *Compte rendu*, I, p. 501, 1907.
[2] "Das Goldvorkommen im Siebenbürgischen Erzgebirge und sein Verhältnis zum Nebengestein der Gänge." *Zeits. f. prak. Geol.*, May, 1907, p. 144.

unfortunate since the words metallography and metallographic have been employed for years to designate the microscopic study of the textures of alloys and the like, when in polished slabs. L. de Launay has used the words metallogeny and metallogenetic, which are open to no objections.[1] In the opening sentences of a discussion of a topic of this kind, Professor de Launay states, that the relationships of metallogeny and structural geology will explain the special type of ore body, characteristic of any region, and that we should investigate the orginal conditions of crystallization, depth, etc., under which the ores have formed; the position of the fractures now filled, or of other mineralized fissures, with reference to the phenomena of folding and faulting; and the connection with outbreaks of eruptive rocks, etc. While this has often been done in the restricted way, we may hereafter expect broader generalizations over such provinces as may exhibit the fundamental resemblances. While in the United States we have long recognized special areas, which exhibit these, yet in many parts of the country, great diversity enters and no single type is solely characteristic. Nevertheless we may perhaps often speak of metallogenetic provinces when they do display marked characteristics.

The literature of iron ores has been specially enriched by a valuable paper by Professor H. Sjögren upon "The Geological Relations of the Scandinavian Iron Ores."[2] This general summary is particularly interesting to Americans because it places at the command of English-speaking peoples much that was inaccessible in Swedish and yet much that afforded striking parallels with our own ore bodies. Prof. Sjögren regards the great magnetite mass at Kirunavaara as a magmatic segregation which was afterwards intruded into the position where now seen between its contrasted foot and hanging. This conclusion will be of interest when compared with reviews of papers on this enormous deposit, which have been given in previous volumes of THE MINERAL INDUSTRY.

W. H. Weed has brought together in readable and easily accessible form a vast amount of information regarding copper.[3] The red metal has been such a prominent object of mining in recent years as to have attracted more attention than any other. No part of the globe can be considered as too inaccessible to deserve attention provided the ores carry fair percentages. In the older regions improvements in concentration and smelting have made phenomenally low-grade deposits available. Mr. Weed discusses the mining regions by continents beginning with Europe and ending with North America. Many citations are given which make the book extremely serviceable.

---

[1] "La Métallogénie de l'Italie, etc." Compte rendu de Congrés geologique international. 10 Session, I, p. 555. 1907.
[2] *Bulletin, A. I. M. E.*, Nov. 1907, p. 877.
[3] "Copper Mines of the World." New York, Hill Publishing Company.

The usual list of papers of areal description have appeared during the year from the various official surveys at home and abroad, and not a few in the journals and magazines. Data upon southern Nevada have been afforded in a timely way by Bulletin 303, United States Geological Survey; upon Goldfield, Bullfrog and other districts, by F. L. Ransome, G. H. Garrey and W. H. Emmons. Many additional geologic details appear in Bulletin 308, upon much the same general region, by S. H. Ball. The Marysville Mining District, Montana, by Joseph Barrell, is the subject of Professional Paper 57. Prof. Barrell gives a very interesting discussion of contact metamorphism and of the mechanics of igneous intrusion. The phenomena of the former type at Marysville, must be different from those of Mexico and the southwest, since the author fails to find the usual overwhelming evidence of contributions from the intrusive. Such changes as occur are chiefly referred to water gas, hot waters, heat and pressure. In the volumes of the Survey on mineral resources, Mr. Lindgren's review of the precious metal industry are of exceptional interest and importance, and now constitute one of the most valuable features of the official publications. The bulletins too, which now afford brief statements of the Survey's contributions to economic geology, during the preceding year are filled with brief but important papers. No. 315 covers the year 1907.

In the number of *Economic Geology*, for June, 1907, pp. 380-418, John A. Read has given a valuable description of the copper mines in the foothill belt at Copperopolis, California. This bit of individual enterprise in the study and description of a mining area, is most commendable and should be an incentive to younger mining engineers who have the requisite geological training for independent field-work.

In Peru most commendable efforts are being put forth by the official corps of engineers to issue from time to time bulletins upon the mineral resources and other geological problems of the republic. In the series, Americans who look from time to time to Peruvian enterprises for investments will find not a few papers valuable for reference.[1] The Mexican Geological Survey or "Instituto Geologico," is also rendering applied science efficient service and through the medium of its *Boletinos* publishes from time to time important contributions. Now that its officers have off their hands the issue of the *Compte Rendu* of Tenth International Geological Congress, a most formidable task, we may hope for renewed activity.

[1] They are published as Boletinos del Cuerpo de Ingenieros de Minas del Peru. 53 is the latest received.

SAMPLING AND ASSAYING.

By F. F. Colcord.

Apparatus and Appliances.

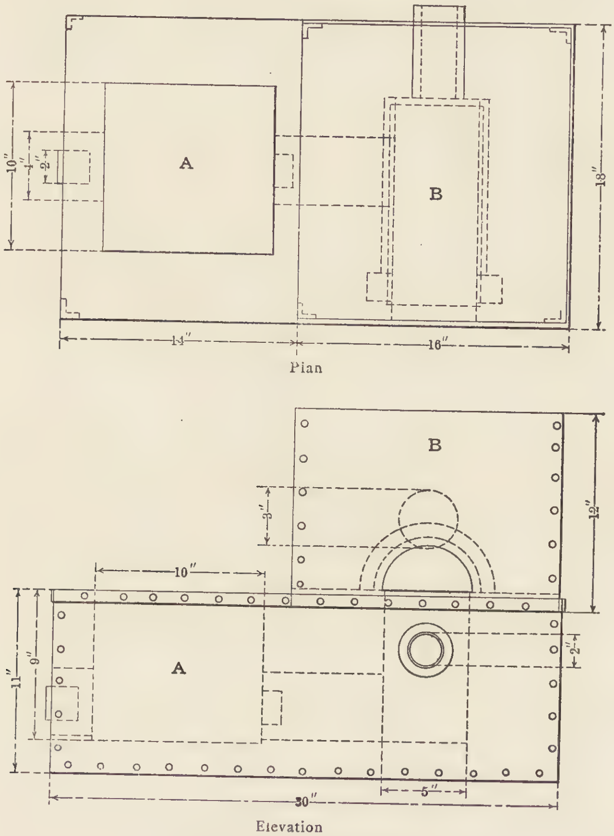
The following is a summary of the more important literature appearing during 1907 on sampling, assaying and kindred subjects.

Slag Mold.—J. J. Bailey (*Min. Reporter*, May 23, 1907) describes a new form of pouring mold for assayers, that is designed to aid the separation of slag and metal. The slag and metal are poured into the mold, from which the metal settles out through a small orifice at the bottom into a small compartment on the under side of the mold. When cool, the mold is raised from its base and a slight tap causes the metal to drop out of its compartment, leaving the major portion of the slag in the upper compartment.

Preheating Assays.—W. J. Fleck (*Min. and Sci. Press*, May 4, 1907) describes an arrangement whereby the capacity of a portable gasolene furnace may be increased materially when the cost of fuel, the expense of building, dust and slow heating, operate against the building of a larger furnace. The arrangement consists of a large standard charcoal furnace erected beside the small gasolene furnace, and fired with dry wood. The charged crucibles are first placed in the charcoal furnace and heated until slow fusion begins. The crucibles are then quickly transferred to the gasolene furnace, which is kept at a high heat, when quiet fusion begins at once, and in 10 minutes the crucibles are usually ready to pour. With this arrangement 16 assays may be melted and cupeled in a single-crucible, No. 5, Hoskins furnace in four hours, or 50 assays in a No. 6, four-crucible furnace in the same time. The flux used is first dried in open pans at a temperature of 200 deg. F., and then intimately ground and mixed with ore in a small wedgewood or glass mortar. The mixture is then transferred to the crucible, and a quick quiet fusion and a clean homogeneous slag is the result, the crucible being only slightly attacked on account of the short time of fusion. By this simple elimination of moisture and gases the need of salt or any other cover is obviated.

Gasolene Furnace.—W. E. Darrow (*Min. and Sci. Press*, XCV, 1907, pp. 749-750) has designed a gasolene-fired, assay furnace which he has found superior to some on the market. Two rectangular, sheet-steel

boxes *A* and *B*, compose the shell of the furnace. The box *A* contains the melting chamber, and the fire-box and supports for the muffle. The sides and corners of *A* and the top of *B* are filled with fine concrete. Over the concrete is a layer composed of 4 parts old plumbago crucibles (16 mesh); 2 parts asbestos packing; 1 part fire-clay; and 2.5 parts water. This mixture is spongy and is worked into place; it is also used for the



DARROW'S ASSAY FURNACE.

top and cover of the furnace. The next layer consists of 4 parts old clay crucibles (20 mesh) and 1 part fire-clay, mixed with water and is used for the firebox lining and roof over the muffle. The muffle has a bearing all around over the firebox except where two ports on either side in front connect with the firebox. In the roof over the muffle is a longitudinal groove extending the whole length and connecting with the chimney. The melting chamber and muffle are each fired with a 1¾ in. Cary burner with 2-in. fire-bosses.

The author in sampling the sulphide ores with which he deals, rolls and quarters down the sample in the usual way to about 20 oz., which is crushed through a 150-mesh screen. The ore, after crushing, is rolled and mixed on a cloth and passed through a 40-mesh sieve; this operation is repeated several times, in order to break up any little balls that may form and to insure a thorough mixture. A heavy litharge flux with a charge of 0.5 a.t. of ore and 22 to 25 grams of niter is used. The fusions, in 20-gram crucibles, are run slowly at first, then at a high heat for 5 min., and then an excess of gasolene is turned on so that the melting chamber is filled with a reducing atmosphere. This condition is maintained for 10 min., when the excess gasolene is shut off; this allows the crucibles to heat up again, and they are then poured.

Assay Office.—E. W. Buskett (*Eng. and Min. Journ.*, LXXXIV, pp. 541-542) has described the plan and equipment of a laboratory designed by him. The design calls for a frame building 30x40 ft., containing rooms for chemical and assay work, and the balances, with the necessary arrangements of tables, hood and furnace. Detail drawings are given showing the construction of the various parts of the equipment.

Producer-Gas Furnace.—Oskar Nagel (*Electrochem. and Met. Ind.*, V, p. 398) has shown the design of a muffle furnace, fired by producer-gas wherein the waste gases heat the air used with the producer-gas.

Furnace Arrangement.—G. T. Holloway (*Trans.* I.M.M., XVI, pp. 341-349) has presented the working drawings and specifications for a set of crucible and muffle furnaces. The plans call for two coke-fired, wind furnaces and one coke-fired, 4-muffle furnace. The dimensions of the furnaces, thickness of walls, size of apertures, etc., allow the use of ordinary fire-brick with the minimum of cutting. The crucible furnaces have sloping tops, protected with sheet iron, with a depth inside above the grate bars, of 17 in. at the front, and 23 in. at the back, and an area 15 in. square. The furnaces open into a horizontal flue at the back 9x9 in., which connects with the chimney of the muffle furnace. The doors are made of tile, bound with iron and mounted on wheels allowing lateral movement. The muffle furnace is arranged for 4 muffles, 15x9x6 in., two at the top, side by side, and the other two below, one under the other. The front of this furnace is not bricked in, as is usual, but after the muffles are in place, is stopped up with a stiff mixture of fire-clay and silicate of soda. The specifications include all the material for the building of the furnaces, starting from a concrete foundation. These plans recommend themselves to one contemplating the building of assay furnaces.

Improvised Assay Furnace.—W. R. Wade (*Eng. and Min. Journ.*, LXXXIII, p. 441) shows his ingenuity in the use of a blacksmith's forge as an assay furnace. The crucible fusions were very readily made; for cupelling, a large crucible with a hole in the bottom was laid on its side

in the fire. Half-burnt coal was piled around the crucible, in which two cupels were run at a time.

Sampling.

Ore Containing Metallics.—F. A. Thompson (*West. Chem. and Met*, III, pp. 16-19) has submitted the results of sampling an ore, containing metallics, not to show the proper method of sampling , but to show the difficulties attending the operation. The ore was porphyritic, the feldspar being pretty well kaolinized and in places was heavily iron stained. The gold and silver were present almost entirely as metallics, flakes of gold being visible to the naked eye; sometimes pieces of gold as large as grains of rice were visible.

The whole lot, weighing about 10 tons, was crushed to 0.5 in. size, and one-fifth cut out with a Vezin sampler. This two-ton cut was roll-crushed to about 0.12 in., and carefully riffled twice on a Jones sampler, having riffles 1 in. wide. This 500-℔. sample was put through smaller rolls to about 0.06 in. size and cut once with the riffle sampler. Each half was treated separately (being called original and duplicate) by cutting once with the riffle and rolling to about 0.025 in. size, and was again cut to about 10 ℔. The 10-℔. sample was passed through a coffee grinder several times, riffled down to approximately 2 ℔., and sifted on a 100-mesh sieve, about 90 per cent. passing through without further grinding. The oversize was bucked on a board until the metallics, which often showed up as ribbons of gold on the board, passed through the sieve. The pulp was mixed, coned and quartered, and divided into samples for the mine and sampler. Results of the assay were as follows: Sampler original, 9.29 oz.; Mine original, 11.22 oz.; Sampler duplicate, 12.60 oz.; Mine duplicate, 13.74 oz.; Average, 11.71 oz. per ton.

The results on each sample being the average of at least four assays of 0.5 a.t. each, which upon the sample differed at times as much as 0.5 oz., it was decided to resample, as the differences were evidently there. The resample was started with the one-fifth portion of the original sampling, and the operation from that point to the division into the original and duplicate was repeated, with more care if possible, than at first. The sample now, instead of being divided into two parts, was divided into four, and each part treated separately and reduced as before. The four final portions were ground in the coffee mill and the pulp weighed before screening. The metallics were then sifted out on a 100-mesh sieve, wrapped in lead foil and assayed, two by the mine and two by the sampler, each accepting the other's results. The through-screen portion was divided into samples for the mine, sampler and umpire.

RESULTS OF SECOND ASSAY.

Oz. gold per ton.

	Pulp			Metallics		Final Results
	Mine	Sampler	Average	Mine	Sampler	
First original	10.23	10.00	10.11	....	2.23	12.34
First duplicate	15.88	14.36	15.12	3.20	....	18.32
Second original	16.80	14.24	15.52	....	7.04	22.56
Second duplicate	16.90	15.23	16.07	2.39	....	18.46
Average	....	....	....	....	....	17.92

Settlement was on this basis, and the experiment stopped. In the first sampling it was evident that metallics remained on the bucking board and muller, but in either case the results are not conclusive, and it is a question whether an accurate sample can be obtained on such material. The ratio of weight of sample to size of particles is well within the limits prescribed by sampling authorities, but it would have been better to have made four separate samples on the whole lot rather than on the one-fifth cut.

Sampling Rich Ore.—L. M. King (*Min. and Sci. Press*, XCIV, pp. 241-242) presents the results of sampling a high-grade, telluride, gold ore from Goldfield, Nevada. The lot weighed about 50 tons and for sampling purposes was divided into 10 lots of approximately 5 tons each, which were sampled separately. Each lot was crushed to pass a 0.25-in. screen and coned twice to mix it thoroughly. The lot was then sampled down on riffles of the Brunton type to about 2000 ℔., crushed to pass a 10-mesh screen and again coned twice. The sample was next split by the riffles into two portions of 1000 ℔. each, and these in turn were split once to 500 ℔. each, making four samples in all. Each of these samples was treated as an independent sample, riffled down to about 20 or 25 ℔., and dried and ground to pass 100 mesh. As a rule, the four samples from a lot agreed closely, but occasionally one of the four would vary considerably from the other three; for instance, the writer obtained these gold results on two lots, in ounces gold per ton.

RESULTS OF ASSAY ON DUPLICATE SAMPLES.

Lot	Sample 1	Sample 2	Sample 3	Sample 4
3,416	562.92	562.27	564.91	549.15
3,421	830.84	827.94	812.56	827.04

A basic litharge flux was used in assaying. The fusions were heated slowly for 30 min. and strongly for 20 min., throwing down a 25-gram lead button from 0.25-a.t. charge.

Assay Methods.

Sulphide for Reducing Agent.—C. A. Rose (*West. Chem. and Met.*, III, pp. 216-217) has suggested the use of gold- and silver-free iron sulphide as a reducing agent in place of carbonaceous matter in the assaying of silicious ores. The author uses for 0.5 a.t. of ore, 20 grams sodium bicarbonate; 10 grams potassium carbonate; 80 grams litharge; and 3 grams iron sulphide, and thus avoids the danger of boiling over that attends the use of carbonaceous reducing agent.

Heavy Litharge Flux.—B. M. Snyder (*West. Chem. and Met.*, III, pp. 125-129) advocates the litharge method of fire assaying. The method is simple, easily carried out and requires little skill of manipulation. The gold recovery is practically complete but the silver requires special precautions to obtain good results. Crucibles of coal placed in the muffle with the fusions, thus creating a reducing atmosphere, will aid the silver recovery. An elimination of 88 per cent. of the copper in a 50 per cent. matte was obtained in treating 0.1 a.t. of the matte with 8 a. t. of litharge. Three stock fluxes used by the author were: For silicious ores containing over 50 per cent. silica—24 parts litharge; 2 parts sodium carbonate; 2 parts potassium carbonate. For basic ores—the same as for silicious ores except the addition of 3 parts of silica. For mattes—32 parts litharge; 1 part each of sodium and potassium carbonate; and two parts of silica. A small amount of alkali and silica improve the slag, lessen the crucible corrosion, and make a cleaner, more coherent lead button. With ores, 0.5 a.t. is taken of the ore to 4 to 5 a.t. of flux, using reducing or oxidizing agent as necessary and a cover of salt. For mattes, 0.25 a.t. is mixed with 8 to 9 a.t. of matte flux with salt for a cover. The copper-bullion method is: Mix 0.1 a.t. of the copper with 0.8 gram sulphur and cover with 8 to 9 a.t. of matte flux and salt.

Cobalt Silver Ores.—R. W. Lodge (*Bull.* 17, A.I.M.E.), has described the effect of high litharge in the crucible assay for silver. It is shown that the use of a large excess of litharge in the assay of some ores gives results that are uneven and low. The particular figures given were obtained in the assaying of rich, arsenical, nickel-cobalt ores from Ontario, Canada. The two tables below support the author's contentions. The ore in Table I contained 5.06 per cent. nickel and 9.12 per cent. cobalt, while the ore in Table II originally contained: Nickel, 0.3; cobalt, 8.0; arsenic, 55.0; and had been amalgamated.

TABLE I.—CRUCIBLE-ASSAYS.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Lead Button.	Silver in Lead.	Silver in Slag.	Silver in Cupel.
	g.	g.	g.	g.	g.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.
$\frac{1}{20}$ A. T.	10	10	30	1.5	19	2051.4	9.6	34.0
$\frac{1}{20}$ A. T.	10	10	40	1.5	21	2056	....	....
$\frac{1}{20}$ A. T.	10	10	40	1.5	21	2050	....	....
$\frac{1}{20}$ A. T.	10	10	80	1.5	30	1968.6	....	....
$\frac{1}{20}$ A. T.	10	10	80	1.5	22	1944.6	135.2	35.0
$\frac{1}{20}$ A. T.	10	10	80	1.5	21	1984.8	70.2	34.6
$\frac{1}{20}$ A. T.	10	10	80	1.5	21	1914.8	....	....

TABLE II.—CRUCIBLE-ASSAYS.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Lead Button.	Silver.
	g.	g.	g.	g.	g.	Oz. per Ton.
$\frac{1}{20}$ A. T.	10	6	35	1.5	24	293
$\frac{1}{20}$ A. T.	20	6	35	1.5	23	292.8
$\frac{1}{20}$ A. T.	20	6	35	1.5	25	291.8
$\frac{1}{20}$ A. T.	10	6	60	1.5	24	288.0
$\frac{1}{20}$ A. T.	10	6	80	1.5	26	283.6
$\frac{1}{20}$ A. T.	10	6	100	1.5	26	279.8

The author is unable to explain the reasons for the low and uneven results and, while some of these ores yield satisfactory results by the crucible assay, he prefers on account of this uncertainty to use the scorification method on ores from the Cobalt district, which carry much nickel. With fine grinding of these ores, the greater part of the silver pellets may be removed and good results obtained by scorifying 0.05 or 0.1 a.t. charges with 3 to 8 grams borax; 65 or more grams of lead; some silica; and using a medium high temperature. If the ore, however, is either silicious or calcareous, poor in silver, and has only a small percentage of nickel or similar impurity, the crucible method may be used to advantage if the litharge is kept low, and such an amount of ore taken as to avoid the use of niter or high litharge.

Results on Telluride Ores.—J. N. McLeod (*Min. World*, May 18, 1907) has detailed some of his experiences in assaying telluride ores. The various samples all contained tellurium or tellurium combined with other metals. Sample No. 1 was run with the usual Cripple Creek, high-litharge flux with 0.25 a.t. of ore, and inquarted with silver. A total absence of any button was the result after cupellation. Another charge with more litharge and silver gave the same result. The cupel from this charge was assayed without obtaining any button, but the assay of the second cupel yielded a small one. The combination assay, however, gave a result of 50 oz. gold and 3000 oz. silver. Sample No.2, which contained selenium as well as tellurium, yielded by the combination method 7.5 oz.

gold and by the scorification method, using 0.05 a.t. of ore and an excess of lead and rescorifying twice, 100 oz. gold, while a wet method gave 156 oz. gold. Other samples gave just as striking results, so it is evident that all telluride ores do not yield their gold under the same treatment.

Losses with Telluride Ores.—G. T. Holloway and L. E. B. Pearse (*Trans.* I.M.M., Bull. 39) have made an exhaustive study of the causes of the gold losses in the assaying of telluride ores. Several methods for the qualitative determination of tellurium are described. Early experiments proved that the direct loss of gold by volatilization during roasting was slight, and was not effected by the presence of mercury. The loss of gold by volatilization during the cupellation of lead buttons, obtained from any ordinary telluride assay, was found to be extremely small, in fact not greater than in the tellurium-free lead buttons. The cupellation loss evidently depends on the ratio of lead to tellurium in the lead button; for instance, an alloy containing lead, 60; gold, 20; tellurium, 20, gave an absorption loss in cupelling of 7.5 per cent. gold. The cupellation of this same alloy with a sufficient excess of lead gave a normal volatilization and absorption loss. There is practically no loss of gold in the slag in scorifying highly tellurous lead, but there is a condition in the scorifying which greatly affects the loss in the subsequent cupellation. If a 50-gram lead button was scorified down to two-thirds its weight, it was found that a considerable portion of the tellurium was expelled, but if the scorification was carried further, and especially if the lead was allowed to remain under the slag for any length of time, the tellurium was reduced from the slag, re-entered the lead and enriched it to a greater extent than before scorification. The enriched button from such a scorification as described, would yield a heavy loss in cupellation. The authors' summary is: Firstly, if the lead button from the crucible assay be kept large the tellurium will be mostly eliminated in the earlier stages of cupellation and the gold loss will not be appreciably effected. Secondly, a preliminary scorification of the large lead button to two-thirds of its original weight is advantageous. Thirdly, if the proportion of tellurium to gold be small in comparison with the lead in the alloy to be cupelled, the loss will be normal.

The scorification method for the original treatment of the ore is not satisfactory on account of the smallness of the charge, the danger of mechanical loss and poor decomposition of the ore. The authors recommend the crucible fusion method, after trials of many different methods, as best suited for the assaying of telluride ores, both for gold and silver. Among the various charges recommended for different grades and kinds of telluride ores is the following for average-grade, highly silicious ores (in grams): Ore, 50; red lead, 120; soda ash, 60; borax glass, 10; flour, 7.

In general it was found best to use a charge which yielded a basic slag fusible at a low red heat, as an acid slag gave low results. The addition of fluorspar to the charge was beneficial. The lead button obtained should be large, and should be scorified preliminary to cupellation as mentioned before. Salt should not be used for a cover. Fine grinding is essential in the sampling and assaying of telluride ores. The authors recommend that none of the wet methods be employed except possibly in the cases of certain high-copper or other uncommon ores. The conditions for the wet assay, if used, are described, as well as methods for working up slags and cupels.

Combination Method for Copper.—A. R. Crook (*Eng. and Min. Journ.* LXXXIII, p. 853) finds the combination wet and dry method of assay superior to the dry method alone for many copper ores. When the residues in the combination method are not suitable for scorification, the writer recommends their fusion in crucibles. The method employed is to dissolve 1 a.t. of the ore in dilute nitric acid with the aid of heat. The dissolved silver and suspended gold are then precipitated in the unfiltered solution by the addition of 2 to 4 c.c. normal salt solution, 5 c.c. sulphuric acid, and 10 c.c. lead acetate solution. The solution is allowed to settle over night, and in the morning is filtered and washed. The residues are dried and burned in roasting dishes. After burning, the residues are mixed with 1.5 a.t. litharge; 1 a.t. sodium carbonate; 10 grams borax and 1.5 gram argols in a 20-gram crucible, covered with salt and the usual procedure followed.

Assay of Highly Silicious Ore.—G. B. Hogenraad (*Journ.* Chem., Met. and Min. Soc. of South Africa, Aug. 1907) has given the results of some experiments in fire assaying at the Redjang Lebong mine, Sumatra. The general run of the material to be assayed ranges from 76 to 86 per cent. silica with small amounts of alumina and iron, and traces of other metals insufficient to interfere. For 1 a.t. of ore there was formerly used 69 grams sodium carbonate; 23 grams borax glass; 23 grams litharge; and 1.45 grams charcoal. The slag was poor and the results low in silver. The author wisely diminished the quantity of reducing agent and soda, increased the litharge and omitted the borax. The results in gold and silver were higher with the new flux, but not wholly because the borax was omitted, as was claimed. The new flux was 50 grams sodium carbonate; 49.5 grams litharge; and 0.45 gram charcoal to each charge of 1 a.t. of ore. The expedient of running checks with the silver work was also adopted.

Fire Assay for Lead.—C. A. Cooper (*West. Chem. and Met.*, III, pp. 130-135) has added to the literature on the uncertainty of the fire assay for lead. Working with concentrates containing about 32 lead; 9 zinc; and 22.5 iron, the fire assay with 10-gram charges was 3.35 per

cent. below the wet assay, and with 20-gram charges 2.78 per cent., while another assayer obtained differances of 2.65 and 2.25 per cent. respectively with 10- and 20-gram charges, between the wet and dry assays. The difficulties of the molybdate method for low leads is also noted.

Assay of Cyanide Solutions.

By Silver Nitrate.—H. W. Gendar (*Min. and Sci. Press*, XCIV, p. 401) assays cyanide solutions for gold by precipitating the gold as the double cyanide of gold and silver. A solution of silver nitrate is added to 30 c.c. of the working solution until a precipitate ceases to form. The precipitate is collected on a filter paper, reduced on charcoal with a blowpipe, and the button parted as usual.

By Cement Copper.—Douglass Muir (*Min. and Sci. Press*, XCIV, p. 564) assays cyanide solutions by adding an excess of sulphuric acid and then precipitating the gold with 0.5 to 1 gram of cement copper. The precipitate is treated by the crucible method.

By Copper Cyanide.—Augustus MacDonald (*Min. and Sci. Press*, XCIV, p. 719) adds an excess of weak copper sulphate solution to the cyanide solution to be assayed, and then acidifies with dilute sulphuric acid. The copper cyanide precipitate collects all the gold and silver, is filtered off and assayed by the crucible method. A small piece of cyanide is added to weak solutions in order to give a sufficient precipitate.

Chiddey Method.—W. F. Tindale (*Min. and Sci. Press*, XCIV, p. 720) uses the Chiddey method for cyanide solutions as being applicable to both gold and silver solutions. The author takes 10 a.t. of solution; 5 grams zinc shavings; 40 c.c. of a 20-per cent. lead acetate solution; boils the mixture and then adds 40 c.c. hydrochloric acid. When action has ceased, the solution is again boiled, the lead collected and the usual procedure followed. The average of 28 determinations by the Argall method gave 0.0183 oz. gold and 1.416 oz. silver, while the Chiddey method gave 0.0181 oz. gold and 1.438 oz. silver.

Notes on Chiddey Method.—J. L. Martel (*West. Chem. and Met.*, III, pp. 66-67) has given some details of the Chiddey method. The general objections to the method are the imperfect elimination of the zinc, causing corrosion of the cupel and scattering the button, and the precipitation of lead chloride, rendering the lead brittle and hard to collect. The writer uses the same quantities of reagents as noted in W. F. Tindale's paper. Emphasis is laid on the complete precipitation of the lead before the hydrochloric acid is added, and it is also noted that the lead will be more spongy if the solution is not boiled at this point. After the zinc is dissolved the solution should not be boiled on account of the loss due to hydrocyanic acid. The use of finger cots, spatulas, etc., is avoided if the

supernatant liquor is decanted and the lead dropped into a basin of water where it may be collected and squeezed together with the fingers.

Assay of Gold and Silver Bullion.

Modification of Volhard Method.—E. A. Smith (*Trans.* I. M. M., XIV, pp. 154-158) describes a modification of the Volhard method for the assay of silver bullion, whereby the end point is determined with greater certainty and accuracy. The modification consists of adding sufficient ammonium thiocyanate to the check assay to intensify the red color of the ferric thiocyanate, and to use this color as a standard of comparison, instead of the usual first pink coloration. The normal ammonium thiocyanate is prepared of such strength that 100 c.c. is equivalent to about 1.0003 to 1.0005 gram of silver. To standardize the normal solution, 1 gram of pure silver is dissolved in 10 c.c. of dilute nitric acid (sp.gr. 1.2) in a 250-c.c. Gay-Lussac bottle and heated until all traces of nitrous acid are expelled. The solution is then diluted with 50 c.c. of the indicator (50 c.c. contains 2 c.c. of a saturated solution of ferric alum decolorized with nitric acid) and 100 c.c. of normal thiocyanate added to it from a pipette. The bottle is then stoppered and well shaken for two minutes, and allowed to settle, leaving a perfectly clear red colored solution. The addition of 0.5 c.c. decinormal solution should render the solution colorless, if not, the normal solution should be so adjusted as to secure this result. The color obtained is used as a standard of comparison and all assays worked to it.

In assaying silver bullion such an amount is taken as to contain about 1 gram of silver. The bullion is dissolved and treated like the proof. The decinormal thiocyanate is added drop by drop, with a thorough shaking after each addition, until the color corresponds exactly to that of the proof. The addition of one drop (0.05 c.c. equivalent to 0.05 mg. of silver) makes a perceptible difference in the color. The comparison of the assays with the check is best made by placing the bottles on a shelf with a back of opal glass, and facing the direct light from a window. In some cases it is advisable to filter and then compare the solutions. When copper is present the strength of the normal thiocyanate should be increased, and an approximate amount of copper added to the proof to correspond to that present in the bullion. The usual volumetric precautions should be followed and the bottles should be shaken at once after the addition of the normal thiocyanate.

Comparison of Methods for Silver Bullion.—Dr. Kirke Rose (*Trans.* I. M. M., XVI, pp. 158-161) in the discussion of the foregoing paper quotes the following results as illustrating the relative accuracy of the Gay-Lussac, the Volhard, and the cupellation methods of assaying silver

bullion as ordinarily carried out. The assays were made to compare two samples of fine silver. One gram of the silver was taken for assay in each case.

COMPARISON OF METHODS FOR SILVER BULLION.

Gay-Lussac.		Volhard.		Cupellation.	
Sample A	Sample B	Sample A	Sample B	Sample A	Sample B
999.85	999.85	1000.15	999.75	987.95	987.85
1000.05	999.95	1000.00	999.80	987.80	987.35
1000.00	999.75	1000.10	999.85	987.25	987.45
999.90	999.80	999.85	1000.00	987.35	987.30
999.80	999.70	999.80	999.95	988.00	988.15
1000.00	....	....	....	987.35	987.25
999.85	....	....	....	....	....
1000.00	....	....	....	....	....
1000.05	....	....	....	....	....
999.90	....	....	....	....	....
999.94	999.81	999.98	999.87	987.62	987.56

The greatest divergence from the mean in the 15 assays by the Gay-Lussac method was 0.14; in the 10 assays by the Volhard method, 0.18; and in the 12 cupellation assays, 0.59 degree of fineness.

Approximate Method.—J. E. Clennell (*Eng. and Min. Journ.*, LXXXIII, pp. 1099-1100) describes a method of approximate analysis of gold and silver bullion for gold, silver, selenium, lead, copper, iron and zinc. It is preferable to determine the gold and silver on separate portions, but all determinations may be made on one portion. One gram of drillings is dissolved in a flask with two treatments of 75-per cent. nitric acid, boiling each time. The residue is washed by decantation, dried, ignited, and weighed for gold. If necessary this residue is examined for its purity by dissolving in aqua regia, washing the residue after this treatment by decantation, and deducting its weight from that of the previous gold residue.

The combined washings and acids from the treatment of the gold with nitric acid are boiled, and the silver precipitated with about 5.c.c. of strong hydrochloric acid. The silver chloride is washed well by decantation, the decantations poured through a filter, and the bulk of the precipitate retained in the original flask. A known amount of an approximately 1-per cent. solution of potassium cyanide containing a little alkali, sufficient to dissolve all the silver chloride, is poured through the filter and caught in the original flask. When most of the silver chloride is dissolved the solution is decanted through the filter. Any chloride still undissolved is treated with a few cubic centimeters of concentrated ammonia water which is also poured through the filter. After the filter has been well washed, 10 c.c. of a 1-per cent. solution of potassium iodide is

added to the filtrate and the residual cyanide titrated with silver nitrate solution. The silver present is then calculated from the standardization of the solutions.

The filtrates from the chloride precipitation are evaporated to dryness on the water bath with 0.5 gram of sodium chloride. The evaporation is repeated several times with hydrochloric acid and the residue finally taken up in the least quantity of dilute hydrochloric acid. The solution is then boiled with more hydrochloric acid until the last traces of nitric acid are expelled. The solution is diluted to between 50 and 100 c.c. and filtered. Sulphur dioxide gas is passed through the hot solution; the solution boiled, filtered and the selenium weighed. The author also tried the volumetric estimation of selenium according to the following reaction: $4\ KI + 4\ HCl + H_2SeO_3 = Se + 4\ I + 4\ KCl + 3\ H_2O$, and titrated the free iodine with thiosulphate.

The filtrate from the selenium precipitation is evaporated to fumes with 10 c.c. sulphuric acid, cooled, diluted with water, 10 c.c. alcohol added, and the precipitate allowed to settle. The lead sulphate is filtered off, dissolved in ammonium acetate, the solution filtered and the filtrate evaporated with sulphuric acid to fumes. The lead sulphate is then estimated gravimetrically or volumetrically.

The alcohol is expelled from the lead sulphate filtrate by boiling and the copper precipitated with hydrogen sulphide. The copper sulphide is filtered off, dissolved in nitric acid, using potassium chlorate if necessary, and the copper estimated volumetrically or colorimetrically.

The filtrate from the copper sulphide precipitation is boiled to expel the hydrogen sulphide, oxidized with nitric acid and the iron thrown down with ammonia. The iron hydrate is filtered off and the iron determined colorimetrically, as it is usually small in amount. The filtrate from the iron is slightly acidulated with sulphuric acid and the zinc precipitated with hydrogen sulphide and subsequently determined by any of the usual methods.

Rapid Modification of Volhard Method.—J. E. Clennell (*Trans.* I.M.M., XVI, pp. 164-165) has described a modification of the Volhard method for silver, useful for a rough preliminary bullion assay. A silver nitrate solution is made up to contain 0.0025 gram silver per cubic centimeter, and a potassium or ammonium thiocyanate solution to correspond exactly to the silver solution. A portion of 0.1 gram of the bullion is accurately weighed out and dissolved in about 50 c.c. of nitric acid (25 per cent. by volume) the solution after a good boiling is decanted into a larger flask and the residue again boiled with stronger acid (75 per cent.). The residue is washed, dried, ignited and weighed as gold. The combined acid solution and washings from the gold are cooled, 5 c.c. ferric indicator and a measured excess of the thiocyanate added, and the whole thoroughly

shaken. The solution is then carefully titrated back with the standard silver solution until the red color disappears. An accuracy of 1.25 per 1000 is claimed.

Assay Office Management.

Stamp-Mill Assaying.—W. B. Blyth (*Ballarat Sch. Min. Mag.*, Vol. IX, No. 4) has described the sampling and assaying in connection with a large stamp mill. The mine samples as received weigh 5 to 10 ℔. and are crushed to 0.25 in., sampled down with a riffle sampling tray and reduced by further grinding to pass a 40-mesh sieve. Compressed air is used to clean the grinders after each sample. The mill and concentrate heads are sampled at regular intervals during the 24 hours, the samples subsequently dried, quartered and ground through 60-mesh. Sands heads, sands residues and concentrates to be cyanided, are sampled in the vats by taking about 16 spear samples to the full depth of each vat. These samples are dried, quartered down and ground through 40-mesh. Solution samples are taken at the delivery and outlet pipes of the zinc boxes. Stock fluxes are made in advance of requirements and kept in tin boxes. Of a silicious ore, 2 a.t. requires 5.5 a.t. of the following flux: 10 parts sodium bicarbonate; 5 parts borax glass; 6 parts litharge; with about two grams of flour for reducing agent. A flux of 5 parts litharge; 3.5 parts sodium bicarbonate; and 2.5 parts borax glass is used for concentrates. Every fifth sample is run in duplicate, the fusions made in G crucibles, silver foil added for inquarting, and borax for a cover. The fusions are run in coke-fired, wind furnaces and in cupellation Mabor cupels are used. The parting is done in No. 000 porcelain crucibles with a single treatment of one part nitric acid to 2.5 parts water. In assaying solutions, 4 a.t. are measured into a beaker with 4.5 grams of zinc shavings and 15 c.c. of a 10-per cent. solution of lead acetate. The contents of the beaker are brought to a boil, 20 c.c. of strong hydrochloric acid added, and when action has ceased the solution is again boiled until the zinc is entirely dissolved. The solution is then decanted, the lead washed, gathered into a ball and squeezed dry and cupelled with the addition of silver.

Assay Record System.—R. K. Meade (*Chem. Engr.*, Aug. 1907) has described a laboratory record system based on card or loose-leaf filing. Every sample is marked by tag or otherwise, with all the necessary information, so that its identification is always clear. It may be well to also keep this information in a record book. The original data of the analysis of each sample are kept on a card suitable for a filing cabinet, where it is filed according to the classification adopted. The reports of analyses are also made on cards for the filing cabinets of the recipients. Loose-leaf ledgers are used for records of extended experiments.

Identification of Cupels.—H. Monckton (*Journ.* Chem. Met. and Min. Soc. of South Africa, Aug., 1907, pp. 54-55) has adopted a scheme to avoid mistakes due to the interchange of the assays during the work of mine sample assaying. To certain of the assays, portions of 0.025 gram of pure copper are added, and the distinct green stain subsequently imparted to the cupel furnishes a ready means of noting any mistakes. Batches of 28 are run by the author, and in the first batch, copper is added to No. 1 sample and every fourth sample thereafter. In the next batch, copper is added to the second sample and every fourth one thereafter and so on.

Recent Publications.

C. H. Fulton ("A Manual of Fire Assaying") has published a book on assaying that is a valuable contribution to this extensive field and includes the notes on assaying previously reviewed.[1] The opening chapters are devoted to the description of various types of furnaces and tools. The sampling of ores and the use of the different reagents in assaying are well explained. The instructions in regard to balances and weights are to the point. The noteworthy chapters are those on reduction and oxidation, and slags. Cupellation, parting, different methods for the assay of impure ores and their reactions are all treated with the fullness they require. Other chapters take up special methods of assay, errors in assaying, the assay of bullion, and the assay of ores containing platinium, iridium, gold, silver, etc. The whole book is exceptionally good and commends itself not only to the student but to the experienced assayer as well.

E. W. Buskett ("Fire Assaying") has written a brief book on the fire assay of lead, gold, and silver ores. In the discussion of sampling, various types of crushers are illustrated with cuts, as well as the apparatus and furnaces for the fire work. The use of the fluxes and reagents, the assay of acid and basic ores, and of lead ores are briefly covered. The usual methods of sampling and assaying base and fine bullion are given. The chapter on methods of handling work is well worth the attention of the novice. Under the heading of laboratory tests, are described amalgamation, chlorination, and cyaniding on a small scale. The author has perhaps covered the field as well as possible in the brief space of the 78 pages devoted to the work.

L. S. Austin ("The Fire Assay of Gold, Silver, and Lead in Ores and Metallurgical Products") has brought out a book following a single system of assaying, intended for ores and metallurgical products in the Rocky Mountain States. The sampling of ores is described in a rather brief chapter. The general precautions in the work, the apparatus and

[1] *The Mineral Industry*, Vol. XV.

several types of furnaces are well described. The use of brass forceps instead of ivory tipped ones for handling fine weights is to be condemned. The usual fluxes and reagents are described. The scorification and crucible methods are given with their chemical reactions. Then follow chapters on cupellation and parting, special methods, as roasting, assay of matte, of high grade silver sulphides, of cyanide solutions, of base bullion, of silver and gold bars, of blister copper and of lead ores for lead. The book is brief but covers its intended field.

Additional Bibliography.

A. Hollard and L. Bertriaux (*Bull.* Soc. d'Encour., 1906) have described in detail the analysis of crude copper.

N. Dègoutin (*Bull.* Soc. l'Ind. Min., V, pp. 795-928, and 1167-1181) has written an extensive article on the practical study of gold minerals. The entire field, from the preliminary observations to the final quantitative estimation of the gold in ore, is covered and the work constitutes a manual of assaying.

A. McA. Johnston (*Journ.* Chem. Met. and Min. Soc. of South Africa, Oct., 1907) has presented in a very able paper the economic value of the laboratory.

P. Leidler (*Zeit. Chem. u. Ind. Kolloide,* 1907, 2, p. 103) has devised a new scheme for the estimation of gold when in solution as auric chloride or aurichloric acid by means of a solution of sucrose.

A. Fournier (*Compt. Rend.*, 1907, 144, pp. 378-381) describes a wet method for the determination of gold.

F. C. Hughes (*Trans.* Min. and Geol. Inst. of India, II, Part 1, July, 1907) describes the sampling of ores and bullions, especially silver bullion for mint purposes.

C. Toombs (*Journ.* Chem. Met. and Min. Soc. of South Africa, VII, pp. 277-279, 331-332, 360-362, and 411-415; and VIII, pp. 44-45) has described the assay of the pulp as delivered from the screen at the Meyer & Charlton gold mine under "the new metallurgy."

PROGRESS IN ORE DRESSING AND COAL WASHING IN 1907.

By Robert H. Richards and Charles E. Locke.

Among the special features which stand out prominently in the literature during 1907 are the following: Prof. Bring's investigation on the current in jigs has thrown great light upon this matter. The large installations in Nevada and in Utah are necessary for the solution of the problem of handling low-grade ores at a profit. It is probable that this feature may be considerably extended in other parts of the West. In connection with the Joplin district, which for years has been representative as a district of small temporary plants, the more permanent character of the sheet ground has led to larger mills and a commercially better process.

In fine-grinding, the tube-mill question still stands out prominently and a large amount of literature has been written upon this subject. Like any new application, the field as yet does not seem to have been completely determined. There appears to be a tendency to use these machines for concentration purposes in spite of their well known tendency to slime the ore. Flotation processes vie with tube-mills in the amount of consideration which they have received. Elmore's new process appears to promise more economical results than did his earlier process. McQuisten's process is a bold step in the field of flotation and its outcome will be watched with much interest.

Ore Crushing.

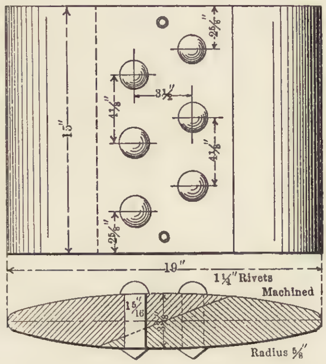
Crushing Machinery.

Breaking Piece for Jaw Breakers.[1]—This arrangement is well shown in the figure. The toggle between the swinging jaw and the pitman is cast in halves, the two faces machined and riveted together with six $1\frac{1}{4}$-in. rivets. The plane of contact makes an angle of 20 deg. with the plane of direct compression. The back toggle is cast in the usual solid form. The toggle shown is designed for a 9x15-in. Blake breaker crushing moderately compact quartz through a throat opening having a maximum width of 2 in. The advantages claimed over other forms of breaking

[1] G. E. Brown. Inst. Mining and Metallurgy, Vol. XVI (1906-07), p. 195. *Min. Reporter*, Vol. LV (1907), p. 114.

pieces are: (1) It is located very close to the initial point of the stress and there is no danger that the breaker will give way at an unexpected point; (2) when the toggle shears the jaw falls back and the breaker readily clears itself of ore.

Coarse and Fine Rolls.[1]—The Lührig coarse rolls for crushing material of about 65 to 25 mm. have the following points. The foundations are



BREAKING PIECE FOR A ROCK BREAKER

made of steel channels riveted together, which is claimed to be more solid than cast iron. Only the fixed roll is driven, the transmission coming through a clutch coupling in the shaft from the large pulley mounted on outboard bearings which acts also as a fly wheel. The boxes for the roll shafts are split on an incline, which makes it easier to remove the roll shafts for the purpose of turning down or changing the roll shells, and at the same time is mechanically better for withstanding the pressure from crushing. The rolls are made as large as 1000 mm. diameter and 350 mm. face. Shells are made thick, are turned down frequently and are worn very thin. Dust caps are placed over the ends of the shafts. The boxes of the movable roll slide in guides in the steel frame. There is plenty of space between the boxes and the ends of the rolls. Rubber springs are used instead of steel. The roll cores are of cast iron. Coarse rolls of 1000x350 mm. size make 40 r.p.m., have a driving pulley 2500x250 mm., require about 13 h.p. and crush about 10,500 kg. per hour.

Fine-crushing Lührig rolls have the following features: The movable roll is mounted on swinging pillow blocks, pivoted into the steel frame below, a form of mounting which is shown by experience to be unsuited to coarse rolls of 1000 mm. diameter treating material of over 16 mm. size. Two tension rods and rubber springs serve to hold the swinging

[1] C. Bloemeke. *Metallurgie,* Vol. IV (1907), p. 277.

pillow blocks up to their work and nuts on the ends of the rods regulate the spring pressure. Inclined caps are used on the boxes.

New Fine-Crushing Machines.[1] — The Humboldt high-speed rolls make about 150 r.p.m., have a diameter of 1000 mm. and a face of only 175 mm. They are designed for fine crushing of hard ores. These rolls are solidly built and have the pillow blocks of the fixed roll cast in one piece with the bed plate. The pillow blocks of the movable roll slide on guides on the bed plate and are held up to their work by tension rods passing through springs behind the fixed pillow blocks. To avoid waste of power and unnecessary wear of shells the rolls are not allowed to come in contact. The springs are set tight so as to yield only when a piece of iron or other hard substance forces the rolls apart. The fixed roll is driven by a large pulley while the movable roll makes a less number of revolutions and has a small driving pulley. A special grinding arrangement may be attached for truing up the rolls. The rolls are inclosed in a tight casing.

Fine rolls made by the Amme, Giesecke & Konegen Company are 300 mm. diameter and 1000 mm. face and are set nearly vertically one roll above the other. A roller feeder above and a feed plate deliver the ore horizontally between the two rolls. The upper roll is fixed and the lower is movable, being held up by springs. A hand wheel regulates the parallelism of the two rolls and also the fineness of the crushing. By throwing a lever the rolls may be quickly thrown apart in case of emergency and as quickly moved back again. The lower roll makes fewer revolutions than the upper, ratios of 1½ to 1, 2 to 1, 2½ to 1, and 3 to 1, according to the material and the fineness. These rolls were designed for crushing cement clinker and similar materials but they have been installed at Krompach, in Hungary, for grinding middlings containing copper pyrites in spathic iron below 2½ mm.

Hardinge Patent Conical Mill.[2]—This is a tube-mill which instead of being cylindrical is made up of two cones whose bases meet at the center of the mill and whose two apices form the ends of the mill. The advantage claimed is that there is a gradation of material from the center to the ends. At the center are found the coarse particles of ore and the heaviest balls or pebbles. Toward the ends the sizes of the ore particles and the pebbles diminish. This gradation aids the discharge of particles that are already crushed sufficiently fine and keeps the coarse particles in the zone of greatest crushing effect.

Results of comparative tests are given showing that the Hardinge mill had more comminutive effect and less power consumption than either an American or a German built tube-mill. The following sizing tests were

[1] C. Bloemeke. *Metallurgie*, Vol. IV (1907), p. 422.
[2] H. W. Hardinge. *Min. Wld.*, Vol. XXVII (1907), p. 1005. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 295.

made on a Hardinge mill charged with 2000 lb. of pebbles, consuming 15 h.p., running at 25 r.p.m., crushing three tons per hour.

TEST OF HARDINGE MILL.

Size	Feed.	Product.
	Per cent.	Per cent.
On 10 mesh	25.3	0.0
Through 10 on 20-mesh	38.2	0.0
Through 20 on 40-mesh	12.2	0.0
Through 40 on 60-mesh	6.6	1.4
Through 60 on 80-mesh	5.5	5.4
Through 80 on 100	2.3	2.5
Through 100-mesh	9.5	90.8

Feeder for Tube-Mill.[1]—A feeder using the principle of the well-known Frenier spiral sand pump has been fastened to a pipe extending from the feed end of a 16x4-ft. trunnion tube-mill. Such a feeder works satisfactorily and overcomes a difficulty which formerly existed due to the excessive wear of the packing gland at the feed end. By the use of a perforated grating inside the mill it is possible to fill the mill more than half full of pebbles. The feeder is made of sheet metal 6 in. wide with 6 in. between the curves and a 6-in. hole into the mill. It has fed over 3000 tons and shows little wear. Pulp as thick as 7 parts ore to 3 parts water has been fed. Pebbles are also put in through this feeder.

Crushing and Grinding Practice.

Milling Practice with Lane Chile Mill.[2]—At the Bonita mill two systems have been tried side by side. One system had five 1000-lb. stamps with 6-in. drop, 5-in. discharge and a 25-mesh wire screen, followed by amalgamated plates, 5 ft. wide and 20 ft. long, and one Wilfley table. The other system had five 1000-lb. stamps with 6-in. drop, 4½-in. discharge and ¼-in. screen followed by a 10-ft. Lane mill with 8-mesh screen and 6-in. discharge, amalgamated plates 5 ft. wide and 18 ft. long and two Wilfley tables.

The extraction at the foot of the plates in the stamp system was 51 per cent. while in the Lane system it was 65 per cent; at the foot of the Wilfleys the corresponding extractions were 80 per cent. and 91 per cent. The Lane mill made 6 r.p.m. and crushed 40 tons per 24 hr. requiring 12 h.p., while the stamps dropped 96 times per minute and crushed 16 tons per 24 hr. Sizing tests were as follows:

---

[1] U. C. Groch and F. J. Nagel, *Min. and Sci. Press*, Vol. XCIV (1907), p. 541. W. H. Fox, *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1133.

[2] J. A. Stewart. *Min. Reporter*, Vol. LV (1907), p. 543. *Journ.* Chem., Met. and Min. Soc. S. Africa, Vol. VII (1906-07), p. 422. *Min. and Sci. Press*, Vol. XCIV (1907), p. 147. *Electrochem. and Met. Ind.*, Vol. V (1907), pp. 99, 146.

COMPARISON OF STAMPS AND LANE MILL

Size.	Wilfley Concentrates.		Wilfley Tailings.	
	Stamp System.	Lane System.	Stamp System.	Lane System.
	Per cent.	Per cent.	Per cent.	Per cent.
Through 30-mesh	97	99	90	96
Through 40-mesh	89	96	70	88
Through 60-mesh	58	92	42	66
Through 80-mesh	51	88	36	56
Through 100-mesh	22	80	14	36

Tube Milling at the Waihi Mine, New Zealand.[1]—The ore consists of hard chalcedonic quartz containing fine gold and was formerly crushed in stamps through 40-mesh screens. The stamp duty was 2.89 tons per day and the extraction was 88 to 90 per cent. of the gold and 74 to 78 per cent. of the silver.

Three Davidson 22-ft. tube-mills have been installed. Each mill is loaded with 5.5 tons of flint pebbles and requires 50 h.p. at 27.5 r.p.m. The pebble consumption is 600 ℔. per mill per week. Barry honeycomb liners costing about $175 each are now used in place of quartzite blocks which used to cost $400. The 90 stamps weigh 1000 ℔. each and crush 354 tons per day through 20-mesh screens. The pulp containing 10 parts water to 1 part sand is elevated to sizing boxes and the coarse sand from the spigots containing 2 parts water to 1 of sand and amounting to about 230 tons in 24 hr., flows to the three mills. Sizing tests are given in the table.

TEST OF DAVIDSON TUBE-MILLS

Size.	Feed.	Product.	Size.	Feed.	Product.
	Per cent.	Per cent.		Per cent.	Per cent.
On 30 mesh	5.32	0.03	Through 60 on 100-mesh	13.96	7.43
Through 30 on 40-mesh	9.77	0.12	Through 100 on 150-mesh	12.29	18.42
Through 40 on 60-mesh	15.94	1.13	Through 150-mesh	42.72	72.87

The cost of grinding in the tube-mills is 12.5c. for power, 14.0c. for flints and liners, 1.5c. for labor, repairs and stores, or a total of 28c. per ton ground, equivalent to 18.2c. per ton stamped. Each mill treats 80 tons per day.

The chief benefits from the tube-mills are: (1) Increased extraction amounting to 36c. per ton crushed. (2) Tonnage increased by 36 per cent. (3) Saving of 75 per cent. on the cost of screens, since the 20-mesh cost less and lasts longer than the 40-mesh. (4) Amalgamation improved by from 5 to 7 per cent. (5) The slime, owing to the contained fine sand, is

[1] E. G. Banks. *Bull.* Am. Inst. Min. Eng., Jan. 1907, p. 63. *Journ.* Chem., Metal. and Min. Soc. S. Africa, Vol. VII (1906-07), p. 348. *Min. Wld.*, Vol. XXVI (1907), pp. 306,446. *Min. Journ.*, Vol. LXXXI (1907), p. 287. *Mines and Minerals*, Vol. XXVII (1907), pp. 464,492. *New Zealand Mines Record*, Vol. X (1907), p. 477.

more easily treated. The reduction in milling cost is fully 12c. per ton and this added to the 36c. improved extraction represents a total saving of 48c. per ton stamped or $169 per day.

Tube Mill Liners.[1]—At the Komata Reefs a tube-mill, 16x4 ft., was lined with hard cast-iron segments with steel bars, 2x1 in., placed over each joint of the liners and bolted through the shell of the mill. This treats daily 70 tons of material from stamps having a 10-mesh screen. During nine months the liners have worn from 1¼ in. down to ¾ in. thick, and the steel bars have worn somewhat rounded on their front edges. It is estimated that the liners will grind 30,240 tons. They cost $450 or 1.4c. per ton of ore ground. Ten tons of flint pebbles were used in nine months which cost $252.33 or 1.6c. per ton ground.

At the Standard Consolidated Mining Company's mill at Bodie, California,[2] a 22-ft.x5-ft. Allis-Chalmers tube-mill was successively lined with (1) special steel plates, (2) pine blocks on end, (3) mountain mahogany blocks, (4) rough quartz blocks set in cement, (5) wrought iron bars. The last proved most satisfactory. The bars, 8x1-in. are placed lengthwise in 7-ft. and 15-ft. lengths and bolted through the shell. They last three to four months and are quickly changed. The mill grinds 50 to 75 tons in 24 hours.

Tube Mills at Guanajuato, Mexico.[3]—The size of the Abbe tube-mills used here is 4.5x20-ft. and each treats about 80 tons of pulp per day. Sizing tests of feed and product are as follows: On 40-mesh, feed 11.2 and product 0.5; through 40 on 50-mesh, 11.2 and 1.7; through 50 on 60-mesh, 8.9 and 2.9; through 60 on 80-mesh, 16.6 and 6.0; through 80 on 100-mesh, 16.3 and 16.2; through 100 on 120-mesh, 26.1 and 21.8; through 120-mesh, 9.7 and 51.2 per cent.

End thrust of the mill is opposed by angling the supporting rollers. Silex linings last eight months. The mill is kept a little more than half full of pebbles and the pebble consumption is about ¾ lb. per ton of ore ground. The power required to start the mill is about 60 h.p. but this drops to 43 as soon as the mill reaches its running speed.

Grinding Pans vs. Tube-Mills.[4]—Comparative tests were made at the Dolores mill, Chihuahua, Mexico. In test No. 1 the ore was crushed with cyanide solution by 15 stamps weighing 900 lb. each, through a 20-mesh screen. The pulp passed to two Bryan mills with 60-mesh screens, then to four pans in series, thence through two settlers and finally through two cone classifiers. The spigot discharge of the classifiers was ground in a tube-mill and then went back to the classifiers. In test

---

[1] S. D. McMiken. *New Zealand Mines Record*, Vol. XI (1907), p. 108.

[2] W. W. Bradley. *Min. and Sci. Press*, Vol. XCIV (1907), p. 17. *Journ.* Chem. Metal. and Min. Soc. S. Africa, Vol. VII (1907), p. 417. *Electrochem. and Met. Ind.*, Vol. V (1907), p. 101.

[3] C. W. Van Law. *Min. and Sci. Press*, Vol. XCV (1907), p. 205. *Electrochem. and Met. Ind.*, Vol. V. (1907), p. 422.

[4] Robert Clarke, *Min. and Sci. Press*, Vol. XCIV. (1907), p. 431. *Journ.* Chem., Met. and Min. Soc. S. Africa, Vol. VIII. 1907), p. 63.

No. 2 the pulp from the Bryans passed directly to the tube-mill and thence to the classifiers. The spigot discharge of the classifiers went through the pans and settlers and to the classifiers again. An average of the two tests showed that the tube-mill slimed 36.93 per cent. and the pans 43.02 per cent. of the coarse material in the feed, an advantage of 6.09 per cent. in favor of the pans.

The pans were Wheeler 5-ft. size with plain flat shoes and dies. They made 65 r.p.m. and required 12.5 h.p. each. The tube-mill was a sectional Allis-Chalmers, with inside dimensions 16.5 ft. x3.5 ft. It was lined with white iron and had a load of 4.5 tons of flint pebbles. It required 20 h.p. when run at 35 r.p.m. The wear of pans is considerable and would more than offset the advantage of sliming. The tube-mill also requires less power.

Tube-Mills in South Africa.[1]—Tube-mills serve a two-fold purpose in South Africa: First, to improve extraction, and second, to enlarge milling capacity. To show the saving by the use of tube-mills the following calculations are made. A mill of 100 stamps, crushing 500 tons, costs $187,500. A tube-mill with accessories costs about $13,000. It is shown that in September, 1906, the installation of 58 tube-mills had advanced the effective stamps from 3410, actually in use, to an equivalent of 3995 stamps, an increase of 585 stamps. This number of stamps at $1875 would cost $1,096,875; 58 tube-mills at $13,000 cost $754,000, showing a saving in favor of tube-mills of $342,875.

Owing to the newness of the installation the tube-mills are not working at their full efficiency so that the above figure should be somewhat larger.

Tube-Mills, Pebbles and Linings.[2]—Two tube-mills, each 22x5½-ft., lined with silex blocks, 6x6x4-in., were run under the same conditions except that one was loaded with pieces of banket and the other with Danish pebbles. Using 60-mesh and 90-mesh screens (holes 0.0098-in. and 0.0063-in. respectively) the following results were obtained from

COMPARATIVE RESULTS OF MILLS

Size.	No. 1 Mill.		No. 2 Mill.	
	Feed.	Product.	Feed.	Product.
	Per cent.	Per cent.	Per cent.	Per cent.
On 60-mesh	68.68	9.73	68.68	11.32
Through 60 on 90-mesh	19.57	26.28	19.57	27.24
Through 90-mesh	11.75	63.99	11.75	61.44

[1] *South African Mines*, Oct. 27, 1907. *Mines and Minerals*, Vol. XXVII (1907), p. 297. *Revue de Metallurgie*, Vol. IV (1907), p. 563.

[2] K. L. Graham. *Journ.* Chem., Met. and Min. Soc. S. Africa, Vol. VII (1906-07), pp. 317, 368. Vol. VIII, pp. 51,78. *Mines and Minerals*, Vol. XXVIII (1907), p. 126. *Electrochem. and Met. Ind.*, Vol. V (1907), p. 422. *Min. and Sci. Press*, Vol. XCV (1907), p. 555.

one month's run, during which No. 1 mill was fed with 171.5 tons of banket, and No. 2 mill with 10 tons Danish pebbles.

The advantage appears to be with the banket. There was indication that the banket caused less wear of the lining. Ten tons of pebbles cost $300 while the cost of 171.5 tons of banket was $85 and there was a further advantage that the banket was auriferous. The banket is fed automatically from a conveyor belt supplied with a plow to scrape off the necessary amount for each tube-mill. This falls into a hopper and thence finds its way into the tube-mill through a long-turn elbow 5½ in. diameter extending 3 in. into the hollow trunnion of the mill. This elbow has near its bottom a 1½-in. orifice for introducing the feed water. The ratio of ore to solids by weight is about 1 to 1. Any coarse pieces of banket finding their way into the discharge are caught on a little trommel, revolving with the tube-mill, and returned to the feed.

The author's experience indicates that silex linings are the best. The average cost of three linings on the 22x5-ft. mills, using silex blocks 6x6x4 in., was $775 and they last four months. A Davidson mill, 20x5 ft. at the Geldenhuis Deep Ltd. was relined in 2½ days.

In the discussion following this paper other instances were cited showing that banket was fully as efficient or even more so than pebbles. Three tube-mills taking 720 tons of pulp per 24 hr. from a 160-stamp mill using 17-mesh battery screens, require 23 tons of banket daily to keep them filled to the most efficient crushing point.

To obviate the increased wear of lining which occurs at a distance of 3 ft. from the feed end, a mill has been lined with blocks 8 in. thick for the first 7 ft., then 7½ in. for 1 ft. and finally 7 in. for the remainder. Such a lining wore through at a distance of 12 ft. from the feed end and from this point to the discharge end it was thin, while toward the feed end it was still 3 in. thick. There was a tendency for the large pebbles to accumulate at the discharge end.

Some difference of opinion exists as to the relative merits of silex linings and linings made of native chert. The claim is made that if the native chert is properly dressed and laid it will be equal to silex. One requisite for keeping wear of liners down to the lowest point is to maintain a full load of pebbles. Where a smaller amount of pebbles is used there is a tendency for the pebbles to slide instead of roll.

Theory of Crushing.[1]—The power required in crushing rock depends on the hardness of the rock, the lamination, the crystallization and the mineral aggregate. It varies directly as the amount of reduction although it takes a little more power to reduce ¼-in. lumps to ⅛-in. than ½-in. lumps to ¼-in. owing to the larger amount of fines formed in the former

---

[1] E. A. Hersam, *Min. and Sci. Press*, Vol. XCV (1907), p. 621.

case. The energy used in crushing is expended in the following principal ways: (1) In the transmission to and in the machine: (*a*) friction in the bearings, (*b*) friction in the flexible belts, drives, etc. (2) Between the fragments of ore: (*a*) Friction resulting in heat. (3) Within the fragments of ore: (*a*) Plastic deformation, (*b*) unrecovered elastic deformation, (*c*) breaking, (*d*) excessive breaking producing dust. All of the preceding except the actual breaking are causes of wasted work. The friction of the machinery may be lessened by proper care and good lubrication, and under favorable conditions is probably not over 10 per cent. of the total energy used. The loss of energy among the ore particles, either through pure friction or through deformation of the particles without rupture, makes itself evident in dust and heat. The time factor is important in crushing since the quicker the application of the crushing force the less the elastic deformation, and therefore, up to mechanical limits, high speed is more economical than low speed. Some machines take advantage of elastic deformation and others do not. In the former case when the particle of ore breaks the elastic deformation causes the particle to fly into several pieces. Machines which crush by direct pressure usually rupture the ore particle to the center while machines which act by grinding exert a force which is more tangential and the result is that the particle is abraded. Grinding is necessary, however, wherever fine crushing is desired.

The work required for crushing may be measured by the increased surface produced and in practice is determined through a comparison of the results of careful sizing tests made on the ore both before and after crushing. The surface on an irregular ore particle varies from 1.2 times to 1.7 times the surface of a cube having the same mass. The work required per square inch of surface produced varies with different ores and even with the same ore under different conditions. Calculations gave values of from 2 to 5 ft. lb., or even more.

To sum up, the fullest economy in crushing will be found in a speed that is sufficiently high for tough ore, an amplitude that is adequate for elastic ore and in a moderation of both speed and amplitude for brittle ore. Whenever the reduction in size is intended to be great the fine must be protected from further crushing either by water suspension, repeated sizing, or abundant opportunity for rearrangement of the compressed ore by the provision of interstitial space that accompanies uniform sizing It depends upon producing the effect of crushing rather than of grinding, particularly upon coarse sizes; and employing hard and non-elastic surfaces in contact with the ore. It demands solid foundations for machinery, care, cleanliness, and protection of bearings, tested lubricants, and the avoidance of all unnecessary irregularity of shape in high-speed running parts designed for the construction.

Wet Concentration.

Theory and Research.

Standard Screens.[1]—The sizes of the Institution of Mining and Metallurgy standard laboratory screens for use in making sizing tests and for the correlation of screens used in commercial and other work are shown in the accompanying table.

STANDARD SCREEN SIZES.

Mesh, or Apertures per Linear Inch.	Diameter of Wire.		Aperture.		Screening Area.
	Inch.	mm.	Inch.	mm	Per cent.
5	0.1	2.540	0.1	2.540	25.00
8	0.063	1.600	0.062	1.574	24.60
10	0.05	1.270	0.05	1.270	25.00
12	0.0417	1.059	0.0416	1.056	24.92
16	0.0313	0.795	0.0312	0.792	24.92
20	0.025	0.635	0.025	0.635	25.00
30	0.0167	0.424	0.0166	0.421	24.80
40	0.0125	0.317	0.0125	0.317	25.00
50	0.01	0.254	0.01	0.254	25.00
60	0.0083	0.211	0.0083	0.211	24.80
70	0.0071	0.180	0.0071	0.180	24.70
80	0.0063	0.160	0.0062	0.157	24.60
90	0.0055	0.139	0.0055	0.139	24.50
100	0.005	0.127	0.005	0.127	25.00
120	0.0041	0.104	0.0042	0.107	25.40
150	0.0033	0.084	0.0033	0.084	24.50
200	0.0025	0.063	0.0025	0.063	25.00

The proposed definition of mesh for laboratory screens requires that the wire be of the same diameter as the size of the hole. The advantages claimed are: 1. That definite ratio and corresponding arithmetical progression of both aperture and mesh is secured. 2. That by adopting a screening area of 25 per cent. the wires are absolutely "locked" in position, thereby preventing shifting and consequent irregularity in the size of aperture. 3. That, the ratio between wire and aperture in all sizes being constant, the angle of taper of the hole is also constant.

Velocity of Galena and Quartz Falling in Water.[2]—This paper contains the tabulated results of elaborate experiments on grains ranging all the way from 12.85 mm. diameter down to 0.03 mm. Above 0.28 mm. the rate of fall was noted directly in a sorting tube; below 0.28 mm. the elutriation method was employed. The data obtained has made it possible for the author to study the laws of settling more completely than he has ever done before. The results show that there is a critical point which divides the field into two parts. This critical point occurs at about 0.13 mm. diameter for grains of galena and 0.20 mm. for quartz.

[1] *Electrochem. and Met. Ind.*, Vol. V. (1907), p. 512. *Min. Reporter*, Vol. LV (1907), p. 215. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 526.
[2] Robert H. Richards, *Bull.* Am. Inst. Min. Eng. May (1907), p. 435.

Grains coarser than those at the critical point, falling freely in water are subject to the law of eddying resistance and their velocities are given by the formula,

$$V = C\sqrt{D(\delta - 1)}$$

where V is the velocity of fall in mm. per sec., D is the diameter of the grain in mm., δ is the specific gravity of the mineral and C is a constant. The value of C for quartz is 87 and for galena 100.

Grains finer than those at the critical point are subject to the law of viscous resistance and their velocities are represented by the formula

$$V = K(\delta - 1)D^2$$

where V, δ and D are the same as in the preceding formula, and K is a constant whose value for quartz is 424 and for galena 631. The critical point is not sharply defined and consequently there is a transition stage in which neither law applies exactly.

Inspection of the two formulas shows that for coarse grains of a given mineral, under the law of eddying resistance, the velocity of fall varies as the square root of the diameter, while for fine grains of the same mineral under the law of viscous resistance, the velocity varies as the square of the diameter. This explains the extremely slow rate of settling of very fine grains.

Settling of Sands in Water.[1]—A very complete study has been made upon the action of sands in classifiers and jigs. First the theoretical speculations of various authors are considered. Next are given the results of free settling experiments on (*a*) limestone alone, (*b*) magnetite and limestone, (*c*) blende and quartz, (*d*) magnetite and quartz, (*e*) galena and quartz. Free settling factor found for blende and quartz is 1.665; for the magnetite and quartz, 1.933; for galena and quartz, 2.984. Further experiments were made under hindered settling conditions to determine the effect of suction and interstitial currents upon the material which passes a jig bed into the hutch. For this purpose mixtures of granite and magnetite, of limestone and magnetite, and of pyrrhotite and hematite were used. In these experiments the factor between limestone and magnetite found in the hutch product varied all the way from 1.825 up to 6.065 the magnetite grains being always larger than the accompanying limestone. When suction was reduced to zero the factor for the ratio of magnetite to limestone in the hutch product is reduced to 2.332. From the foregoing the author concludes *that the interstitial currents are working in the same direction as downward currents* and the result is the placing of coarse heavy particles alongside of smaller

[1] G. G. Bring, *Jern-Kontorets Annaler* (1906), p. 321.

light particles. Quiet water or upward currents ordinarily always place fine magnetite and coarse limestone together but when downward currents work in the interstices of a jig bed just the opposite takes place. In the latter case the jig bed acts much the same as an ordinary sieve, sifting mixed sizes of heavy and light minerals. It is not possible to determine a factor for the product passing through a jig bed because the ratio of sizes of heavy and light grains varies with the conditions under which the jig is run. The net result of the foregoing experiments is to show that in quiet, or upward moving currents, hindered settling conditions prevail (see Richards, "Close Sizing Before Jigging." Am. Inst. Min. Eng. Vol. XXIV (1894), p. 409), which cause the coarse light particles to settle with the fine heavy particles, but in a filtering bed another set of conditions comes in which makes a product consisting of coarse light particles and fine heavy particles. The question now arises which of these conditions prevails in practice, both on jigs which make only a discharge product above the sieve and on jigs which make a hutch product through the sieve. To settle this question further experiments were undertaken. First a jig was studied which made only a discharge product. Fifteen kg. of 2.5-, 3-, 4-, 5-, 6-, 7-, 8- and 10-mm. sizes of granite were successively jigged with 15 kg. of similar sizes of magnetite on the same jig. In each test three successive discharge products were drawn from above the sieve. In all tests the first product amounting to 4 kg. was usually clean magnetite. The quality of the other two products of 3 kg. each is shown in the accompanying table.

GRANITE IN SECOND AND THIRD DISCHARGES.

Diameter of granite in mm.	Diameter of magnetite in millimeters.							
	2.25	3.00	4.00	5.00	6.00	7.00	8.00	10.00
	Per cent. granite in second discharge of 3 kilograms.							
2.25	1.63	2.30	2.87	9.98	11.10	19.93	22.09	40.82
3.00	1.81	3.48	3.76	10.97	14.45	18.17	23.64	41.34
4.00	1.47	6.05	4.05	9.06	16.63	19.30	23.76	29.10
5.00	2.93	4.52	3.82	10.20	18.50	18.35	21.97	30.38
6.00	1.66	4.34	4.86	9.53	12.93	18.08	23.43	30.28
7.00	2.52	3.11	1.94	9.21	10.86	15.95	18.02	30.98
8.00	2.88	2.75	2.96	7.12	14.83	19.71	15.95	26.21
10.00	1.43	0.55	0.75	8.64	13.16	9.58	14.30	18.37
	Per cent. granite in third discharge of 3 kilograms							
2.25	12.67	14.39	12.51	18.63	31.41	31.66	30.05	51.83
3.00	13.87	22.04	18.07	24.85	31.36	32.42	32.22	57.70
4.00	13.58	17.57	19.10	22.07	28.64	32.17	34.70	57.14
5.00	20.79	23.51	19.30	27.28	23.94	25.83	32.78	38.12
6.00	18.63	19.62	21.06	21.76	25.52	26.06	34.98	46.97
7.00	20.98	16.33	14.56	22.89	25.33	25.05	30.55	38.50
8.00	20.68	17.34	17.82	19.11	26.46	26.22	27.70	36.34
10.00	11.09	6.22	10.97	14.38	18.38	20.98	20.15	27.70

The table shows that when 2.25-mm. granite is jigged with successive sizes of magnetite the percentage of granite increases from 1.63 to 40.82 (see first line). This is due to the increasing size of the interstices. Further when 5-mm. magnetite is jigged with successive sizes of granite the percentage of granite remains about constant at 9 or 10 per cent. (see fifth column). The conclusion reached from this is that the upward stream and the downward insterstitial currents respectively are balancing each other. Further experiments on mixed sizes show that interstitial currents alone are not able to place the grains in this way but they need the aid of downward currents. In other words, *on a jig making a discharge product, both the conditions previously mentioned are acting, but the first (that of upwards currents) is predominating.*

A series of experiments made on a *jig delivering a hutch product* show that the *second condition (that of downward and interstitial currents) is predominating.*

All this work throws light on the question of sizing and shows that Rittinger's principle that coarse particles of gangue must be removed so that they will not come down with fine particles of heavy concentrates, is correct only for coarse jigs making discharge products. On the other hand with fine jigs making hutch products, sizing is done to get rid of the fine gangue particles which otherwise will be sucked down and cause a poor product. For the same reason, both from a theoretical and a practical point of view, it is correct to subject the feed for a jig producing a hutch product to hydraulic classification, because such a jig wants the gangue grains coarser than the grains of concentrates to do the best work.

It will be seen that it is impossible to compute a sieve scale to suit all conditions owing to the above mentioned variation in factor where downward and interstitial currents are acting. The author shows, however, how in any special case it is possible to determine a sieve scale which will yield a jig product of any desired richness. His method involves an experimental run on mixed sizes, a sizing test of the product and a chemical analysis of each product. Since the sizes of the product are less and less clean toward the fine end it is only necessary to calculate at what point the fine sizes shall be eliminated in order to leave a range of coarse sizes which shall be of any desired tenor in richness.

Woodbury Jigging System.[1]—In this system, a series of one-compartment jigs with head plungers having accelerated and retarded motions are used. The first is a classifier jig and removes the slimes by suitable overflow for further treatment. It also makes a concentrates discharge, a hutch product and a tailing product for the next jig. Before this material reaches the next jig it passes through an unwaterer and the water taken out is used in the hutch of the second jig. This arrangement economizes

[1] G. E. Edwards, *Min. Wld.*, Vol. XXVI (1907), p. 512. J. T. Glidden, *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1063.

water. The second jig and those succeeding make concentrates, hutch and tailings to the next jig. This system requires no previous sizing of material but treats natural products direct. These jigs are 48-in. wide, have screens 30-in. long, and have a capacity of from 200 to 500 tons per 24 hr.

On native-copper ores at Lake Superior a series of four jigs has a capacity of 200 tons. The first or classifier jig has a 24-in. screen and the three following jigs have 48-in. screens. The feed is crushed to $\frac{3}{16}$ or $\frac{1}{4}$-in. in steam stamps. The first jig makes clean slime to go to tables, hutch to go to tables, clean concentrates on the sieve and tailings to the second jig. The second jig makes clean concentrates, hutch to tables and tailings to the third jig. The third and fourth jigs are similar to the second and make hutches to tables, middlings to be recrushed and a final tailing to waste.

Comparative Work with Jigs and Ferraris Tables.[1]—These machines have been run side by side treating an ore containing galena, calamine, cerrusite and gangue. On material between 1.5 and 1 mm. a five-sieve bedded jig making 300 strokes of 4 mm. per minute, yielded lead concentrates with 60.5 per cent. lead, zinc concentrates with 50.14 per cent. zinc, and tailings with 0.27 lead and 1.76 zinc. On this same material a Ferraris jerking table making 320 strokes of 25 mm. per minute and sloping 4 per cent. gave lead concentrates with 42.50 per cent. lead and zinc concentrates with 41.50 per cent. zinc. The tailings contained 0.30 lead and 4.21 zinc.

Material between 1 and 0.5 mm. was also treated under the same conditions except that the jig stroke was 2.5 mm., the table strokes were 340 per minute of 20 mm. each, and the inclination was 3 per cent. The jig yielded lead concentrates with 41 per cent. lead, zine concentrates with 54 per cent. zinc and tailings with 0.20 lead and 2.76 zinc. The table yielded lead concentrates with 34 per cent. lead, zinc concentrates with 60 per cent. zinc and tailings with 0.17 lead and 4.81 zinc. In the preceding tests, middling products were made which were recrushed for treatment on slime tables.

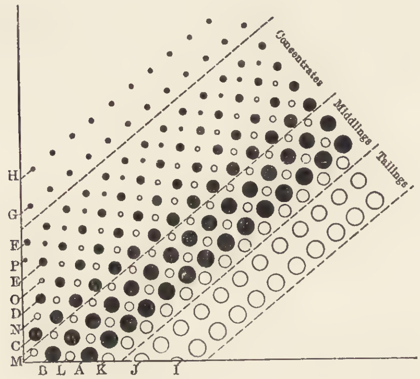
The tests show that jig has the advantage over the table on the sizes over 0.5 mm. Jerking tables give good results on stuff between 0.5 and 0.2 mm. Below 0.2 mm. slime tables are preferred.

The Wilfley Table.[2]—To determine the relative advantages of a natural-product feed, a sized-product feed and a classified-product feed an elaborate set of experiments was made, using artificial mixtures of pure galena and pure quartz. When treating a natural product, sometimes called mixed feed, that is, a product containing grains of both minerals ranging from a maximum down to dust, the concentrates contain considerable medium

[1] Mr. Lenicque. *Compt. Rend.*, Soc. de l'Ind. Min. (1907), p. 301.
[2] Robert H. Richards, *Bull.* Am. Inst. Min. Eng., July 1907, p. 627. *Min. Journ.*, Vol. LXXXII (1907), pp. 298, 343, 375 *Technology Quarterly*, Vol. XX (1907), p. 453.

sized quartz, the middlings are large in quantity and contain galena in the coarse and fine sizes, and the tailings contain fine galena. When treating sized or classified products, however, the concentrates and tailings are both practically clean and the middlings are small in amount and of such a character that they may be readily separated into concentrates and tailings by feeding them back upon the same table. Since a classified product already has the coarse quartz associated with fine galena it follows that the work of the Wilfley table is aided just so much and a classified-product feed therefore has an advantage over a sized-product feed. This is offset to a certain extent by the imperfections of mill classifiers which prevent the millmen from obtaining as good a quality of classified products as they can of sized products.

The accompanying figure shows the arrangement of grains on a Wilfley table treating a natural product. *A B C D E F G H* represent galena and



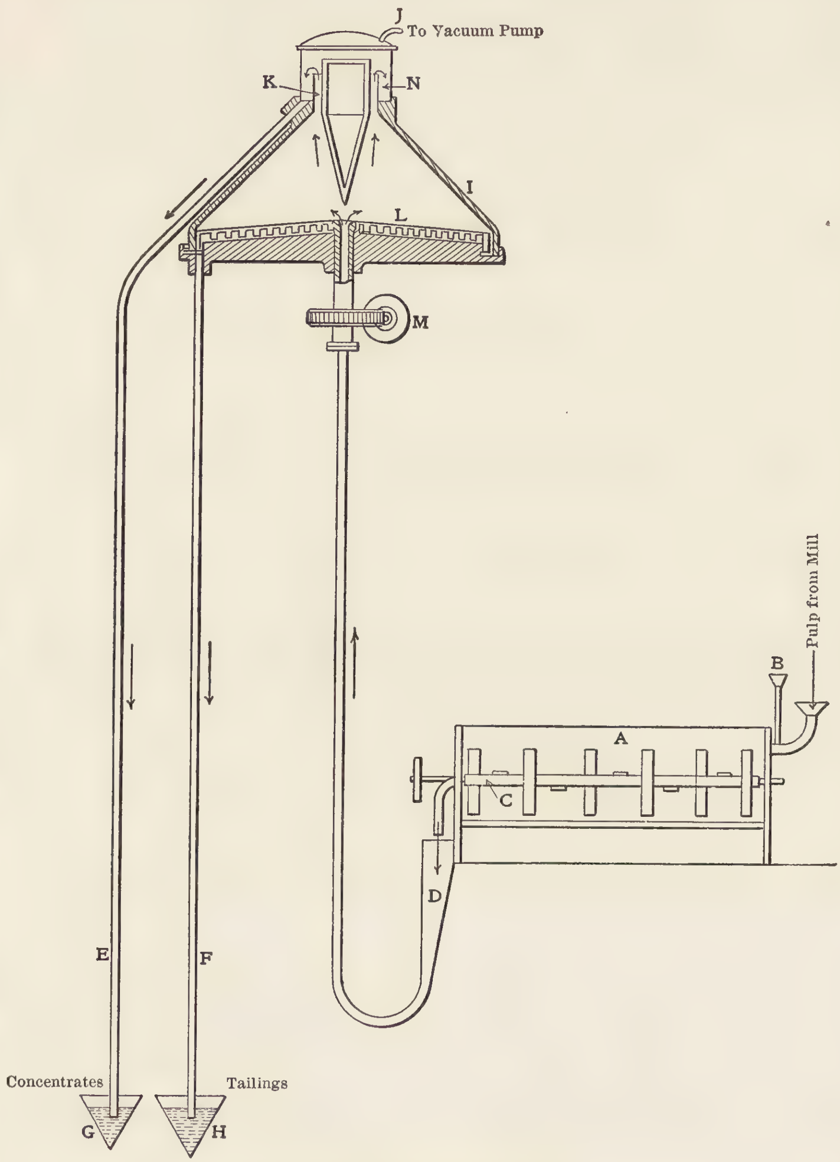
IDEAL SKETCH OF THE ARRANGEMENT OF GRAINS BY A WILFLEY TABLE.

I J K L M N O P represent quartz. The fine galena grains *G* and *H* are unable to withstand the wash water and to retain their position on the table, and therefore are washed down into the middlings, tailings and slimes. In the case of a sized-product feed, for example, a mixture of *C* and *K*, there should be a clean line of separation; similarly in the case of a classified-product feed, for example, a mixture of *C* and *O*, there should be an even better separation, provided good classification can be attained.

Elmore Vacuum Flotation Process.[1]—This process is based upon (1) the

---

[1] A. S. Elmore. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), pp. 908, 1110. *Mines and Minerals*, Vol. XXVIII (1907), p. 24. *Metallurgie*, Vol. IV (1907), p. 478. *Min. Journ.*, Vol. LXXXI (1907), p. 623. *Journ.* Chem. Met. and Min. Soc. S. Africa, Vol. VII (1907), p. 421. *Revue de Metallurgie*, Vol. IV (1907) p. 598. *Min. Reporter*, Vol. LV (1907), p. 537. *Genie Civil*, Vol. LI (1907), p. 144. *Bull. de l'Ind. Min.*, Series IV, Vol. VII (1907), p. 199.



ELMORE VACUUM CONCENTRATOR.

selective action of oil for metallic mineral particles which action is increased in some cases by the presence of an acid; (2) the fact that air or gases dissolved in water are liberated in a vacuum and attach themselves to the oil covered particles and help to float them.

In the accompanying figure, pulp from a crushing mill flows continuously into the mixer *A* into which is introduced small quantities of oil and, if required, acid also, at the point *B*. After agitation by the beaters *C*, the pulp flows continuously into the funnel *D*. The concentrates discharge pipe *E* and the tailings discharge pipe *F* are both sealed with water in the tanks *G* and *H*. Upon application of suction at *J*, pulp ascends from *D* to fill the conical separating chamber *I*. The downflow in the tailings pipe *F* is slightly less than the upflow from *D* so that there is a constant flow of concentrates over the annual lip *K* into the concentrates pipe *E*. The rakes *L* are caused to rotate slowly by the worm gear *M*, the angle of the teeth being such as to move the material toward the periphery. The pipe *D* is 25 to 30 ft. long and the pipes *E* and *F* are a few feet longer, thereby causing a siphon action. Glass windows are supplied to observe the overflow of concentrates at *K*.

The capacity of a unit with conical separating chamber of 5 ft. diameter is 35 to 45 tons in 24 hr. and the power required is 2 to 2.5 h.p. The oil consumption is 3 to 10 ℔. per ton of ore treated. For a plant treating 200 tons in 24 hr. the following costs would be representative for America. Oil, 7 ℔. per ton at 1c. per pound, \$0.07; acid (where required), 6 ℔. per ton at 1c., \$0.06; labor (at \$2.50 for 8 hr.), \$0.11; total, 0.24. To this must be added the cost of 0.05 h.p. per ton.

Concentrating Machinery.

New Century Disintegrating Screen.[1]—This is a wash trommel mounted in a tank of water. The screen may have holes varying from 50 mesh to 1½ in. Inside the screen are round bars 1 in. apart to help the disintegrating. On the outside of the screen are elevator buckets for spraying water down through the screen. Coarse washed material is discharged at the end while fine waste falls through the screen into the tank below and passes off in the overflow of this tank.

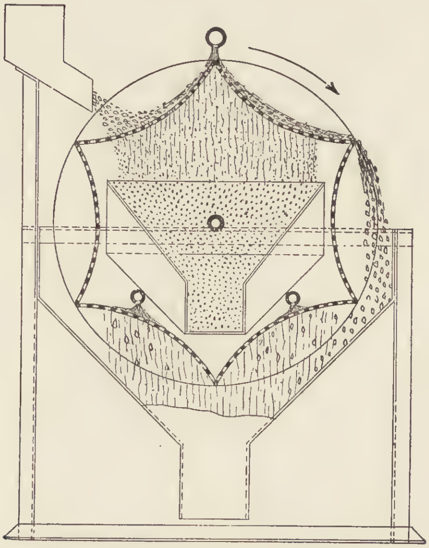
King Revolving Screen.[2]—This is a hexagonal trommel on a horizontal axis with outside feed, and wash water jets and hoppers inside for the undersize and outside for the oversize. The parts are well shown in the

---

[1] *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 846.

[2] *Min. Reporter*, Vol. LV (1907), p. 79. *Min. and Sci. Press*, Vol. XCIV (1907), p. 318. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 236. *Revue de Métallurgie*, Vol. IV (1907), p. 445.

figure. Advantages claimed for this apparatus over trommels are: (1) Low first cost. (2) Reduced wear of screen cloth. (3) Large, effective screening area. (4) Freedom from blinding. (5) Low power consumption.



SECTION OF KING ORE SCREEN SHOWING FLOW OF PULP.]

The Newaygo Screen.[1]—This is a fixed inclined screen enclosed in a dust-tight, steel housing. It is made in three sizes, all 6 ft. long on the incline and 4, 6 and 8 ft. wide respectively. The separating screen is overlaid with a coarser screen to save wear of the expensive fine screen. The screen cloth is automatically held taut and is tapped lightly by numerous hinged hammers attached to revolving shafts. These blows are received by reinforcing flat, iron strips attached to the screen cloth. The effect of the blows is to prevent clogging. Owing to the fact that the screen is inclined, the size of the undersize is considerably smaller than the size of the screen holes.

Humboldt Riffle Tables.[2]—The Humboldt new jerking table resembles the Wilfley table in having riffles and wash-water but adds another action which imparts to the wash-water side of the table a rising motion as it moves forward and falling motion as it moves backward. In other words, the table has a tilting motion at right angles to its longitudinal bumping motion. This table works well on fine sands and coarse slimes but not on fine slimes which were formerly treated on round tables.

The Humboldt quick jerking table is designed to treat these fine slimes.

---

[1] *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1120. *Iron Age*, Vol. LXXX (1907), p. 988.
[2] Mr. Garnatz, *Metallurgie*, Vol. IV (1907), p. 512.

The table top is supported on flat vertical springs or toggles and two bumps are given to the table at every revolution of the driving shaft or a total of 720 bumps per minute. The table has (1) a spring and bumping post arrangement, and (2) a pushing rod operated by an eccentric. The table is moved forward by the pushing rod, the spring being compressed at the same time. The reversal occurs and the table makes part of its backward motion under the action of the spring until its motion is arrested by the bumping post. This gives one bump. In the meantime the eccentric continues back and finishes its return stroke and starts again its forward stroke, continuing until the pushing rod catches up to the table, thereby giving a second bump. Then the cycle of operations is repeated. The table surface may be made single or it may be divided by a cross partition making thereby practically two tables driven by the same head motion. This arrangement means greater capacity with less power and less space. The rapid speed of bumps on these tables has the effect of settling very fine particles of concentrates between the riffles and delivering them at the concentrates end of the table. A double table requires about 60 liters of wash-water per minute and consumes from $\frac{1}{3}$ to $\frac{1}{2}$ h.p. These Humboldt tables have the diagonal ending of the riffles similar to the Wilfley table. They differ from the Wilfley in that they bump toward the head motion instead of away from it, and the concentrates come off the end nearer the head motion.

The Groppel Table.[1]—This is a table of the Wilfley type but differing in the driving mechanism, the method of suspension and the arrangement of the table top. The mechanism is the crank-arm variety. The table is suspended at its four corners by four thin, flat, spring rods. A heavy shaft supported on cast iron uprights at each end of the table extends longitudinally above the longer axis of the table and at each end of this shaft are two horizontal arms which are attached to the upper ends of the spring rods. Rotation of the shaft changes the slope of the table. The table top is 3 m. wide and 1.35 m. long. For an easy separation of one mineral from gangue this may be divided by a cross partition and the riffles and launders rearranged so as to make what is practically two tables each 1.5 m. long and 1.35 m. wide.

In a test on a table with an undivided top, installed at the Weiss mine at Bensberg, 4613.8 kg. of fine sand containing 9.88 per cent. zinc and 0.16 per cent. lead, both in the form of sulphides, yielded 555.1 kg. of zinc concentrates with 44 per cent. zinc and only 0.24 per cent. lead, 5.5 kg. of lead concentrates with 74.68 per cent. lead, and 1660.5 kg. of tailings with only 1.67 per cent. zinc; also rich middling products with 33.35 per cent. zinc and 27.48 per cent. lead, and poor middling with 7.03 per cent. zinc. The saving in the concentrates was 53 per cent. of the zinc

[1] C. Bloemeke. *Metallurgie*, Vol. IV (1907), p. 193.

and 55 per cent. of the lead. The table treated 419.4 kg. per hr. in this test. Its capacity varies from 400 to 600 kg. per hr. at a speed of 300 r.p.m. The throw varies up to 40 mm. The wash water required is 20 to 25 liters per min. and the power about 0.6 h.p.

The Deister Concentrator at the Baltic Mill.[1]—This is a riffle table of the Wilfley type but differs from the Wilfley in having (1) its greatest dimension at right angles to the line of the riffles and the direction of bump, (2) alternately long and short riffles, the long riffles extending across the whole width of the table, (3) high speed of 320 r.p.m. Material from the hydraulic classifier passes through two settling tanks in series. The products from gooseneck spigots of these settling tanks are treated on the tables; the overflow, carrying about 60 tons of dry material in 24 hr. goes to waste. Comparative tests of the Deister with the Overstrom table showed that the Deister saved 56.6 per cent. of the copper in the concentrates or 68.8 per cent. in the concentrates and middlings. The corresponding figures for the Overstrom were 53.1 and 65.2 per cent. respectively.

Traylor Concentrating Table.[2]—This is a riffle table of the Wilfley type. It is made in two parts hinged together. The feed end is kept level both lengthwise and crosswise of the table. This prevents the slimes from being washed straight across the table into the tailings. The concentrates end may be raised and lowered to get the concentrates to discharge at the desired point. The riffles are formed by grooves cut into the table top and the whole top is overlaid with a sheet of hard rubber fitting into the riffles. This catches and holds the valuable minerals better than any other smooth surface material.

Sherman Concentrating Table.[3]—This resembles the well known Wilfley slimer in that it has a series of wooden trays, each 5½ ft. by 11¼ in., which are fastened to sprocket chains and travel slowly forward in a direction at right angles to their length. They also receive a reciprocating motion in a direction paralled to their length. A pair of step pulleys serves to regulate the rate of travel and also the number of strokes (182, 226, 270, 314 or 358 per min.). The length of stroke may also be varied from ⅜ in. to 1¼ in. The feed is distributed over a width of 12 in. at the head of the trays. The trays thus loaded next pass past two bumpers situated at the head end 12 in. apart which serve to settle the heaviest particles to the bottom and stratify the pulp. Wash water is fed over the trays during the rest of their forward motion after they leave the feed box. After leaving the wash-water box they pass around the end of the frame thereby becoming inverted; concentrates are then washed off by jets of water and the trays

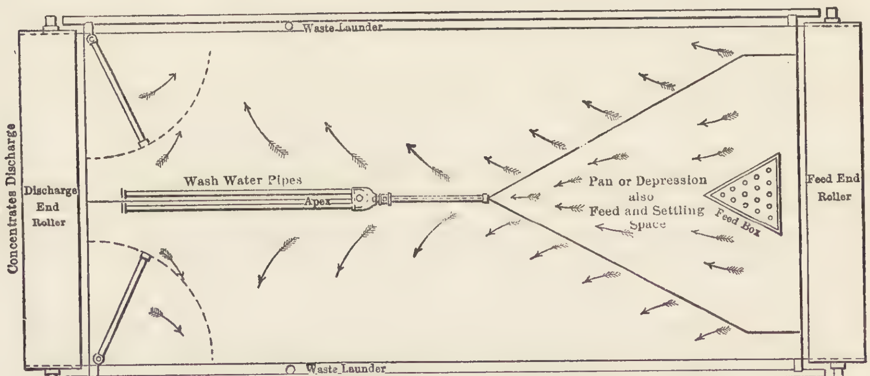
---

[1] L. S. Austin, *Min. and Sci. Press*, Vol. XCIV (1907), p. 827.
[2] *Min. Reporter*, Vol. LVI (1907), p. 498. *Min. and Sci. Press*, Vol. XCV (1907), p. 692. *Min. Wld.*, Vol. XXVII (1907), p. 979. R. Meeks, *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 968.
[3] C. E. Arnold. *Min. and Sci. Press*, Vol. XCIV (1907), p. 600. *Metallurgie*, Vol. IV (1907), p. 632.

return on the under side of the frame to pass up around the other end and reappear at the feed box. Two trays may be feeding and eight trays washing at one time. While being fed a tray is inclined about ⅜ in. per ft. toward the feed or closed end. After passing the bumpers the slope gradually changes until the final washing is done with the discharge or open end lower than the feed end. This facilitates the removal of the last of the gangue. A weighted idler on the driving belt has the effect of causing this belt to slip in case anything goes wrong with the sprocket chains and thus the table is stopped without doing any damage.

The capacity of the table is 7.05 tons of solids per 24 hr. The feed has 0.3 per cent. through 80- on 100-mesh; 6.4 per cent. through 100- on 150-mesh; 5.8 per cent. through 150- on 200-mesh and 87.5 per cent. through 200-mesh. The feed assays 2. per cent. lead and 9.83 oz. silver per ton; the concentrates 10.21 per cent. lead and 33.93 oz. silver; the tailings, a trace of lead and 3.16 oz. silver. Nearly all of the lead and 74 per cent. of the silver are saved with concentration of about five into one. Of the silver in the tailings, 31.55 per cent. is in the form of silver chloride which slimes so badly as to make its recovery practically impossible. Of the concentrates 98.6 per cent. are finer than 200-mesh.

Akins and Evans Slime Concentrator.[1]—This machine has an endless belt like the ordinary end-shake vanner but the belt surface is of canvas and, instead of being in one plane, has a triangular depression near the head end. This depression receives the feed and serves as a settling space. The base



DECK PLAN, A. & E. CONCENTRATOR

of this triangle corresponds to the head roll over which the concentrates pass. From the apex of the triangle, which is located nearly half way down the table, a ridge extends down the center line of the vanner to the tail roll.

[1] *Min. Reporter*, Vol. LV (1907), p. 206. *Min. and Sci. Press*, Vol. XCIV (1907), p. 350. *Min. Wld.*, Vol. XXVI (1907), p. 338. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 713. *Revue de Metallurgie*, Vol. V (1907), p. 105.

Vertically above this ridge is located the wash water pipe. The combination of wash water and slope forces the tailings toward the sides of the belt where they are caught in launders. The arrangement is well shown in the figures. The belt travel may be regulated at 26, 33, 43 and 56 in. per min. by means of step cone pulleys. The machine receives 225 end shakes per min.

Accessory Apparatus.

Cleaning Conveyer Belts.[1]—For removing wet and sticky materials jets of compressed air under 90-lb. pressure are directed against the under side of the belt just after it has passed over its discharge pulley. The jets are $\frac{1}{16}$ in. diameter and $\frac{1}{2}$ in. center to center extending across the full width of the belt. This device is in use at the Green Cananea mill in Mexico and the Old Dominion mill in Arizona.

Milling Practice.

Modern Tables.[2]—The greatest advances in the last decade have been in the development of bumping tables. The latest bumping tables are like the Rittinger in that the bump is at right angles to the slope and the flow of pulp. The Ferraris was the first table of the new type to be introduced in Germany. This was followed by the Overstrom and the Humboldt. The Overstrom riffle table has been in use at Raibl for a year. Its results are given in the accompanying table.

OVERSTROM TABLE AT RAIBL.

	Weight in Kg.	Per cent. of total Feed	Zinc.		Lead.	
			Per cent.	Weight Kg.	Per cent.	Weight Kg.
Feed per hour	123.50	100	24.9	30.751	5.5	6.792
Lead concentrates	6.283	5.09	12.8	0.804	66.0	4.147
Zinc concentrates	33.550	27.17	41.3	13.856	1.7	0.570
Total concentrates	39.833	32.26	47.67	14.660	69.45	4.717
Lead middlings	11.111	8.99	27.5	3.055	10.1	1.122
Zinc middlings	61.333	49.66	17.2	10.549	0.95	0.583
Total middlings	72.444	58.65	44.24	13.604	25.10	1.705
Total concentrates and middlings	112.277	90.91	91.91	28.264	94.55	6.422
Tailings	11.223	9.09	8.09	2.487	5.45	0.370

Water used in Milling.[3]—Stamp mills at Lake Superior use as much as 40 tons of water per ton of ore. The Bullion Beck & Champion mill in the Tintic district, Utah, used only 2.7 tons water per ton of ore and of this only 0.9 ton was fresh water, the balance being repumped from settling pond. At Washington, Arizona, a mill treating 100 tons of ore per day had

[1] *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 282. *Journ.* Chem., Met. and Min. Soc. S. Africa, Vol. VII (1907), p. 389.
[2] Ulrich Horel, *Oester. Zeit. f. B.-u. H.-wesen*, Vol. LV (1907), p. 197. *Revue de Metallurgie*, Vol. IV (1907), p. 596.
[3] *Eng. and Min. Journ.* Vol. LXXXIV (1907), p. 599.

216,000 gal. water in circulation or 9 tons of water per ton of ore, but required only 6500 gal. daily, or 3 per cent., to replace the loss. At Broken Hill, N. S. W., a mill used 1,647,000 gal. per day but the quantity supplied fresh was only 30,000 gal. or 2 per cent.

The Hercules Mill.[1]—This comparatively new mill represents the most approved lead-silver milling methods of the Cœur d'Alene district of Idaho. The ore is lowered from storage bins on the hillside to (1).

1. Ore bin of about 200 tons capacity. To (2).
2. Shaking grizzly with 2-in. spaces. Oversize to (3); undersize to (4).
3. Blake breaker, 12x15 in. To (4).
4. Bin of about 50 tons capacity. By shaking feeder to (5).
5. One No. 1 trommel, 9 ft. long, 4 ft. diameter. Sectional with two 20-mm. screens and one 36-mm. screen. Over 36-mm. to (6); undersize of 36-mm. to (11); undersize of 20-mm. to (7).
6. No. 1 or coarse rolls. To (7).
7. No. 1 bucket elevator. To (8).
8. One No. 2 trommel. Like No. 1 trommel except it has one 20-mm. screen and two 12-mm. screens. Over 20-mm. to (12); undersize of 20-mm. to (15); undersize of 12-mm. to (9).
9. Two No. 3 trommels, 7½ ft. long, 3 ft. diameter. Sectional with two 3-mm. screens and one 5-mm. screen. Over 5-mm. to (16); undersize of 5-mm. to (17); undersize of 3-mm. to (10).
10. No. 1 hydraulic classifier with three spigots. Three spigots to (18); overflow to (30).
11. One No. 1 bull jig with two sieves. 1st discharge concentrates; 2d discharge and both hutches to (13); tailings to waste.
12. One No. 2 bull jig. Products like (11).
13. No. 2 elevator. To (14).
14. Humphrey rolls. To (7).
15. Two No. 3 jigs with three sieves. 1st and 2d discharges concentrates; 3d discharge to (19); hutches to (24); tailings to waste.
16. Two No. 4 jigs. 1st and second discharges and 1st hutch concentrates; 3d discharge to (19); 2d and 3d hutches to (24); tailings to waste.
17. Two No. 5 jigs. Products like (16).
18. Three No. 6 jigs with two sieves. Hutches concentrates; tailings to (24).
19. Unwatering shaking screen. Oversize through bin and feeder to (20); undersize to (24).
20. No. 3 or Sturtevant centrifugal rolls. To (21).
21. No. 3 elevator. To (22).
22. No. 4 trommel with 1½-mm. holes. Oversize either to (23) or to (26); undersize to (30).
23. No. 4 or Sturtevant centrifugal rolls. To (21).
24. Fine bin. Settlings by spigots to (26); overflow to (25).
25. Series of flat settling tanks. Settlings periodically to (21); overflow used as wash water on jigs or to waste.
26. One 6-ft. Huntington mill. To (27).
27. No. 3 hydraulic classifier with three spigots. Spigots to (28); overflow to (21).
28. Four Wilfley tables. Concentrates; middlings to (29); tailings to waste.
29. Three Wilfley tables. Concentrates; tailings to waste.
30. No. 4 hydraulic classifier with six spigots. Spigots to (32); overflow to (31).
31. Two long, double rows of V-shaped settling tanks, making a total length of tank of over 500 ft. Coarse material to (32); fine material to (33).
32. Eleven Wilfley tables. Concentrates; middlings to (29); tailings to waste.
33. Five 6-ft. Frue vanners. Concentrates; tailings to (34).
34. One 6-ft. Frue vanner. Concentrates; tailings to waste.
35. All concentrates run to a storage bin where they are drained, and the water pumped back for use in the mill.

The mill treats about 200 tons in 24 hr. and yields about 40 tons of concentrates containing 50 to 60 per cent. lead and 50 oz. silver per ton. Average assays for three months are shown in the accompanying table.

HERCULES MILL PRODUCTS.

	Ounces Silver per ton.			Lead Per Cent.			Zinc, Per Cent.			Iron, Per Cent.		
	Sept.	Oct.	Nov.	Sept.	Oct.	Nov.	Sept.	Oct.	Nov.	Sept.	Oct.	Nov.
Mill feed	7.20	6.55	6.11	7.69	6.55	6.32	3.01	2.32	2.67	8.50	8.03	8.47
Coarse jig concentrates	50.44	51.44	52.33	60.15	60.14	63.54	......	......	......	......	......	......
Coarse jig tailings	1.20	1.43	1.21	0.76	0.70	0.57	......	......	......	......	......	......
No. 1 Wilfley concentrates	32.45	34.05	32.48	46.97	48.72	47.46	6.71	6.77	7.70	......	......	......
No. 2 Wilfley concentrates	27.12	32.86	29.76	35.78	45.62	40.40	9.73	6.61	8.81	21.04	18.07	19.14
No. 1 vanner concentrates	25.74	39.40	36.25	34.13	55.20	53.69	......	......	......	......	......	......
No. 2 vanner concentrates	......	34.26	22.22	......	32.50	20.00	......	......	......	......	......	......
General fine tailings	3.00	3.80	3.60	1.73	1.74	1.93	2.92	2.69	3.20	......	......	......
General fine concentrates	33.10	37.32	35.59	47.70	48.93	52.48	......	......	......	......	......	......

[1] Scott Turner. *Min. and Sci. Press*, Vol. XCIV (1907), p. 568.

At one time a zinc product containing 30 per cent. zinc and 12 per cent. lead was made on Nos. 3, 4 and 5 jigs but to do this it was necessary to make a lower-grade lead product and the scheme was abandoned. This mill makes high-grade jig concentrates which means a large middling product to be reground and slimed. It is figured that there was an extra loss of $600 per day in the tailings during a period of strike when green mill men were being broken in.

The mill is run by six 3-phase induction motors starting on 110 and running on 220 volts, power being brought on a 60,000-volt line from Spokane, about 100 miles distant. One motor of 15 h.p. runs the breaker; one of 25 h. p. runs the jigs, No. 3 trommels, No. 2 elevator, shaking screens, small pump, etc.; one of 50 h.p. runs No. 1 and No. 2 rolls, No. 1 and No. 2 trommels, No. 1 elevator, shaking feeder, etc.; one of 50 h.p. runs No. 3 and No. 4 rolls, Huntington mill and No. 4 trommel; one of 40 h.p. runs No. 3 elevator, tables, vanners and machine shop; one of 15 h.p. runs the Smith Vaile pump on the lower floor. During one month, while the mill was treating an average of 250 tons in 24 hr., the power consumed was about 106 kw. hours per hour or 142 h.p. hours of 14.2 h.p. per ton of ore. This includes the whole mill, breaker for first-class ore, mill lights, sample driers, pumps and machine shop. The mill is heated by steam from a wood-burning boiler. It is situated on a side hill with a drop of 125 ft. from the feed track to the lower pump floor.

The mill is run with three shifts of 8 hr. each. On each shift are a breaker feeder, jig man, Huntington man, vanner man and shift boss. On the day shift are also a repair man and helper, machinist and helper, one or two roustabouts, mill carpenter and general foreman. The mill men all get $3.50 per shift and the shift bosses $4.50.

Originally the mill contained Hodge Lake Superior jigs, impact screens and large unwatering wheels. The Hodge jigs gave difficulty owing to uneven work of the screens which were lively on the outside and dead nearest the plunger. This caused poor work and low capacity. Harz jigs have been substituted. Impact screens gave trouble from clogging with wood pulp. The unwatering wheels proved troublesome and wasteful of power. The Humphrey rolls are an innovation that has proved successful. The 6-ft. Huntington mill is too large for one man to handle easily and two 5-ft. mills would be better on this score, and at the same time in case of a break on one the other could be kept running.

Improvements in Milling Missouri Zinc Ores.[1]—In the present milling system the galena is mostly saved; the loss comes largely in the blende. Three forms of ore occurrence may be noted: (*A*) Coarsely crystallized blende which breaks free from the gangue in crushing. (*B*) Blende in hard black chert of high specific gravity which forms a large amount of

[1] W. E. Ford. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 868.

included grains in crushing. (*C*) Soft mud, sand and boulders containing small and irregular crystals of blende partially disseminated. The Missouri system of Blake breaker, Cornish rolls, roughing jig and cleaning jig was developed on ores of Class *A* to which it is admirably suited, and gives an extraction of about 85 per cent., but on ores of classes *B* and *C* this system is not able to make an extraction of over 65 per cent.

It is probable that improvements can best be made by adding to the present system rather than by a radical change to any new system. The first step should be to forecast the life of the mine and to test the ore by crushing, sizing, hand picking and hand panning. If the ore is found to belong in classes *B* or *C*, and the probable life of the mine will warrant it, it appears wise to elaborate the simple Missouri system along the following lines: (1) Careful, graded crushing down to 7 or 9 mm., avoiding the production of fines. (2) Sizing by 7-, 4-, and 2-mm. screens before jigging. (3) Roughing jigs for each size having each jig adjusted for the size which it is to treat. (4) Recrushing the roughing jig middlings by rigid rolls. (5) Concentration of the roughing jig concentrates on cleaning jigs. (6) Recrushing of the cleaning jigs tailings. This scheme reduces the formation of fines (below 1 mm.) and the same time removes them so that they will not be lost in jig tailings. These fines should be carefully classified and concentrated on tables.

Milling in Joplin District.[1]—In the typical mill the ore goes to (1).

1. Culling platform. Waste picked out to dump; residue to (2).
2. Shaking grizzly. Oversize to (3); undersize to (5).
3. Blake breaker. To (4).
4. No. 1 rolls. To (5).
5. Elevator. To (6).
6. Trommel. Oversize to (7); undersize to (8).
7. No. 2 rolls. To (8).
8. Rougher jig with six compartments; first two discharges concentrates; rest of discharges to (10); first hutch, lead concentrates; rest of hutches to (12); tailings sand to (9); tailings water to (14).
9. Tailings elevator to dump.
10. Chats elevator. To (11).
11. Chats rolls. To (13).
12. Elevator. To (13).
13. Cleaner jig with six compartments; first two discharges, concentrates; rest of discharges to (11); first two hutches lead concentrates; 3d and 4th, zinc concentrates; 5th and 6th to (15); tailings to (15).
14. Settling tank with three spigots. Spigots to (16); overflow to settling pond.
15. Settling tank with three spigots. Spigots to (16); overflow to settling pond.
16. Elevator. To (17).
17. Trommel. Oversize to (9); undersize to (18).
18. Tables. Lead concentrates; zinc concentrates; middlings to (16); tailings to (9).
    In some mills the sludge instead of going to (14) and (15) goes to (19).
19. Elevator. To (20).
20. Trommel. Oversize to (21); undersize to (22).
21. Rolls. To (19).
22. Unwatering tank. Settlings to (23); overflow to settling pond.
23. Sand jig with four compartments, 4th discharge to (21); four hutches zinc concentrates; tailings by elevator to (24).
24. Impact screen. To (25).
25. Tables. Zinc concentrates; tailings to (9).

The grizzlies (2) are usually of 2- or 3-in. round iron or worn mine rails spaced 5 to 6 in. apart. Both fixed and shaking forms are used. Breakers (3) are of the Blake jaw type from 6x14 in. to 12x24 in. and crushing to from $\frac{1}{2}$ to $1\frac{1}{2}$ in. Jaw plates and cheek plates are of cast-iron or steel, usually plain but sometimes corrugated. Rolls are of a local Cornish type.

[1] R. L. Herrick. *Mines and Minerals*, Vol. XXVIII (1907), p. 145.

No. 1 rolls (4) are most frequently 14x36 in. or 16x42 in., and the No. 2 rolls (7) 14x30 in. or 14x36 in. Some mills have No. 1 and No. 2 rolls of the same size for ease of changing parts. Chats rolls (11) are 14x24 or 14x30 in. Roll shells of cast iron or steel, made by local foundries, are cheaper in the end than manganese steel. Chilled iron costs 2½c. per lb. and lasts from two to three weeks, local steel costs 3c. and lasts five to six weeks, manganese steel costs 12c. per pound and lasts only 10 days longer than local steel.

Trommels are nearly all cylindrical 36x72 in. and 48x96 in. Both plate and wire cloth are used, the former preferred. The holes in No. 1 trommel (6) range from ⅛ to ⅝ in., and in the sludge trommel (20) from ½ to 2 mm. The number of compartments in the jigs ranges from four to seven, usually five or six. The compartments in the rougher jigs (8) vary from 34x46 in. to 36x48 in., while on the cleaner jigs (13) they are 30x42 in. The jig screens are cast-iron grates with slots $\frac{1}{16}$ to ⅛ in. wide.

Tables (18) and (25) are either the Kirk, Cooley, or Wilfley, the first two being of local manufacture. Elevators are of the usual bucket type on rubber belts. Concrete boots are used in one mill. Owing to the level ground tailings elevators (9) are necessary to lift the material in some cases to a hight of over 75 ft. Settling tanks vary greatly in size. Some of the newer mills have tanks of large capacity with two compartments each 12x20 ft. and 6 ft. deep. One compartment receives the jig sludge and lets the overflow go to the settling pond while its mate has its settled sludge flushed out with just enough water to take it to the elevator leading to the tables or sand jig.

Costs of milling for the small-capacity mills range from 45 to 70c. per ton, average about 60c. In the new, large mills on sheet ground the cost is about 33c. per ton. No. 3 mill of the American Zinc, Lead and Smelting Company, during the six months ending May, 1907, averaged 358 tons per day at a cost of 18c. per ton. No. 2 mill of the same company averaged 135 tons at a cost of 26c. Both treat ores averaging below 3 per cent. zinc.

One feature of the district is sludge mills which receive the sludge from mills that have no sludge machinery of their own. At the Murphy Friell sludge plant the sludge is shoveled from wagons to (1).

1. Bucket elevator. To (2).
2. Trommel with 1½-mm. holes. Oversize to an adjacent 200-ton tailings mill similar to the standard mill of the region minus the tables; undersize to (3).
3. Trommel with 1-mm. holes. Oversize like (2); undersize to (4).
4. Two fixed conical building buddles, run alternately, 18 ft. diameter, feed cone 6 ft. diameter, slope ¾ in. per ft. Six revolving arms each with a canvas sweep 2 ft. long. Holds 10 tons; concentrates shoveled to receiving tank and thence to (5); middlings shoveled to another receiving tank and thence separately to (5); tailings to waste.
5. Bucket elevator. To (6).
6. One buddle like (4). If feed is poor it makes concentrates and middlings to be rerun and tailings to waste. If feed is richer it makes concentrates to (7); rich and poor middlings separately to receiving tanks in (4); tailings to waste.
7. Two Wilfley tables. Mixed lead and zinc concentrates (about 2 tons per month) saved and separated on (5) and (6); zinc concentrates to market; tailings to (8).
8. Box. Heavier material shoveled out and run on (6); light material carried away by wash water to dump.

The crew for this plant consists of three shovelers per shift at $2 per day,

a table man at $2.25 and the mill boss. The daily output is about four tons from two shifts of 10 hr. each.

Colorado Zinc Ores.[1]—The Mogul mill at Gladstone, Colo., treats a low-grade complex ore containing a considerable quantity of zinc blende. The ore is crushed at the mine to about 2-in. size and is delivered by aerial tramway to (1).

1. Stamp mill bins. By automatic Boss feeders of the Challenge type to (2).
2. Forty stamps crushing to 20 mesh. Weight 1500 lbs. Drop 4 to 5 in., 104 drops per min. Double discharge mortars. Duty 6 tons per stamp per 24 hr. To (3).
3. Fifteen Card tables in series. Each table has a sizing arrangement which intercepts the coarse material in the pulp and passes the residue on to the next table. Beyond table 8 the feed is mostly slimes finer than 200 mesh. These tables make four products: A lead product by launder to unwatering apparatus and thence to Boss drying table from which it is delivered to shipping bins; zinc-iron product to (4); silica to (5); slimes to (7).
4. Five Card tables. Concentrates to (9); middlings to (6); tailings to waste.
5. Four Card tables. Concentrates to (9); middlings to (6); tailings to waste.
6. Two Card tables. Concentrates to (9); tailings to waste.
7. Unwatering tanks. Spigot to (8); overflow to waste.
8. Card and Wilfley tables. Concentrates to (9); tailings to waste.
9. Unwatering tanks. Settlings to Boss driers and thence by conveyor to (10).
10. Eight Blake-Morscher electrostatic separators. Zinc product; iron product.

Each Blake-Morscher machine has four separating rolls and treats four to six tons per day, or an average of about 40 tons for the whole set. The concentration is about 4 to 1, the product running about 36 to 40 and up to 43 per cent. zinc. The ore treated in the mill runs about 2.5 to 5 per cent. lead, 6 to 7 per cent. zinc and about 1.5 per cent. copper. The mill is run by electricity generated in its own power house from steam boilers and a DeLaval turbine of 300 h.p. running at 9000 r.p.m. and geared to two 100-kw. generators. A 40-h.p. motor runs the tables and driers. Five men per shift are required to run the mill.

Kimberly-Wilfley Mill.[2]—This new mill in process of erection at Kokomo, Colo., is designed to treat zinc-iron sulphides. The ore from the mines comes to (1).

1. Grizzly with ½-in. spaces. Oversize to (2); undersize to (3).
2. Blake breaker, 15x20 in. To (3).
3. Fine grizzlies. Oversize to (4); undersize to (5).
4. Rolls, 42x16 in. To (5).
5. Elevators. To (6).
6. Jeffrey screen with 14-mesh holes. Oversize to (7); undersize to (13).
7. Rolls, 36x12 in. By elevator to (8).
8. Jeffrey screen with 14-mesh holes. Oversize to (9); undersize to (13).
9. Rolls, 36x12 in. By elevator to (10).
10. Jeffrey screen with 14-mesh holes. Oversize to (11); undersize to (13).
11. Rolls, 36x12 in. By elevator to (12).
12. Jeffrey screen with 14-mesh holes. Oversize to (11); undersize to (13).
13. Belt conveyor 12 in. wide. To (14).
14. Storage bin holding 400 tons. By two Challenge feeders and two elevators to (15).
15. Wilfley roaster. Roasted product elevated to (16); flue-dust washed to (22).
16. Cooling cylinder, 4 ft. diameter, 30 ft. long,. By 12 inch conveying belt to (17).
17. Ten bins for roasted ore. To (18).
18. Ten Dings magnetic separators each treating one ton per hour. Iron product containing a large part of the gold and silver by 10-in. conveying belt to shipping bin; zinc blende, galena and gangue by 10-in. belt conveyor and elevator to (19).
19. Two Richards classifiers, 17 ft. long, with four spigots each. Spigots to (20); overflow to (22).
20. Eight Wilfley tables. Lead concentrates to (21); iron product to (21); zinc product to (21); middlings to (21); slimes to (22).
21. Eight Wilfley tables, two for each product of (20). These make clean products.
22. Buckingham filter and settling tank, 30x11x10. 9 ft. Coarse spigots to (23); fine spigots to (24); overflow to waste.
23. Two Wilfley tables. Iron product with some gold and silver; lead product with considerable silver; zinc product with 36 to 45 per cent. zinc.
24. Two Frue vanners with egg-shell belts. Products like (23).

Milling Porphyry Ore of Bingham.[3]—Owing to the lack of room in

[1] G. P. Scholl, *Mines and Minerals*, Vol. XXVII (1907), p. 498.
[2] *Min. Wld.*, Vol. XXVII (1907), p. 69.
[3] W. R. Ingalls. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 479. P. Barker. *Min. Reporter*, Vol. LVI (1907), pp 376, 536. L. H. Beason. *Min. and Sci. Press*, Vol. XCIV (1907), p, 474.

Bingham Cañon the ores from the Boston Consolidated and the Utah Copper Company's mines are sent 27 miles to Garfield, near the Salt Lake, at a cost of $0.27 per ton. The ore contains chalcocite and bornite with a considerable percentage of chalcopyrite, all finely disseminated through the porphyry gangue.

The Utah Copper Company's mill is built in 12 sections, each having a capacity of 500 tons in 24 hr. Six sections constitute a "unit." The two units are exactly alike and each has its coarse crushing department. The ore brought from the mine in railroad cars is weighed on scales and dumped to (1).

1. Coarse ore storage bin, 600 ft. long, 41 ft. wide, 28 ft. deep, flat bottom, holding 25,000 tons. To (3) direct or by 5-ton electric larries via bin to (2).

Only one unit is described.

2. Two grizzlies, 6 ft. wide, 10 ft. long, with 1.5-in. spaces. Oversize to (3); undersize to (4).
3. Two Gates breakers, No. 7½ K, crushing to about 1.5 in. To (4).
4. Two bucket elevators with 24-in. belt and 8x24-in. buckets. Speed 400 ft. per minute. Lift 58 ft. to (5).
5. Four trommels with 1.25-in. holes, 9 ft. long, 4 ft. diameter, sloping 2 in. per ft. and making 24 r.p.m. Oversize to (6); undersize to (7).
6. One pair of rolls, 20x54 in., 60 r.p.m. To (4).
7. Bucket elevator with 24-in. belt and 8x24-in. buckets. Speed 375 ft. per minute. Lift 59 ft. to sampler and thence to (8). There are three modified Vezin samplers in series with mixers between, each sampler cutting out 5 per cent. so that the final sample is 0.0125 per cent. or 0.25 lb. per ton of original feed.
8. Two portable 24-in. reversible conveyors, 150 ft. long, mounted on frames running on tracks to deliver to any point of (9).
9. Crushed ore bin for both units, 600 ft. long, 24 ft. wide, 25.5 ft. deep, hopper bottom, holding 15,000 tons. By gates to (10).

One section only is described.

10. Two plunger feeders. The plunger has a 4-in. stroke, 40 r.p.m., and works freely in a 9x11-in. box. To (11).
11. Elevator with 24-in. belt and 8x24-in. buckets, 360 ft. per minute, lift 54 ft. to (12).
12. Four trommels with 0.088-in. holes, 7.5 ft. long, 3.33 ft. diameter, slope 1½-in. per ft., 22 r.p.m. Oversize to (13) undersize to (14).
13. One pair of rolls, 15x37½ in., 80 r.p.m. To (11).
14. Two dewatering screens with 0.088-in. holes, 4.66 ft. long, 2.33 ft. wide, sloping 45 deg. Oversize to (15); undersize to (20).
15. Six jigs with two compartments each. Screens 20x30 in. with 0.088-in. slots. Hutches to (16); tailings to (17).
16. Four Wilfley tables. Concentrates to (28); middlings and tailings to (17).
17. Bucket elevator with 24-in. belt and 8x24-in. buckets, 400 ft. per min. Lift 58 ft. to (18).
18. Three feed tanks, sheet steel cylinders 10 ft. diameter and 8 ft. high. Spigots to (19); overflow to (23).
19. Three Chili mills with rolled slot cloth screens, 0.027-in. openings. To (20).
20. Two hydraulic classifiers each with four expanding hoppers in series. 1st spigot to (15); 2d to (15); 3d to (26); 4th to (22); overflow to (21) and (23).
21. Ten cone tanks, 9.5 ft. diameter with 60-deg. slope. Spigots to (24); overflow to (29).
22. Twenty Johnston 6-ft. vanners with corrugated belts. Concentrates to (28); tailings to (27).
23. Sixteen cone tanks, 7 ft. diameter, with a 60-deg. slope. Spigots to (24); overflow to (29).
24. Forty Johnston 6-ft. vanners with smooth belts. Concentrates to (28); tailings to (25).
25. Ten cone tanks like (21). Spigots to (22) and (26); overflow to (29).
26. Thirty-two Johnston 6-ft. vanners with corrugated belts. Concentrates to (28); tailings to (27).
27. Tailings launder to dump. Tailings are sampled automatically at the end of the launder by a swinging cutter. The sample is 0.06 per cent. or 1.2 lb. per ton. This is further reduced by a rotating cutter which takes a second sample of 3 per cent. of the first sample.
28. Centrifugal pump delivering by launder to 10 concrete concentrates bins, 15 ft. wide, 23 ft. long, 11 ft. deep with filter bottoms for drainage and overflow by launders from any bin to any other. Concentrates after draining are removed by clam shell grabs to cars to go to Garfield smelter. Drainings to (29).
29. Settling pond.

The original ore averages 2 per cent. copper, 0.015 oz. gold and 0.15 oz. silver per ton, 67 per cent. silica, 2.5 per cent. iron and 0.25 per cent. lime. The concentrates contain 25 to 30 per cent. copper, 20 to 23 silica, 17 to 20 iron, 1.5 per cent. molybdenum, $5 gold and 2.5 to 3 oz. silver per ton. The tailings contain 0.43 per cent. copper. The extraction is about 70 per cent.

A power plant supplies electric power for the mine and mill. The whole mill requires 4000 to 5000 h.p. of which 1050 h.p. is necessary for pumping water to a hight of 200 ft. One ton of ore requires six tons of

water. The mill building proper is 600 ft. square. The two coarse crushing sections are about 60 ft. square. The frame is of structural steel and the sides of corrugated iron. The roof is of 2-in. plank, shiplap, covered with corrugated iron. The floor is concrete surfaced with cement. The mill is built in steps on a side hill sloping moderately.

The cost of milling is about $0.35 per ton, of which the power is about $0.17. The average wages of all men employed in the mill is $2.60 per day. The cost of the mill was about $4,000,000 of which $1,000,000 was for power plant. Allowing for the part of the power plant that supplies power to the mine, the total cost of the mill was $3,540,000.

Giroux Mill at Ely, Nevada.[1]—This mill is designed to treat 500 tons in 24 hr. The ore is a friable porphyry containing chalcocite, chalcopyrite and pyrite. The ore comes to (1).

1. Receiving bin holding 600 tons. To (2).
2. Rock breaker, 10x24 in. By belt elevator to (3).
3. Trommel. Oversize to (2); undersize to (4).
4. Bin holding 200 tons. To (5).
5. Trommel with ¼-in. holes. Oversize to (6); undersize to (7).
6. One set of rolls, 42x16 in. To (7).
7. Belt elevator. To (8).
8. Trommels with 6-mesh and 16-mesh holes. Oversize to (9); undersize to (10).
9. One 6-ft. Huntington mill with 16-mesh screens. To (10).
10. Hydraulic classifiers. Products to (11).
11. Forty-five Wilfley tables and six Frue vanners. Concentrates to filter bin; tailings to waste.

Mine water is used, supplemented by a supply through a 12-in. pipe line 13 miles long.

Concentration of Cobalt Ores.[2]—The Coniagas concentrator for the treatment of low-grade cobalt and silver ores at Cobalt, Ontario, uses the following system:

1. Breaker. By No. 1 elevator to (2).
2. Bins. To (3).
3. No. 1 rolls. By No. 2 elevator to (4).
4. No. 1 trommel with $\frac{5}{16}$-in. and ¾-in. holes. Over ¾-in. to (5); through ¾-on $\frac{5}{16}$-in. to (8); through $\frac{5}{16}$-in. to (7).
5. No. 2 rolls. By No. 3 elevator to (6).
6. No. 2 trommel with ⅜-in. and $\frac{3}{16}$-in. holes. Over ⅜-in. to (8); through ⅜-on $\frac{3}{16}$-in. to (9); through $\frac{3}{16}$-in. to (7).
7. No. 3 trommel with $\frac{3}{16}$-in. holes. Oversize to (10); undersize to (12).
8. Coarse jigs. Concentrates to bins; middlings to (5); tailings to (11).
9. Medium jigs. Concentrates to bins; middlings to (11); tailings to waste.
10. Fine jigs. Product like (9).
11. Huntington mill. To (12).
12. Hydraulic classifier with two cones. 1st spigot to (13); 2d to (14); overflow to (15).
13. Wilfley table. Concentrates to bins; middlings returned to table; tailings to waste.
14. Frue vanner. Concentrates to bins; tailings to waste.
15. Callow tank. Settlings to (16); overflow to waste.
16. Frue vanner. Concentrates to bins; tailings to waste.

Mill of the Britannia Copper Company, B. C.[3]—The ore of this company contains copper and iron pyrites in a schistose siliceous gangue containing about 2 or 2½ per cent. copper. The first mill erected crushed the ore too fine so that there was a high loss in slimes; it is now being changed over to the following scheme, using gradual reduction.

1. Mill bins. By plunger feeders to (2).
2. No. 1 trommel with 2½-in. holes. Oversize to (3); undersize to (4).
3. Picking belt. Ore to smelter; waste to dump.

---

[1] W. R. Ingalls. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 675.
[2] *Canadian Min. Journ.*, Vol. I (1907), p. 276. Ralph Stokes, *Min. Wld.*, Vol. XXVII (1907), p. 306.
[3] R. Stokes. *Min. Wld.* Vol. XXVII (1907), p. 273.

4. No. 2 trommel with 1-in. holes. Oversize to (5); undersize to (7).
5. No. 1 jig. Concentrates to smelter; tailings to (6).
6. No. 1 rolls, 36x15 in. To (7).
7. No. 3 trommel with ½-in. holes. Oversize to (8); undersize to (9).
8. No. 2 jigs. Concentrates to smelter; tailings to (14).
9. No. 4 trommel with ¼-in. holes. Oversize to (10); undersize to (11).
10. No. 3 jigs. Concentrates to smelter; tailings to (14).
11. Hydraulic classifier. Spigot to (12); overflow to (13).
12. No. 4 jigs. Concentrates to smelter; middlings to (19); tailings to waste.
13. Wilfley tables. Concentrates to smelter; tailings to waste.
14. No. 2 rolls. To (15).
15. No. 5 trommel with ¼-in. holes. Oversize by elevator to (14); undersize to (16).
16. Hydraulic classifier. Spigot to (17); overflow to (18).
17. No. 5 jigs. Concentrates to smelter; middlings to (19); tailings to waste.
18. Wilfley tables. Concentrates to smelter; tailings to waste.
19. Chili mill. To (20).
20. Hydraulic classifier. Spigot to (21); overflow to vanners.
21. Wilfley tables. Concentrates to smelter; tailings to waste.

Milling at the St. Eugene Mine, B. C.[1]—The ore contains galena, pyrite, blende, garnet and a siliceous gangue. It is brought to (1).

1. Ore bins. To (2).
2. Grizzly with 1-in. holes. Oversize to (3); undersize to (4).
3. Blake breaker. To (4).
4. Screen feeder. Oversize to (5); undersize to (6).
5. Rolls, 12x36 in. To (6).
6. No. 1 trommel with 25-mm. holes. Oversize to (13); undersize to (7).
7. No. 2 trommel with 15-mm. holes. Oversize to (14); undersize to (8).
8. No. 3 trommel with 10-mm. holes. Oversize to (16); undersize to (9).
9. No. 4 trommel with 7-mm. holes. Oversize to (16); undersize to (10).
10. No. 5 trommel with 6-mm. holes. Oversize to (16); undersize to (11).
11. No. 6 trommel with 3-mm. holes. Oversize to (16); undersize to (12).
12. Hydraulic classifier. Spigot to (17); overflow to (22).
13. Four Harz jigs each with two compartments. Concentrates; middlings to (15); tailings to waste.
14. One jig with three compartments. Products like (13).
15. Rolls, 10x20 in. By elevator to (6).
16. Six jigs with two and three compartments. Concentrates; middlings to (18); tailings to waste.
17. Three jigs. Concentrates; middlings to (18); tailings to (24).
18. Three 5-ft. Huntington mills. By elevator to (19).
19. No. 7 trommel with 3-mm. holes. Oversize to (18); undersize to (20).
20. Hydraulic classifier. Spigot to (21); overflow to (22).
21. Three jigs. Products like (17).
22. Tank and settling boxes. Products to (23).
23. Five Wilfleys, 13 Frue vanners, four double-deck Wilfleys and one Deister table, ranging in order from coarse to fine. Concentrates; tailings to (24).
24. No. 8 trommel with 1.5-mm. holes. Oversize to (25); undersize to (26).
25. One 5-ft. Huntington mill. To (24).
26. Classifiers followed by six 4-ft. Frue vanners, two Wilfleys and two Deister tables.

During the first 5½ months of 1907 the mill produced 39,800 tons of concentrates from 212,700 tons of ore milled. The ore contains 4 per cent. zinc and the lead concentrates contain 6 per cent zinc. The recovery of values of lead and silver are about 12 per cent. and 6.5 or 7 oz. per ton respectively. The coarse jig concentrates run from 50 to 60 per cent. lead, the fine jig 60 to 70 per cent. and the slime concentrate (about 28 per cent. of the total output) 55 to 65 per cent. lead. The tailings contain about 1 per cent. lead and 3 to 3.5 per cent. zinc. About 1 per cent. of the ore is picked out as first-class ore to go direct to the smelter.

Proposed improvements are picking belt to remove shipping ore and Callow traveling belt screens to replace the hydraulic classifiers. The Deister table is much liked.

Working Costs on the Rand and in California.[2]—In discussion of costs

[1] R. Stokes. *Min. Wld.*, Vol. XXVII (1907), p. 967.
[2] Ross E. Browne, S. African Assoc. Eng., June 1, 1907. *Min. and Sci. Press*, Vol. XCV (1907), p. 113. *Min. Journ.*, Vol. LXXXI (1907), p. 867, and Vol. LXXXII (1907), pp. 12, 81, 146, 295. *Mines and Minerals*, Vol. XXVIII (1907), p. 75.

the accompanying table is given, comparing a normal Rand mine and the Oneida and Empire mines, California.

COMPARISON OF RAND AND CALIFORNIA MINES

	Rand	Oneida	Empire
Average vertical depth of mining, in feet	1,200	1,600	1,000
Number of stamps erected	131	60	40
Cost of plant (equipment)	$2,125,000	$252,700	$227,000
Cost of plant per stamp	16,220	4,200	5,660
Percentage of assay value actually recovered by free amalgamation	55	61	67
Percentage of assay value actually recovered by cyaniding	34	....	....
Percentage of assay value actually recovered by concentraion	....	26	14
Percentage of assay value actually recovered—total	89	87	91
Average width of stopes in inches	69	71	48
Tons of ore developed per month	16,000	10,000	4,000
Tons of ore mined per month	17,779	7,640	3,090
Percentage sorted out as waste	13.16	....	....
Tons of ore milled per month	15,654	7,640	3,090
Average number of stamps running	111	54	40
Number of days run per month	29.00	29.75	....
Weight of stamp in pounds	1,137	1,050	900
Average stamp duty in tons per 24 hours	4.85	4.78	2.53
Average dip of lode or vein	30°	65°	28°
Yield per ton milled	$8.76	$3.36	....
Mine water pumped, Imperial gallons per 24 hrs	163,149	58,408	750,000
Mine water pumped, Imperial gallons per ton milled	303	226	7,500
Percentage of ore cyanided	100	....	....
Percentage of concentrates produced	....	1.4	....
Percentage of sulphides in ore	2.5	1.5	2.4
Working costs per ton of ore milled	$5.32	$2.73	$4.68
Working costs per ton allowing for waste sorted out	$4.91	$2.67	$4.51
Estimated costs if California mills as large as Rand	$4.91	Under$2.50	Under$3.75
Persons employed per ton of ore	1.95	0.54	1

Milling at Broken Hill, N.S.W.[1]—The ores of this district contain galena, blende, rhodonite, quartz, calcite and garnet.

The crude ore contains from 10 to 19 per cent. zinc and the zinc in the concentrates lies between 6 and 10 per cent. Costs of milling run very uniformly between $1 and $1.10 per ton.

At the Proprietary mill the ore is dumped upon (1).

1. Two grizzlies with tapered manganese steel bars lasting about a year. Oversize to bin holding about 400 tons and thence to (2); undersize to (3).
2. Five No. 5 Gates breakers crushing to 2 in. These have tough steel heads lasting 1350 hr. and manganese steel liners lasting about 2700 hr. This combination gives good gripping properties which would not be present if both parts were of manganese steel. To (3).
3. Bins. By cars and hydraulic elevator to bins and thence by three roller feeders to (4).
4. Three pairs of No. 1 rolls, 36x15 in., set $\frac{3}{8}$ to $\frac{1}{2}$ in. apart, with steel shells, making 37 r.p.m. A scoop sample is here taken by hand from the feed and careful tests show this sample to be accurate. The best material for roll shells is toughened steel, $4\frac{1}{2}$ in. thick, which lasts approximately 3000 hr. The rolls are shifted sideways occasionally to promote even wear. Crushed product to (5).
5. Three shaking screens with $\frac{3}{32}$ in. holes. Oversize by belt conveyor and belt elevator, to bins and thence by three roller feeders to (6); undersize to (11).
6. Three pairs of No. 2 rolls like (4) except they are $\frac{1}{8}$ to $\frac{3}{16}$ in. apart and make 45 r.p.m. To (7).
7. Three shaking screens with $\frac{3}{32}$ in. holes. Oversize by belt conveyor and bucket elevator to bins and thence by two roller feeders to (8); undersize to (11).
8. Two pairs of No. 3 rolls like (4) except they are $\frac{1}{16}$ in. apart and make 78 r.p.m. To (9).
9. Two shaking screens with $\frac{3}{32}$ in. holes. Oversize by belt conveyor and elevator to bin and thence by roller feeder to (10); undersize to (11).
10. One No. 5 Krupp wet ball-mill with $\frac{3}{32}$ in. screen. To (11).
11. One cone hydraulic classifier 18 in. diameter, 15 in. deep. Spigot to (12); overflow to (21).
12. Coarse jigs of the May type each with four compartments. 1st and 2nd hutches, lead concentrates with 60 per cent. lead to (29); 3rd and 4th hutches, middlings with 10 per cent. lead, 19 per cent. zinc and 10 oz. silver per ton to (14); tailings with 4 per cent. lead, 12 per cent. zinc and 5 oz. silver to (13).
13. Shaking screen, 40-mesh. Oversize to bins and thence by cars to dump for future treatment; undersize to zinc plant.

[1] G. Delprat, *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 317. Austral. Inst. Min. Eng., Vol. XI (1906), p. 164. R. Stokes, *Min. Wld.*, Vol. XXVI (1907), pp. 300, 326, 561, 592. *Min. and Sci. Press*, Vol. XCIV (1907), p. 407. *Genie Civil* Vol. L (1907), p. 318. *Bull.* de l'Ind. Min., Series IV, Vol. VI (1907), p. 525. *Revue de Metallurgie*, Vol. IV (1907), p. 471

14. One No. 4 Ball-mill with $\frac{3}{32}$ in. slotted screen. By bucket elevator to (15).
15. Cone hydraulic classifier. Spigot to (16); overflow to (21).
16. Intermediate jigs with four compartments. 1st and 2nd hutches, lead concentrates with 40 per cent. lead, 13 per cent. zinc and 22 oz. silver to (29); 3rd and 4th hutches and tailings with 9 per cent. lead, 18 per cent. zinc and 8 oz. silver, to (17).
17. One Ball-mill with $\frac{1}{16}$ in. slotted screens. By bucket elevator to (18).
18. One cone hydraulic classifier. Spigot to (19); overflow to (21).
19. Fine jigs with four compartments. 1st and 2nd hutches, lead concentrates like (16); 3rd and 4th hutches middlings to (17); tailings with 5.5 per cent. lead, 18 per cent. zinc and 8 oz. silver, to (20).
20. Shaking screen, 40-mesh. Oversize to bins and thence by cars to waste dump for future treatment; undersize to zinc plant.
21. Spitzkasten with five pockets. 1st spigot to (22); 2nd, 3rd, 4th and 5th to (24); overflow to (28).
22. Wilfley tables. Lead concentrates to (29); middlings to (23); tailings to (27); slimes to (28).
23. Wilfley tables. Lead concentrates to (29); tailings to (27).
24. Luhrig vanners. Lead concentrates to (29); middlings to (25); tailings to (27); slimes to (28).
25. Claassifier. Spigot to (26); overflow to (28).
26. Luhrig vanners. Lead concentrates to (29); tailings to (27).
27. Spitzkasten. Spigot to zinc plant; overflow to (28).
28. Settling tanks for slimes. These slimes contain about 18 per cent. lead, 17 per cent. zinc and 18 oz. silver. After settling they are sintered before going to the smelting plant.
29. Concentrates draining bins. By cars to smelter.

The foregoing description applies to the new mill treating 6000 tons in six days. The old mill has a like capacity and is similar in arrangement.

The ore runs about 15 per cent. lead. The recovery of lead on the course jigs is 47.5 per cent.; on the fine jigs, 13.5 per cent.; on Wilfley tables 3.15 per cent.; on Lührig vanners, 3.15 per cent. The hourly capacity of a double coarse jig is about $5\frac{3}{4}$ tons; double fine jigs, $2\frac{3}{4}$ tons; Wilfley table, $\frac{3}{4}$ ton; Lührig vanner, single belt, $\frac{1}{6}$ ton. There is a selective action in the crushing, the sulphides crushing more easily, so that the feed to No. 1 rolls runs about 15 per cent. lead while that to No. 3 rolls or the ball-mill runs only about 10 per cent. lead. No material is returned again to the same crushing machine.

Trommels were formerly used but were discarded for the following reasons: (1) They passed only 58 per cent. of the material that belonged in the undersize, while shaking screens pass 70 per cent. (2) Shaking screens cost less, require less water, are repaired more quickly, take less room, are more accessible and the screen plate lasts longer (six weeks life on a screen area of 21 sq.ft.). There is no sizing before jigging. Tests have shown that it gives no higher recovery on this ore.

The coarse tailings from the two mills amount to 3500 tons per week; the lead concentrates 2200 tons per week; the fine tailings to the zinc plant, 4800 tons per week; the slimes, 1500 tons per week. The concentrates and slimes contain about 80.5 per cent. of the total lead in the original ore. Considerable scum forms on the water in the mill and this assays 60 to 80 oz. silver against 25 to 30 oz. for ordinary concentrates.

The Broken Hill Proprietary Company has treated over 400,000 tons of accumulated zinc tailings and obtained about 110,000 tons of zinc concentrates. The Delprat process is used in which the tailings are fed into the lead-lined flotation box which is an inverted pyramid with two pockets at the bottom. One of these is for any metal, stones or other heavy foreign matter, and the other is an outlet. A solution of sodium sulphate (salt cake) is fed into the box in an amount in excess of that

required for the bottom outlet so that there is a constant overflow. Superheated steam is used to raise the temperature of the liquid to 180 deg. Bubbles of carbonic acid gas are generated which lift the sulphides to pass out the overflow while the gangue flows out the spigot below and is carried away to waste on a conveyer belt. The overflow passes through a launder, where the bubbles of gas are detached from the sulphides by the splashing and thence goes to a settling vat where a zinc product, averaging 41 per cent. zinc, is settled out and washed. There are six flotation boxes, each treating 10 to 12 tons per hour, and six settling vats each holding 50 tons and used in rotation. The output is about 240 tons of concentrates daily. Experiments are being made with magnetic separators to divide the zinc concentrates further into a rich zinc product and a rich lead product. Regrinding is necessary before this salt cake treatment.

The Central mine has two plants. The old plant treats tailings by Mechernich magnetic separators and the plant has a capacity of 900 tons per week. The new magnetic plant treats about 1200 tons weekly. The material is ground to 2 mm. in ball-mills and the dust is drawn off by wind classifiers. The ball-mill product is screened into three sizes: 2 to 1 mm., 1 to ½ mm. and less than ½ mm. There are five upper and three lower Mechernich magnetic separators for the coarse, eight upper and four lower for the medium, and four upper and two lower for the fine. A quartz-lead product, zinc concentrates and rhodonite tailings are made as in the old plant by dry electric concentrators. The quartz-lead product containing 7 per cent. zinc, 8 per cent. lead and 6 to 7 oz. silver, goes to spitzkasten, jigs and tables. The jigs yield lead concentrates with 60 per cent. lead, zinc concentrates with 39 per cent. zinc and tailings with 6 per cent. zinc, 1 per cent. lead and 1 oz. silver. The tables yield lead concentrates with 72 per cent. lead, 4 per cent. zinc and 22 oz. silver, zinc concentrates with 39 per cent. zinc, 16 per cent. lead and 15 oz. silver, and tailings with 3 per cent. zinc, 1 per cent. lead, and 1 oz. silver.

The Central mine has also an oil-separation plant having a capacity of 1500 tons per week. The tailings containing 21 or 22 per cent. zinc, 5 per cent. lead, and 6 oz. silver, are reduced to ½ mm. in ball-mills and grinding pans, the latter being the most economical and efficient machines. The ground mineral is elevated and fed into the first of six agitation boxes. In the first box sulphuric acid (about 12 lb. per ton) is added to cleanse the particles. In the next four boxes, connected by holes near the bottom of the partitions, the charge is further agitated with oelic acid (2 to 3 lb. per ton) and in the sixth box with steam. Sulphides are floated. The discharge from the agitation boxes runs into a spitzkasten from which the overflow of oil and suspended sulphides passes into cloth-bottomed settling vats, and the underflow passes to a second spitzkasten where the separation is repeated. The concentrates average 46 to 49

per cent. zinc, 8 to 9 per cent. lead and 13 to 15 oz. silver. The tailings average 6 to 7 per cent. zinc, 2½ per cent. lead, and 2½ oz. silver. The cost of treatment is $1.95 per ton. It is proposed to treat the concentrates with caustic soda to remove the oil and take out the lead on tables.

The North mine tailings are to be treated by the DeBavay process, the principal of which involves the dry agitation of the material with carbonic acid gas followed by a separation in water where the sulphides float and the gangue sinks. This same process is to be used for slimes by the Central company.

Central Ore Dressing Plant at Clausthal.[1]—A new plant with a capacity of 360 tons in 10 hr. was built in 1904-5. Four kinds of ore are treated. The Oberes Burgstädter ore contains 4.17 per cent. galena, and 14.12 per cent. blende; the Unteres Bergstädter ore contains 4.93 per cent. galena, 20.95 per cent. blende, and 0.23 per cent. chalcopyrite; the Rosenhofer ore contains 7.83 per cent. galena and 10.51 per cent. blende; the Zellerfelder ore contains 10.34 per cent. galena. The gangue is calcite, quartz, graywacke and slate and, in the Rosenhofer district, also spathic iron in very small quantities. These ores vary in fineness of dissemination. The Unteres Bergstädter ore yields concentrates as coarse as 11 mm. while the Oberes Burgstädter and the Rosenhofer ores require crushing to 4 mm. The ores are treated separately, so that large storage bins are required for the accumulation of one ore while another is being run. The blende is very brittle, so that above 2.5 mm. rolls are used for crushing and roller mills below this point. The mill is divided into two main sections for different ores. Furthermore the slime department is divided into several sections to treat the various materials coming from different parts of the mill. The ores come to receiving bins of 2900 tons capacity and are drawn off into cars and elevated to one or the other of the two sections. The following describes the first section, with a capacity of 180 tons in 10 hr.:

(*a*) COARSE CRUSHING DEPARTMENT.

1. Car Tipple. To (2).
2. Grizzly with 100-mm. spaces. Oversize (20 per cent of the total) to (3); undersize to (6).
3. Spalling platform. Ore to (4); ore attached to waste to (5); waste to dump.
4. Breaker crushing to 65 mm. By car to elevator to (1).
5. Cobbing table. Ore by car to elevator to (1); waste to dump.
6. Hopper. By feeder to (7).
7. Trommel, supplied with water, with 50- and 32-mm. holes. Over 50 mm. to (8); through 50 on 32 mm. to outer ring of (10); through 32 mm. to (21).
8. Picking table 5 m. diameter. Cobbing ore to boxes to be cobbed; waste to dump; residue to (9).
9. Fine breaker. To (13).
10. Double picking and cobbing table 7 m. diameter Outer ring makes galena, blende and siliceous shipping ores, waste to dump and residue to (11). Inner ring makes similiar products except residue to (12).
11. Coarse rolls, 1000 mm. diameter. To (13).
12. Medium rolls, 1000 mm. diameter. To (13).

(*b*) ROLLS DEPARTMENT.

13. Elevator. Ore to (14); overflow water to (63).
14. Trommel with 32-mm. holes. Oversize to inner ring of (10); undersize to (15).
15. Trommel with 22-, 16- and 4-mm. holes. Over 22 mm. to (17); through 22 on 16 mm. to (18); through 16 on 4 mm. to (16); through 4 mm. to (23).
16. Three trommels in series with 11-, 8- and 5.6-mm. holes. Over 11 mm. to (18); through 11 on 8 mm. to (19); through 8 on 5.6 mm. to (19); through 5.6 mm. to (20).

[1] Schennen. *Glückauf*, Vol. XLIII (1907), p. 657. *Min. Journ.*, Vol. LXXXI (1907), p. 861.

17. One three-compartment jig. 1st discharge, lead concentrates to (31); 2d, rich middlings to (35); 3d, poor middlings to (46); tailings to (33).
18. Two three-compartment jigs. 1st discharge, lead concentrates to (32); 2d, rich middlings to (35); 3d, poor middlings to (46); tailings to (58).
19. Two three-compartment jigs. 1st discharge, lead concentrates to smelter; 2d, rich middlings to (35); 3d, poor middlings to (46); tailings to (58).
20. One four-compartment jig. 1st discharge, lead concentrates to smelter; 2d, rich middlings to (35); 3d and 4th, poor middlings to (46); tailings to (58).

(c) MINES FINES DEPARTMENT.

21. Trommel with 22-, 16- and 4-mm. holes. Over 22 mm. to (25); through 22 on 16 mm. to (26); through 16 on 4-mm. to (22); through 4 mm. to (23).
22. Three trommels in series with 11-, 8- and 5.6-mm. holes. Over 11 mm. to (26); through 11 on 8 mm. to two jigs (27); through 8 on 5.6 mm. to two jigs (27); through 5.6 mm. to one jig (27).
23. Three trommels in series with 2.8-, 2- and 1.4-mm. holes. Over 2.8 mm. to (28); through 2.8 on 2 mm. to (28); through 2 on 1.4 mm. to (29); through 1.4 mm. to (24).
24. Hydraulic classifier with four spigots. Spigots to (30); overflow to (63).
25. One three-compartment jig. 1st discharge, lead concentrates to (31); 2d, rich middlings to (35); 3d, poor middlings to (46); tailings to (33).
26. Four three-compartment jigs. 1st discharge, lead concentrates to (32); 2d, rich middlings to (35); 3d, poor middlings to (46); tailings to (58).
27. Five four-compartment jigs. 1st discharge, lead concentrates to smelter; 2d, rich middlings to (35); 3d and 4th, poor middlings to (46); tailings to (58).
28. Four four-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, poor middlings to (46); tailings to (58).
29. Two five-compartment jigs. 1st hutch lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th zinc concentrates to smelter; 5th, poor middlings to (46); tailings to (58).
30. Four five-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, zinc concentrates to smelter; 5th, poor middlings to (58); tailings to waste.
31. Cobbing table. Lead ore to smelter; rich middlings to (35).
32. Cobbing bench. Lead concentrates to smelter; rich middlings to (35).
33. Cobbing table. Poor middlings to (46); waste to dump.
34. Four three-compartment jigs to treat the three sized products and the classifier product. 1st hutch, lead concentrates to smelter; 2d, rich middlings to (35); 3d, zinc concentrates to smelter; tailings are poor middlings to (46).

(d) RICH MIDDLINGS DEPARTMENT.

35. Bucket elevator. Ore to (36); overflow water to (66).
36. Trommel with 2.5-mm. holes. Oversize to (37); undersize to (38).
37. Rolls 1 m. diameter. To (39).
38. Roller mill with 1-mm. screen. To (39).
39. Bucket elevator. Ore to (40); overflow water to (61).
40. Three sectional trommels in series with 5.6-, 4-, 2.8-, 2- and 1.4-mm. holes. Over 5.6 mm. to (37); through 5.6 on 4 mm. to (42); through 4 mm. on 2.8 mm. to (43); through 2.8 on 2 mm. to (43); through 2 on 1.4 mm. to (44); through 1.4 mm. to (41).
41. Hydraulic classifier with two spigots. Spigots to (45); overflow to (61).
42. One four-compartment jig. 1st discharge, lead concentrates to smelter; 2d and 3d, rich middlings to (35); 4th, poor middlings to (46); tailings are poor middlings to (46).
43 Two four-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d, zinc middlings to (34); 4th, poor middlings to (46); tailings are poor middlings to (46).
44. One four-compartment jig. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, zinc concentrates to smelter; tailings are poor middlings to (46).
45. Two four-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, zinc concentrates to smelter; tailings are poor middlings to (46).

(e) POOR MIDDLINGS DEPARTMENT.

46. Bucket elevator. Ore to (47); overflow water to (66).
47. Trommel with 5.6-, 4- and 2.5-mm. holes. Over 5.6 mm. to (48); through 5.6 on 4.0 mm. to (49); through 4.0 mm. to (50).
48. Rolls 1 m. diameter. To (51).
49. Rolls 1 m. diameter. To (51).
50. Three roller mills. To (51).
51. Bucket elevator. Ore to (52); overflow water to (59).
52. Three sectional trommels in series with 10-, 5.6-, 4-, 2.8-, 2- and 1.4-mm. holes. Over 10 mm. to (49); through 10 on 5.6 mm. to (49); through 5.6 on 4 mm. to (54); through 4 on 2.8 mm. to (55); through 2.8 on 2 mm. to (55); through 2 on 1.4 mm. to (56); through 1.4 mm. to (53).
53. Hydraulic classifier with four spigots. Spigots to (57); overflow to (59).
54. Two four-compartment jigs. 1st discharge, lead concentrates to smelter; 2d, rich middlings to (35); 3d and 4th, poor middlings to (46); tailings to (58).
55. Four four-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, poor middlings to (46); tailings to (58).
56. Two five-compartment jigs. 1st hutch, lead concentrates to smelter; 2d, lead middlings to (34); 3d and 4th, zinc concentrates to smelter; 5th, poor middlings to (46); tailings to waste.
57. Four five-compartment jigs. Products like (56).

(f) TAILINGS ELEVATOR.

58. Bucket elevator. Tailings to waste; overflow water to (65).

(g) SLIMES DEPARTMENT.

There is only one slimes department for both sections of the mill.
59. Spitzkasten for poor slimes middlings. Spigots to (60); overflow to (76). There are two four-compartment jigs which may be used here, and yield: 1st hutch, lead concentrates to smelter; 2d hutch to (67); 3d hutch, zinc concentrates to smelter; 4th hutch to (70); and tailings to waste.
60. Six Humboldt riffle tables of the Wilfley type, two Bartsch round bumping tables and two fixed convex round tables. Lead concentrates to smelter; lead middlings to (68); zinc concentrates to smelter; zinc middlings to (70); poor middlings to (73); tailings to waste.

61. Spitzkasten for rich slime middlings. Spigots to (62); overflow to (76).
62. One Humboldt riffle table and two fixed convex round tables. Lead concentrates to smelter; lead middlings to (68); zinc concentrates to smelter; zinc middlings to (70); tailings to (73).
63. Spitzkasten for original slimes. Spigots to (64); overflow to (76).
64. Six Humboldt riffle tables, two Bartsch tables and two fixed convex round tables. Lead concentrates to smelter; lead middlings to (68); zinc concentrates to smelter; zinc middlings to (70); poor middlings to (73); tailings to waste.
65. Spitzkasten for tailings slimes. Spigots to (73); overflow to (76).
66. Spitzkasten for slime overflow of middlings elevators. Spigots to (70); overflow to (76).
67. Pump for lead slime middlings. To (68).
68. Spitzkasten. Spigots to (69); overflow to (76).
69. One Humboldt riffle table and two Bartsch tables. Lead concentrates to smelter; lead middlings to (67); zinc concentrates to smelter; zinc middlings to (70); tailings to (73).
70. Pump for zinc slime middlings. To (71).
71. Spitzkasten. Spigots to (72); overflow to (76).
72. Three Humboldt riffle tables and two fixed convex round tables. Products like (69).
73. Pump for tailings slime. To (74).
74. Spitzkasten. Spigots to (75); overflow to (76).
75. Three Humboldt riffle tables and two fixed convex round tables. Lead middlings to (67); zinc concentrates to smelter; zinc middlings to (70); tailings to waste.
76. Thirty-six settling tanks. Settled slimes are reconcentrated in the old mill; overflow water pumped back to top of mill by two 300-mm. centrifugal pumps.

All jigs have sieves 950 by 450 mm. The three-sieve jigs are accelerated jigs; the four and five-sieve jigs are Harz eccentric jigs. In the slimes department the Humboldt tables treat the coarse material, the Bartsch tables treat the medium material, and the round tables treat the finest material. The coarse mill tailings are used to make a dam and the fine tailings are run in behind the dam. The water filters through the dam and comes out clear.

The second section is practically the same as the first. It treats ores that are more coarsely disseminated and consequently the cobbing and hand picking yield some clean lead ore, zinc ore and copper ore. There are slight differences in the sizes of holes in the trommels in the rich middlings and the poor middlings departments, and the quality of the products from the various compartments of the jigs is not the same as in the first section.

The whole mill requires 20 cu.m. of water per min., of which 18 cu.m. can be drawn from the supply tank at the top of the mill. The 12 cu.m. which go off with the tailings and are lost in running are supplied to the slime department as fresh water. Two 210-h.p. electric motors run the two sections of the mill; one 70-h.p. motor runs the slimes department; two 100-h.p. motors run the two centrifugal pumps and one 2.4-h.p. motor runs the elevator.

A year's average shows that 100 tons of ore yield: 0.99 ton lead concentrates from hand picking; 2.21 from jigging; 0.55 from the tables; 1.05 tons zinc concentrates from hand picking; 0.32 from jig beds; 9.93 from jigging; 4.34 from the tables; and 80.61 tons of tailings.

The lead concentrates average 72.53 per cent. lead and the zinc concentrates 54.39 per cent. zinc. The tailings contain 0.39 per cent. lead and 1.88 per cent. zinc. The old mill used 450 men; the new mill uses only 250 men. The new mill is built of steel and sandstone on a gently sloping site. It covers an area of 5881 sq.m. It is heated by steam and lighted by electricity.

Concentrating Plant at Salberget.[1]—The ore consisting of blende with a little galena in dolomite is delivered to (1).

1. Blake breaker, 330x500 mm. To (2).
2. Trommel with 20-mm. holes. Oversize to (3); undersize to (4).
3. Picking belt. Waste to dump; zinc ore to market; lead ore to market; residue to (4).
4. Blake breaker, 200x300 mm. To (5).
5. Ferraris screen with 6-mm. holes. Oversize to (6); undersize to (7).
6. Coarse rolls. To (7).
7. Ferraris screens with 8-, 6-, 4-, 3- and 2-mm. holes. Four oversizes to (10); undersize to (8).
8. Classifier. Spigot to (14); overflow to (9).
9. Classifier with eight spigots. First two spigots to (15); next three spigots to (16); last three spigots to (17); overflow to sump.
10. Four jigs. Concentrates; middlings to (11); tailings to waste.
11. Fine rolls. To (12).
12. Ferraris screens with 3- and 2-mm. holes. Over 3 mm. to (11); 3 to 2 mm. to (13); under 2-mm. to (9).
13. Jig. Concentrates; middlings to (11); tailings to waste.
14. Jig. Concentrates; middlings to (18); tailings to waste.
15. Two jigs. Concentrates; middlings to (18); tailings to waste.
16. Three jerking riffle tables. Concentrates; middlings to (18); tailings to waste.
17. Three round tables. Concentrates; middlings to (21); tailings to waste.
18. Classifier. Spigot to (19); overflow to (20).
19. Jig. Concentrates; middlings to (11); tailings to waste.
20. Jerking riffle table. Concentrates; middlings to (18); tailings to waste.
21. Classifier. Spigot to (22); overflow to sump.
22. Round table. Concentrates; middlings to (21); tailings to waste.

The original article contains descriptions of the machines, comparisons of the Wilfley, Bartlett and Ferraris tables and results of the treatment.

Tin Ore Dressing at East Pool, Cornwall.[2]—The ore contains tin oxide, wolframite, arsenopyrite, copper pyrites and gangue. The recovery averages 22 ℔. of tin oxide, 22 ℔. of arsenic, 7 ℔. of tungsten and 1 or 2 ℔. of copper per long ton. Probably the recovery is 70 per cent. of the total contents of the ore.

The ore is broken in rock breakers at the mine and trammed to the stamp mill. The stamp pulp is classified into sands and slimes and the sands are treated on Frue vanners or Wilfley tables which make concentrates (tin oxide, tungsten and arsenopyrite) to calciner, and middlings (copper pyrites with some tin oxides) to Frue vanners where the copper pyrites are separated out. At the calciner the arsenic is removed and collected in flues. The calcined material is classified and the sands passed over a second set of Wilfley tables yielding concentrates with about 3 parts tin oxide to 1 of tungsten and about 5 per cent. iron oxide.

Slimes from the classifiers are concentrated on rag frames to about 8 ℔. tin oxide per ton and then put over Acme tables which yield a high-grade concentrate to the calciner. Calcined slimes are again run over Acme tables. The combined concentrates are run through Wetherill magnetic separators to remove the tungsten. The stamp mill has 156 Cornish stamps each crushing 24 cwt. in 24 hr. to 25-mesh size. Two heads of Holman pneumatic stamps were recently installed.

Graphite Dressing.[3]—The Diamond Graphite Company, near Buckingham, Quebec, has an ore containing about 12 per cent. flake graphite in a gangue of schistose and laminated gneiss. At the mill the ore is first calcined in a roasting kiln. It then passes through a Blake breaker,

---

[1] G. Fagerberg, J. *Bihangtill Jern-Kontorets Annaler* (1907), p. 447.
[2] E. Walker. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 941. *Revue de Metallurgie*, Vol. IV (1907), p. 682.
[3] *Can. Min. Journ.*, Vol. I. (1907), p. 79.

crushing rolls, and sizing screen, and, after it has been crushed a second time in rolls, it is delivered to a series of Krom air jigs which do the principal work of separating the flakes from the gangue. The graphite then passes through the polishing and sizing department, after which it is ready for the market. The mill is expected to treat 100 tons in 24 hr. and to produce 6 to 8 tons of flake graphite. The mill is divided into sections and each section is run by an electric motor.

Another mill at Calumet, Ontario, is to use pebble tube-mills for cleaning and polishing the graphite flakes.

Elmore Vacuum Oil Process in Cornwall.[1]—The Dolcoath ore contains sulphides of copper, iron and occasionally zinc, associated with tin oxide and wolfram. The Elmore process seems to be particularly adapted to the saving of the sulphides from this mixture. The ore is crushed in rock breaker and ball mill to 20-mesh size and then passes to the Elmore vacuum plant, from which the tailings may be treated in the regular Cornish way to recover the tin and wolfram. In one test the ore contained 2.41 per cent. copper. The concentrates assayed 17.4 per cent. copper, 8 per cent. arsenic and 6 per cent. zinc. The tailings contained 14 ℔. of tin oxide per ton by the vanning assay and 0.23 per cent. copper. The extraction of copper was 92 per cent. One ton of ore required 3.7 ℔. of acid and 13 ℔. of oil.

The vacuum used at the Tywarnhaile plant is 28-in.[2] The oil required is 5 ℔. per ton; it is a medium petroleum product. The natural acidity of the mine water is sufficient without adding extra acid. The power required is 5 h.p. The amount treated is 25 to 30 tons per day, being pulp from ten 1050 ℔. stamps crushing through 25-mesh screens. The ore treated is from the dumps and contains 0.5 per cent. copper. The concentrates contain 9 per cent. and the tailings 0.08 per cent. copper.

Macquisten's Flotation Method.[3]—This uses no chemical or physical agents, subjects the ore to no preliminary treatment (except fine crushing) but simply causes sulphide minerals to float on the surface of water, while quartz and other gangue minerals sink to the bottom.

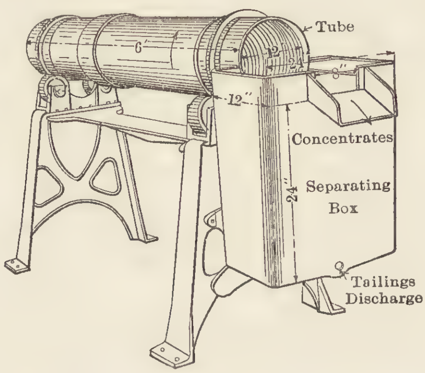
The principle which this machine exploits is based on the different affinities exhibited by the various constituents of sulphide ores to the surface tension of water. Sulphide minerals are positively affected by the surface tension of the water, i.e., they do not penetrate the surface easily, but tend to remain on the surface, while rock constituents are affected by it negatively, i.e., they break through the surface easily and sink through the water. It is believed that the flotation of the sulphide minerals is due to some superficial property which prevents them from becoming wetted, while gangue minerals do not possess this property and readily

[1] E. Walker. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1103.
[2] *Ibid.*, Vol. LXXXIII (1907), p. 1037.
[3] W. R. Ingalls. *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 771.

sink. It is found practically that the ability to float the sulphides is confined to those of a "greasy" character. These sulphides can be floated even after prolonged and repeated immersions in water.

It is obvious that the process is concerned with a delicate balance of small forces. The particles of ore in order to be floated must be small in size in order that their weight shall not cause them to break through the surface tension of the water. They must be brought very carefully upon the surface of the water so that the gangue, or easily wetted portion, may sink and the sulphides which are not well wetted may float off. The angle at which the mineral particles are presented to the surface of the water must be just right in order to insure the maximum flotation. The thickness of the pulp and speed of its delivery to the apparatus for flotation are important considerations.

The first attempt to work this process on a large scale was at the Adelaide Reduction Works at Golconda, Nevada. The ore treated is chalcopyrite with a smaller proportion of pyrrhotite and pyrite, and a little blende and galena, disseminated rather finely in a hard dense quartzose gangue containing spinel and garnet.



SKETCH OF APPARATUS, SHOWING DIMENSIONS.

The ore is crushed to 10-mesh size by two breakers, three sets of rolls and four 5-ft. Huntington mills. This product passes to spitzkasten to remove the slimes and then goes to two Callow screens with 30-mesh holes and the oversize is crushed in two Huntington mills with 40-mesh wire screens. The pulp is thickened and distributed to 25 series of separating tubes with four tubes in series in each set. The tubes (see figure) are made of cast iron. They are 6 ft. long, 1 ft. diameter and weigh about 450 lb. each. Externally they are cast with two tires which rest horizontally on supporting rollers. The discharge end of the tube is entirely open while the feed end is closed except for a hole in the center

large enough to admit the pipe which feeds the pulp. Internally the tube is cast with a helical groove. The tubes used at present have an interior helix of 1.5 in. pitch. The discharge end of the tube connects with a vessel, called the "separating-box," the joint between the latter and the tube being water-tight while the tube is free to revolve. At the side of the separating-box directly opposite to the discharge end of the tube an opening is cut out for the overflow of the surface layer of water, carrying the floating mineral. This opening, or weir, regulates the depth of water in the tube. At Golconda the bottom of the weir is 3 in. above the bottom of the tube, inside; consequently there is 3 in. of water in the tube. The feed of pulp to the tube and the discharge of the tailings from the bottom of the separating-box are so regulated that the water passing over the weir is about $\frac{1}{32}$ in. deep.

In operation the tube is rotated at 30 r.p.m. in a direction corresponding to the helix of the interior. The pulp is thus screwed through the tube and in its advance is repeatedly given an opportunity to slide upon the surface of the water. Once this has been effected the mineral remains on the water until the latter has overflowed into the launder which spouts it into the concentrate collecting tank.

The capacity of a single tube is five tons per 24 hr. The ore assays 2.7 to 3.2 per cent. copper. The material is concentrated in the ratio of 11 to 1. The concentrates assay 22 per cent. copper and the tailings 0.2 per cent. About 90 per cent. of the value in the material treated is saved although this does not include the slimes which amount to about 30 per cent. of the original ore.

One drawback of the process is its unsuccessful treatment of the slimes. The small particles do not settle rapidly enough and pass over the weir into the concentrate tank. Thus far it has been impossible to make more than a two-mineral separation, i.e., sulphides from gangue. Mr. Macquisten is now making experiments along this line and says that some of the experiments seem to indicate favorable prospects. However it seems improbable that the slight difference between the wetting of the various sulphides is sufficient to effect a good separation of the various sulphides on a commercial scale.

Dry Separation.

Magnetic Concentration.

Magnetic Separator at the Eisenzecher Zug Mine.[1]—This machine has two horizontal concentric revolving rings spaced 70 mm. apart and made up of vertical plates of iron separated by cement. At the four quarters of the circle these rings pass between the poles of four U-shaped electro-magnets,

[1] *Zeits. f. B. H., u. S.-wesen.* Vol. LV (1907), p. 131.

these poles pointing upward. Ore is fed downward between the rings at the four points where the rings pass between the poles. Non-magnetic particles continue their descent while magnetic particles are arrested by the magnetism induced in the plates of the rings. Continued revolution of the rings carries these magnetic particles out of the field and they fall off. Their removal is facilitated by a reversal of polarity at the next magnet.

Magnetic Separation.[1]—The author divides the machines into (*a*) low-power magnetic separators with moving magnets; (*b*) low-power magnetic separators with fixed magnets and moving armatures; (*c*) low-power magnetic separators with fixed magnets; (*d*) high-power magnetic separators. The Finney, Gröndal, Erikson, Wenström, Forsgren, Ball-Norton, Froding, Wetherill and Mechernich machines are described. Magnetic separation is especially applied to Swedish iron ores containing magnetite which is strongly magnetic and hematite which is weakly magnetic. Magnetic separation began in Sweden in 1885. In 1894 the old Monarch separator came out and in 1897 the Heberle and the Gröndal machines. The Wenström and Forsgren separators are more for coarse material while the Gröndal is better for fine. The capacity of a separator varies from 0.75 to 2.5 tons per hour. There are at the present time more than 20 plants in Sweden which use magnetic separation. The plant at Herrang is described. Here the Monarch separator is used, and from raw ore containing 45 per cent. iron, 0.003 per cent. phosphorus and 2 per cent. sulphur a concentrate is obtained which contains 60 per cent. iron, 0.003 per cent. phosphorus and only 0.5 per cent. sulphur. The Fröding machine did even better, yielding a concentrate with 62 to 64 per cent. iron from an ore with 25 per cent. iron. At Strassa the ore is a mixture of magnetite, hematite and quartz and contains 41 per cent. iron, 0.014 per cent. phosphorus and 0.08 per cent. sulphur. The concentrates contain 61 per cent. iron, 0.005 per cent. phosphorus and 0.04 per cent. sulphur. The largest plant in Norway is the Voranger where the ore contains 35 to 40 per cent. iron and 0.04 per cent. phosphorus. The ore is first crushed coarse and concentrated and this is followed by fine crushing and final separation. The Dunderlandsdal ore runs high in hematite. It analyzes 35 to 45 per cent. iron and about 0.20 per cent. sulphur. Various methods of concentration have been tried as follows: (*a*) Wet jigging; (*b*) magnetic separation of the magnetite and wet jigging for the hematite; (*c*) roasting to magnetize the hematite followed by magnetic separation; (*d*) removal of the magnetite by weak magnets and the hematite by strong magnets. The last scheme appears to be the best suited to Dunderland ore.

Magnetic Separation of Tin Oxide and Wolfram.[2]—It has been found in Cornish tin mills that magnetic particles, particularly magnetic oxide of

---

[1] C. Bugge. *Oester. Zeit. f. B. u. H.-wesen*, Vol. LV (1907), p. 102. *Teknisk Ugeblad*, Vol. LIII (1906), pp. 209, 217, 234.
[2] *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 112.

iron, adhere to the non-magnetic particles and so cause the non-magnetic constituents to be drawn out with the wolframite. This is especially true where a preliminary roast is used. To prevent this the ore is leached with sulphuric acid and subsequently washed and dried. This process removes the magnetic iron oxide or reduces it to the non-magnetic oxide of iron.

Electrostatic and Electromagnetic Separators.[1]—Of the three electrostatic separators which have been fairly successful in experimental work (the Huff-Dolbear, the Sutton-Steele and the Blake-Morscher) one, the Blake-Morscher, has become a commercial machine. Practically all sulphides except zinc blende are conductors of electricity, and thus a separation may be made of blende from pyrite which is not possible in the wet way.

Of the industrial magnetic separators the Wetherill, the Dings, the Cleveland-Knowles and the International are best known. The Wetherill is used on franklinite ore at Franklin Furnace, N. J. In the West, magnetic separation is used either upon the raw ore to remove marmatite or upon ore which has been given a roast at a low temperature to form magnetic oxysulphide of iron. The latter method is also used in cleaning zinc concentrates in the Wisconsin field.

The treatment of Leadville ore by the Colorado Zinc Company at Denver is as follows: The ore is crushed and sized on King screens. It is then passed over Wilfley tables yielding (*a*) silicious tailings, (*b*) zinc-iron middlings, and (*c*) lead concentrates. The silicious tailings are rejected while the lead concentrates go to a lead smelter. The zinc-iron middlings are dried and passed over a Wetherill separator where the magnetic black jack is eliminated and sold to a spelter plant as zinc ore. The non-magnetic product is treated by Blake-Morscher machines producing an iron product low in zinc and a zinc product suitable for the spelter plant. If the previous magnetic treatments were not inserted, the iron product from the Blake separators would be high in zinc and thus unsuitable for a flux for the lead smelter, and besides the loss in zinc would be heavy.

Electrostatic Separation of Copper Ores.[2]—Copper minerals, as a rule, are conductive of static electricity. The silicates are the least so and the carbonates, being usually impregnations, are not much better. The oxide of copper is a good conductor and the compounds of copper with sulphur, arsenic and antimony are fine conductors. A Blake-Morscher electrostatic machine was tested on native copper ore in the Calumet & Hecla mill and the saving was found to be almost the same as that obtained by wet concentration. The idea of substituting the Blake machine for the present method was not entertained, however, in view of the greater cost, the necessity of working dry, and the money invested in the present installation.

[1] W. M. Johnson. American Electrochemical Society, Vol. XI (1907), p. 265. *Electrochem. and Met. Ind.*, Vol. V (1907), p. 83.
[2] W. G. Swart. Colorado Scientific Society, Vol. VIII (1906), p. 88.

On copper carbonates in a silicious gangue, promising results have been obtained but where the carbonates occur in lime the separation is apt to be poor. At Golconda, Nev., an ore containing chalcopyrite in a heavy garnet and spinel gangue was crushed, dried and treated by a Blake machine without sizing or removal of dust. The results were as follows:

TEST OF BLAKE MACHINE AT GOLCONDA.

	Feed	Concentrate	Tailing
Ounces gold per ton	0.01	0.04	Trace
Ounces silver per ton	4.50	10.65	1.3
Per cent. copper	4.10	14.32	0.17
Per cent. iron	8.80	22.90	....
Per cent. silica	46.35	16.48	....
Per cent. alumina	22.09	7.24	....
Per cent. zinc	2.80	....	....
Per cent. sulphur	4.94	18.30	....
Per cent. lime	8.01	3.65	....

The extraction on this ore has ranged from 60 to 86 per cent., depending upon the grade of concentrates made. Very similar results were obtained on an ore containing bornite in a gangue of reddish garnet and quartz from Eldorado and Amador counties, California.

There is plainly a field for electrostatic separation in arid regions where water for milling purposes is lacking. The Cananea Consolidated Copper Company has secured the rights for Sonora, Mexico. It also serves to obtain copper as a by-product from zinc ores. In one case enargite (arsenical copper) is removed from pyrite which is to be used for acid making. About 75 per cent. of the pyrite is recovered. In another case copper and iron sulphides and oxides are recovered from barren materials in the flue-dust from a copper smelter, thereby relieving the furnace of about 70 per cent. of the flue-dust ordinarily returned.

The three great difficulties in electrostatic separation are insulation, feeding and dust. Air is the best insulator; hard rubber and micanite are the only two satisfactory solid materials.

As to feeders, shaking or tapping feeds are not adapted to unsized products because they cause a stratification of coarse material beneath the fine. Rollers do not work well on material coarser than 12 mesh while material below 40 mesh requires a positive feed.

To avoid dust it is advocated that in many cases it is better to use wet crushing of ores followed by classification and subsequent drying and separation of everything but the finest slime. With the equivalent of good Ohio coal costing $6 per ton at the fire box, 20-mesh material containing 10 per cent. moisture can be dried for from 7 to 9c. per ton, depending on labor cost and the specific gravity of the material.

No definite size for crushing can be stated. Commercial work has been done on 6-mesh jig middlings, consisting of marcasite and blende. On

crude molybdenite ores from Maine, 10 mesh was found the best. In neither of these cases was any sizing done. As a rule the machines do best work on suitable material at 16 to 40 mesh. Sizing is usually more important on meshes coarser than 24 than on the finer grades. Very fine material alone gives low capacity.

Electrostatic Separation.[1]—A series of experiments has been made upon an electrostatic separator made by the Humboldt company which appears to be similar in principle to the American Blake-Morscher machine. The high-potential electricity is supplied by a dynamo, transformer and revolving switch. The results of each experiment are tabulated showing the weights of the products, the analyses and the extraction. The experiments were made: (1) upon various sizes of pyrite-blende middling products from wet concentration, some being coarsely disseminated material and some finely disseminated, some having the blende in the form of "resin blende" with little or no chemically combined iron and some containing the blende as "black jack" with more or less chemically combined iron; (2) upon various sizes of pyrite-blende middlings which contained galena either entirely as free grains, or partly as included grains; (3) upon various sizes of pyrite-blende middlings associated with chalcopyrite; (4) upon various sizes of raw ore with feldspar and atacamite finely disseminated; (5) upon various sizes of raw ore containing quartz, pyrite and chalcopyrite both under conditions of dry crushing and sizing and under conditions of wet crushing coupled with removal of fine slimes by water.

The following conclusions are drawn from the results of the tests: (1) For best results the particles should not be coarser than 1 mm., because the larger the particle the longer the time required for it to receive its charge. Therefore coarse particles of conductors do not remain a sufficient time in the electrostatic field to receive enough charge to cause a repulsion and they consequently descend under the influence of gravity along the same path as non-conductors. (2) Electrostatic separation is not possible on material finer than a No. 200 sieve, because fine particles, both of conductors and non-conductors, receive their charges very quickly and there is not a sufficient difference in these times to allow the conductor to receive its charge and be repelled, before the non-conductor has also received its charge and has been repelled. (3) Dust is harmful to the separation because particles of conducting dust on a particle of non-conductor cause this particle to act as a conductor and *vice versa.* If the ore is crushed dry the dust must be removed by water and since it cannot be treated it becomes a problem whether to add it to the concentrates or to the tailings. (4) A good separation is not possible unless the grains have been well freed

---

[1] Friedr. Esser. *Metallurgie,* Vol. IV (1907), pp. 592, 607. *Oester. Zeits. f. B. u. H.-wesen,* Vol. LV (1907), p. 591. *Electrochem. and Met. Ind.,* Vol. V (1907), p. 469.

by the crushing. A bit of non-conductor attached to a particle of conductor exerts an undue influence in throwing the particle into the non-conductor class and *vice versa.* This is different from wet concentration or magnetic concentration, where an included grain is governed in its movement by the preponderating mineral. (5) It follows that impregnated ores are not susceptible to electrostatic separation. (6) Chemical impurities in a mineral affect the separation. This is especially true of blende. Pure blende is a non-conductor and easily separated from pyrite, but if the blende is of the black variety and contains chemically combined iron it tends to act like a conductor and a poor separation results. (7) Galena when associated with blende and pyrite goes partly into the concentrates and partly into the tailings. (8) Atacamite acts as a non-conductor. (9) Climatic conditions will influence the quality of the separation.

Pneumatic Concentration.

Sutton-Steele Dry Concentrator.[1]—This table which in many respects resembles the Wilfley table used in wet concentration, has been described in previous volumes. A new form of table top has been provided, however, which does away with raised riffles and leaves a smooth and unobstructed table top. Instead of riffles, tapering strips of paper are placed beneath the muslin top of the table these strips conforming exactly in position to the riffles formerly used. These strips form a series of dead lines on the table sufficient to cause a slight retardation similar to raised riffles but not upsetting the stratification of the pulp on the table.

Preparation of Dry Zinc Ore.[2]—The process used at the Bleischarley mine near Beuthen, Upper Silesia, is designed to take out the mine fines for wet concentration, to remove much waste material and dust, and to divide the ore into different mineral classes for the wet separation. The Bleischarley plant does away with hand labor to a considerable extent, and removes zinc fines which would otherwise float away in the wet treatment. Sizing tests have shown that the blende crushes faster than the gangue, which causes an enrichment of the fine sizes. The blende ore, free from lead, is hoisted from the mine and dumped upon (1).

1. Grizzly with 60-mm. spaces. The ore is here washed with a spray of water. The oversize is hand picked into shipping ore, concentrating ore which goes over to crushing department, and waste to dump; undersize to (2).
2. Shaking screen with 30-mm. holes. Oversize to (3); undersize to (8).
3. Conical wash trommel with 10-mm. holes. Oversize through hopper and feeder to (4); undersize to (5).
4. Annular picking table revolving on a central shaft, and sloping toward the center. Blende, pyrite and waste are picked out and thrown down chutes into cars. The residue is scraped off and goes to crushing rolls.
5. Two spitzkasten. Spigots to (7); overflow to (6).
6. Settling tank for blende slimes.
7. Hopper. By elevator to (8).
8. Two screw conveyors. To (9).
9. A cylindrical drier, 1.7 m. diameter and 9 m. long, sloping 4 deg., mounted on rollers. Projections on the inside serve to lift the ore and shower it through the hot gases which pass through at a temperature of 100 deg. C. Dust is drawn off by a fan to (10); dried ore to (11).
10. Settling chamber with a fine screen across the exit. Settled dust is drawn off in bags.
11. Hopper. By elevator to (12).

---

[1] *Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 441.
[2] Piegza, *Glückauf*, Vol. XLIII (1907), p. 963.

12. Two double Schwidtal shaking screens with 15-, 10-, 6- and 2.5-mm. holes. First three oversizes go to washing department. The fourth oversize (6 to 2.5 mm.) sometimes goes to washing department and sometimes by shaking conveyor to (13).
13. Shaking screen with 4-mm. holes. Oversize to washing department; undersize to smelter.

The preceding hopper and elevator (11) and screens (12) and (13) are enclosed and the dust is drawn off by a fan to (14)

14. A conical settling flue with four hoppers in the bottom followed by a settling chamber with zigzag passages. Settled dust drawn off in bags. Air current to (15).
15. Chimney with falling water spray to catch the last of the zinc dust.

The ore coming to this plant contains about 28 per cent. zinc. The coarse products go to jigs but the fine sizes are sufficiently rich to be smelted direct. This is shown in the accompanying table.

RESULTS OF DRY CONCENTRATION AT BLEISCHARLEY.

Size	Zinc Contents after dry preparation	Zinc Contents after jigging	Size	Zinc Contents after dry preparation	Zinc Contents after jigging
	Per Cent.	Per Cent.		Per cent.	Per Cent.
25 to 15 mm..........	28	50	2.5 to 0 mm...........	40	....
15 to 10 mm..........	31	52—55	Dust from (10)........	38—40	....
10 to 6 mm...........	34	48	Dust from flue (14)....	25	....
6 to 4 mm............	36	44	Dust from chamber (14)	19	....
4 to 2.5 mm..........	38	....	Dust from (15)........	19	....

The plant treats about 75 tons of zinc ore free from lead. Power is supplied by a 60-h.p. steam engine.

Coal Washing.

Pittsburg Jig.[1]—This new coal jig has a movable sieve driven by two accelerated, sliding-block mechanisms. The sieve slopes downward toward the tail and the slate is discharged through a tail slot into the jig tank below. The jig box fits tightly in the jig tank. Beneath the jig sieve and moving with it is a box with upward opening, hinged valves which are open on the down stroke and closed on the up-stroke. This makes a pulsion jig without suction. On the up-stroke, water is lifted and washes the coal over the tail of the jig and over the side of the jig tank upon a perforated inclined screen. The water falls into a tank below the screen which is connected to the jig tank by twelve hinged valves, while the drained coal is removed by a drag conveyor.

Anthracite Breaker of the Pacific Coal Company, Ltd., Bankhead, Alberta.[2]—The coal is brought to (1).

1. Automatic feeder. To (2).
2. Bar screen with 3-in. spaces. Oversize to (3); undersize to (28).
3. Platform. Material is separated into coal to (4), bony coal to (14), slate to (43).
4. Rolls, 36x36 in., with steel teeth $1\frac{3}{4}$ in. square, $3\frac{1}{2}$ in. long, spaced $4\frac{3}{4}$ in. apart and set diagonally. To (5).
5. Broken screen with $3\frac{1}{4}$-in. holes. Oversize hand picked into broken coal to (44) and slate to (43); undersize to (6).
6. Egg screen with $2\frac{1}{4}$-in. holes. Oversize hand picked into egg coal to (44) and slate to (43); undersize to (7).
7. Elevator. To (8).
8. Stove screen wth $1\frac{1}{2}$-in. holes. Oversize hand picked into stove coal to (44) and slate to (43); undersize to (9)
9. Nut screen with 1-in. holes. Oversize hand picked into nut coal to (44) and slate to (43); undersize to (10).
10. Pea screen with $\frac{9}{16}$ in. holes. Oversize pea coal to (44); undersize to (11).
11. No. 1 buckwheat screen with $\frac{5}{16}$-in. holes. Oversize passes over slater bars yielding No. 1 buckwheat to (44) and slate which is flat and does not pass through the slater bars to (43); undersize to (12).

---

[1] *Mines and Minerals*, Vol. XXVII (1907), p. 329.

[2] L. Stockett and B. R. Warden. Canadian Mining Institute, Vol. IX (1906), p. 261. *Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 857.

12. No. 2 buckwheat screen with ¼-in. holes Oversize No. 2 buckwheat to (44) or burnt under boilers; undersize to (13).
13. Dust bin. To briquet plant.
14. Rolls like (4). To (15).
15. Broken screen with 3¼-in. holes. Oversize to (16); undersize to (17).
16. Picking belt. Broken coal to (19) or (44); bone to (20); slate to (43).
17. Egg screen with 2¼-in. holes. Oversize to (18); undersize to (21).
18. Picking belt. Egg coal to (19) or (44); bony coal to (20); slate to (43).
19. Rebreaking rolls, 24 in. diameter 36 in. face, with 1¼-in. square steel teeth, 2 in. long, set diagonally and spaced 3¼-in. apart, revolving 133 r.p.m. To (7).
20. Bony rolls, 24 in. diameter, 36 in. face, with 1-in. square steel teeth 1¼-in. long set diagonally and spaced 1$\frac{5}{16}$ in., making 133 r.p.m. To (21).
21. Elevator. To (22).
22. Stove screen with 1½-in. holes. Oversize stove coal to (23); undersize to (24).
23. Emery slate picker. Its products are further hand picked to remove slate from the coal and coal from the slate. Stove coal to (44); slate to (43).
24. Nut screen with 1-in. holes. Oversize to (25); undersize to (26).
25. Emery slate picker followed by hand picking as in (23). Nut coal to (44); slate to (43)
26. Pea screen with $\frac{9}{16}$-in. holes. Oversize to (27); undersize to (11).
27. Emery picker. Pea coal to (44); slate to (43).
28. Chute or dust screen with ¼-in. square holes. Oversize to (29); undersize to (13)
29. Broken screen with 3¼-in. holes and $\frac{3}{16}$-in. holes at the end. Oversize to (16); undersize of 3¼-in to (30); undersize of $\frac{3}{16}$-in. to (13).
30. Egg screen with 2¼-in. holes. Oversize to (18); undersize to (31).
31. Elevator. To (32).
32. Stove screen with 1½-in. holes. Oversize to (33); undersize to (34)
33. Emery pickers. Products like (23).
34. Nut screen with 1-in. holes. Oversize to (35); undersize to (37).
35. Slater bars. Flat pieces (slate) to (43); undersize to (36).
36. Emery pickers. Products like (25).
37. Pea screen with $\frac{9}{16}$-in. holes. Oversize to (38); undersize to (40)
38. Slater bars. Slate to (43); undersize to (39).
39. Emery pickers. Products like (27).
40. No. 1 buckwheat screen with $\frac{5}{16}$-in. holes. Oversize to (41); undersize to (42).
41. Slater bars. Oversize (slate) to (43); undersize (No. 1 buckwheat coal) to (44).
42. No. 2 buckwheat screen with ¼-in. holes. Oversize to (44) or burnt under boilers; undersize to (13).
43. Slate is conveyed to the rock bin and is taken from there in cars to waste dump.
44. Railroad bins are provided for all sizes of coal and retail bins for the egg, stove, nut and pea sizes. Broken coal and egg coal come to these bins by chutes; stove, nut and pea coal are delivered to the bins by spiral chutes to prevent breakage. These bins have gates delivering to a belt conveyer which carries the coal to a lip screen, the oversize of which passes to cars while the undersize is lifted by screenings elevator to (7).

Shaking screens are used. They are built of steel plates and angle iron, suspended on ⅝-in. rods and sloping 2 in. per foot. They are driven by eccentrics making 100 six-inch strokes per minute. Screens are 4½ ft. wide and 12 ft. long with the following exceptions: The No 1 buckwheat screens are 5x18 ft., the No. 2 buckwheat screens 6x18 ft. and the screens (29) to (37) inclusive are 6x12 ft.

The elevators (7) and (21) have buckets 16x8 in. containing 22 lb. of coal when level full and at a speed of 100 buckets per minute each elevator has a capacity of 60 tons per hour. The elevator (31) has buckets 24x12 in., each containing 80 lb. of coal and at a speed of 66 buckets per minute the capacity is 150 tons per hour. The elevator in (44) is like (7) but runs at one-half the speed.

The Emery slate pickers each has a capacity of 100 tons in 10 hours. They are based on the principle that slate is flat and does not travel so fast in a chute as does the more cubical coal. Coal rolls and slate slides and if a space is interposed the coal has enough momentum to carry it across while the slate falls through. Each picker has three such openings.

Power is furnished by a 16- and 28x36-in. cross-compound, horizontal engine capable of developing 350 h.p. Rope drive is used except for a few short drives where rubber belting is used. The building is heated by direct steam in 5000 ft. of 2-in. pipe. Lighting requires 200 incandescent electric lights of 16 c.p. each.

The breaker cost $119,175. The labor required is 1 breaker engineer, 1 oiler, 1 bottom man, 1 car oiler, 1 spragger, 1 top man, 1 weighman, 2 dumpers, 5 platform men, 6 screen men, 3 Emery picker men, 35 hand pickers, 4 dirt bank men, 1 locomotive engineer, 2 locomotive switchmen, 5 laborers, 1 box car engineer, 4 box car loader men, 1 foreman, 1 carpenter and repairman and 1 weighman.

Bituminous Coal Washing in Southern Colorado.[1]—A new plant of the American Smelting and Refining Company at Cokedale crushes the coal in a toothed-roll crusher and elevates it to raw-coal bins. Thence the coal is drawn as needed and crushed in a No. 5, Pennsylvania, swing-hammer crusher and washed on three Pittsburg jigs. The refuse is rewashed on a final jig. The coal from the jigs is sluiced to a sludge recovery tank from which it is removed by two perforated bucket elevators and delivered to the washed coal bins.

The plants of the Colorado Fuel and Iron Company use Forrester jigs. The coal is first crushed in a tooth-roll crusher and is then delivered to a trommel making three sizes, each size being washed separately on jigs. The washed coal passes to unwatering trommels and the water and sludge from these trommels is pumped to settling tanks where the sludge settles out and is delivered to the washed coal bins, together with the oversize of the unwatering trommel. The plant of the Victor Fuel and Iron Company at Hastings uses Stewart jigs. The plant at Gardner, New Mexico, is equipped with New Century jigs.

Coal Washing in the Saar District, Prussia.[2]—Coal to be washed is screened into three sizes, (1) "lump" over 70 or 80 mm., (2) "wash coal" between 70 or 80 mm. and 30 or 40 mm. and (3) "slack" below 30 or 40 mm., by means of shaking screens about 4.5 m. long, 1.6 m. wide and sloping 15 to 16 degrees.

The lump coal is taken awav from the screens by picking belts about 1 m. wide and 8 to 12 m. long and moving at the rate of 0.25 to 0.30 m. per second. The wash coal and slack coal are handled by scraping conveyors, conveying belts and elevators. The Heinitz mine has a scraping conveyor for wash coal which is 29 m. long and 1.2 m. wide and handles 90 tons per hour. The wash coal is further sized either by shaking screens built by Schüchtermann and Cremer or trommels built by Baum. The former give better separation and make less fines.

Coarse jigs are used for material of over 8 mm. size and fine jigs for material under 8 mm. Both are of the fixed sieve type and have continuous discharge of the waste. Coarse jigs treat up to 20 tons per hour or 5 to 6 tons per sq. m. of sieve surface. They make 30 to 40 strokes per min. of 12 to 25 cm. each.

---

[1] *Min. Reporter*, Vol. LVI (1907), p. 312.
[2] Friedrich Okorn. "Berg- u. Hüttenmannisches Jahrbuch," Vol. LV (1907), p. 1.

The fine jigs are generally 2-compartment and have either a feldspar bed (Schüchtermann) or slate bed (Baum). A jig with each sieve 725 mm. wide and 1250 mm. long, treats 5 to 7 tons per hour. The strokes are 120 per minute of 6 to 20 mm. The first sieve removes clean waste while the second sieve takes out a mixture of coal and waste for further treatment.

Coal slimes are treated on a Köhl-Simon sieve. This is a fine screen and has jets of water to wash the clayey material into the undersize while the fine particles of coal are moved forward over the sieve. In a test at the Reden mine 249 cu.m. of pulp containing 10.39 per cent. solid material were treated. This required 264 cu.m. of clear wash water. The products were 4.78 tons of slime coal with 8.16 per cent. ash and 14.87 per cent. moisture, after draining 12 hr., and 508.4 cu.m. of slime water with 30.74 per cent. ash and 60.75 per cent. coal. This water is clarified in settling tanks which discharge continuous spigots of thickened pulp and overflow of clear water to be used over again.

The ratio of water used to material treated is 4 to 1 in the coarse jigging and 6 to 1 in the fine jigging. The unwatering of the washed nut coal is effected by sieves in the Schüchtermann system and by drainage cars in the Baum system. The moisture content of the resulting egg and nut coal is not over 4 per cent. The draining of very fine sizes is effected by perforated conveyors, elevators and cars. Baum uses a special patented unwatering conveyor. Such an apparatus at the Brefeld mine is 9.5 m. long, 1.8 m. wide and consists of draining boxes perforated with 7-mm. holes all over the bottom and half way up the side. It has a speed of 0.125 mm. per second. Material from 8 mm. downward is first screened into two sizes: 8 to 2 mm. and 2 mm. and less. The coarser size falls first into the conveyor and then the finer size is delivered on top. The drained product contains 19 per cent. moisture.

The detailed treatment of coking coal at the Heinitz mine is as follows: Coal ranging from 35 mm. downward, and containing 23 to 25 per cent. ash goes to (1).

1. Sieve with 7-mm. holes. Oversize to (2); undersize to (3).
2. Coarse jigs yielding coal to (5); middlings to (4), and waste.
3. Fine jigs. Products like (2).
4. Roller mill. Product rewashed.
5. Screen with 1½-mm. holes. Oversize to (6); undersize by centrifugal pump to (7).
6. Draining conveyor 25 m. long and 3 m. wide. Product contains 10 to 11 per cent. moisture and goes to (9).
7. Four settling boxes. Spigots to (8); overflow clear water.
8. Köhl screen. Oversize to (6) on top of the coarse material from (5).
9. Two disintegrators. Product to coking ovens.

The washer treats 70 to 90 tons per hour. The coarse-jig coal contains 5.3 per cent. ash, the fine-jig coal 6.5 per cent., the rewashed coal 17.6 per cent. and the slime coal 16 per cent. The final mixed product contains 8.5 per cent. ash and 10 to 12 per cent. moisture.

Additional Bibliography.

For lack of space the following contributions to the literature of ore dressing during 1907 are referred to by title only.

Crushing and Grinding.

Chrome Steel Jaw Plates.—*Min. Reporter,* Vol. LV (1907), p. 12.

New Rock Breaker.—*Min. Wld.,* Vol. XXVII (1907), p. 360.

Bigelow Patent Breaker.—*Min. Journ.,* Vol. LXXXI (1907), p. 285.

Long Life of Crushing Machine.—*Eng. and Min. Journ.,* Vol. LXXXIV (1907), p. 589.

Nordberg Compound Steam Stamp.—*Eng. and Min. Journ.,* Vol. LXXXIV (1907), p. 349.

Development of Chile Mills.—*Min. Reporter,* Vol. LV (1907), p. 515.

Wild Regrinding Mill.—*Min. Wld.,* Vol. XXVI (1907), p. 699. *Min. Reporter,* Vol. LV (1907), p. 251. *Min. and Sci. Press,* Vol. XCIV (1907), p. 414.

Crushing Machines for Lime, Cement, Phosphate, Etc.—*Génie Civil,* Vol. L (1907), p. 422.

El Oro Tube-Mill Lining.—*Min. and Sci. Press,* Vol. XCV (1907), p. 466.

Ferreira Tube-Mill Practice.—*Min. Reporter,* Vol. LV (1907), p. 413.

Tube-Mills on the Rand.—*Min. and Sci. Press,* Vol. XCIV (1907), p. 574.

Tube-Mills at Waihi, New Zealand.—*Journal,* Chem., Met. and Min. Soc. S. Africa, Vol. VIII (1907-08), pp. 10, 53. *Min. Journ.,* Vol. LXXXII (1907), p. 383. *New Zealand Mines Record,* Vol. XI (1907), p. 145.

Computation of Crushing Efficiency of Tube-Mills.—*Journal,* Chem., Met. and Min. Soc. S. Africa, Vol. VII (1906-07), pp. 207, 265, 289. *Electrochem. and Met. Ind.,* Vol. V (1907), p. 191.

Concentrating Machinery.

Wash Trommels.—*Metallurgie,* Vol. IV (1907), p. 515.

Hand Screen at Monteponi.—*Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 1041.

Standardization of Screens.—*Min. and Sci. Press,* Vol. XCIV (1907), p. 52.

Mechanical Settling and Filtering of Water.—*Zeits.* Ver. Deuts. Ing., Vol. L (1906), p. 1947.

Settling Pond.—*Zeits. f. B., H. u. S.-wesen,* Vol. LV (1907), p. 132.

Slimes Settlement.—*Journal,* Chem., Met. and Min. Soc. S. Africa, Vol. VII (1907), June. *Min. Reporter,* Vol. LVI (1907), p. 192.

Advantages of Hand Picking.—*Min. Reporter,* Vol. LV (1907), p. 32.

Lührig Picking Table.—*Metallurgie,* Vol. IV (1907), p. 248.

Traylor Iron Jig.—*Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 425.

Lührig Round Table.—*Zeits. f. B., H. u. S.-wesen,* Vol. LV (1907), p. 129.

Doubled-Decked Round Table.—*Zeits. f. B., H. u. S.-wesen,* Vol. LV (1907), p. 129.

Belt Elevators.—*Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 560.

Wolverine Ore Bins.—*Min. and Sci. Press,* Vol. XCIV (1907), p. 304.

Dust Settling.—*Glückauf,* Vol. XLIII (1907), p. 203.

Dust Settling Apparatus.—Inst. Min. and Met., Vol. XVI (1907), p. 51. *Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 856.

Disposal of Tailings.—*Min. and Sci. Press,* Vol. XCV (1907), pp. 337, 368, 459.

Flow of Water Carrying Sand in Suspension.—*Eng. and Min. Journ.,* Vol. LXXXIV (1907), p. 536.

Milling Practice.

Copper Range Company's Rock Houses.—*Min. and Sci. Press,* Vol. XCIV (1907), p. 375.

Osceola Mill.—*Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 1180.

Calumet & Hecla Tailings Plant.—*Min. Wld.,* Vol. XXVI (1907), p. 762.

Washoe Concentrator.—*Mines and Minerals,* Vol. XXVIII (1907), p. 131.

Milling in the Cœur d'Alene District.—*Min. and Sci. Press,* Vol. XCIV (1907), p. 243.

Milling in the Missouri-Kansas Field.—*Mining Science,* Vol. LVI (1907), p. 608.

Milling in the Joplin District.—*Mines and Minerals,* Vol. XXVIII (1907), p. 158.

Grundling No. 5 Mill, Joplin.—*Min. Wld.,* Vol. XXVII (1907), p. 1043.

Davey No. 3 Mill, Joplin.—*Mines and Minerals,* Vol. XXVIII (1907), p. 171.

Providence Mill, Webb City, Mo.—*Eng. and Min. Journ.,* Vol. LXXXIV (1907), p. 877.

Yellow Dog Mill, Mo.—*Mines and Minerals,* Vol. XXVIII (1907), p. 167.

Milling Sheet Ground Ore in the Joplin District.—*Eng. and Min. Journ.,* Vol. LXXXIV (1907), p. 59.

Milling in Southwest Missouri.—*Min. Reporter,* Vol. LV (1907), p. 134.

Sludge Mill Practice in the Missouri-Kansas District.—*Min. Reporter,* Vol. LVI (1907), p. 418.

Tails and Sludge Milling in the Joplin District.—*Mines and Minerals,* Vol. XXVIII (1907), p. 162.

Milling in the Quapaw District, Okla.—*Mines and Minerals,* Vol. XXVII (1907), p. 445.

Zinc Milling in Wisconsin.—*Eng. and Min. Journ.,* Vol. LXXXIII (1907), p. 11.

Union Mill at Gratz, Ky.—*Eng, and Min. Journ.*, Vol. LXXXIII (1907), p. 658.

Ore Dressing in Colorado.—*Bull.* Colorado School of Mines, Vol. III (1907), p. 229.

Treatment of Zinc Ores in Colorado.—*Mines and Minerals*, Vol. XXVIII (1907), p. 5. *Metallurgie*, Vol. IV (1907), p. 821.

Concentrating Zinc Ore in Colorado.—*Min. Reporter.*, Vol. LV (1907), p. 218.

Milling at Gladstone, Colo.—*Mines and Minerals*, Vol. XXVII (1907), p. 498.

Mill of the Western Chemical Manufacturing Company.—*Mining Science*, Vol. LVI (1907), p. 611.

Old Hundred Mill, Howardsville, Colo.—*Mines and Minerals*, Vol. XXVII (1907), p. 390.

Gold Prince Mill, Colo.—*Mines and Minerals*, Vol. XXVII (1907), p. 337.

Ore Washing at Cripple Creek, Colo.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 352.

Daly-Judge Mill, Park City, Utah.—*Bull.* Colorado School of Mines, Vol. IV (1907), p. 31. *Mines and Minerals*, Vol. XXVIII (1907), pp. 32, 79.

Boston Consolidated Mill, Utah.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 479.

Steptoe Vallev Mill, Nev.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 813.

The Desert Mill, Nev.—*Min. and Sci. Press*, Vol. XCV (1907), p. 494.

Milling at Virginia City, Nev.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 261.

Kelley Mill, N. M.—*Min. Wld.*, Vol. XXVII (1907), p. 267.

Concentrating at Humboldt, Ariz.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 901.

Milling at Mowry, Ariz.—*Mines and Minerals*, Vol. XXVII (1907), p. 529.

Milling at Deloro, Ontario.—*Can. Min. Journ.*, Vol. I (1907), p. 517.

Milling in the Rossland District, B. C.—*Min. Wld.*, Vol. XXVII (1907), p. 1123.

Greene Cananea Concentrator.—*Mines and Minerals*, Vol. XXVII (1907), p. 465.

Jigging by Hand.—*Min. and Sci. Press*, Vol. XCV (1907), p. 557.

Primitive Mexican Dressing plant.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 655.

Milling Practice at the Granadeña Mill, Chihuahua.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 637.

Dolores Mill, Chihuahua.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 221.

Mill of the Guanajuato Reduction and Mines Co.—*Min. and Sci. Press*, Vol. XCIV (1907), p. 284. Vol. XCV (1907), pp. 78, 457.

Braden Copper Company's Mill.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1059.

Stamping and Concentrating on the Rand.—*Journal*, Chem., Met. and Min. Soc. S. Africa, Vol. VIII (1907), p. 1. *Min. and Sci. Press*, Vol. XCV (1907), p. 434. *New Zealand Mines Record*, Vol. XI (1907), p. 147.

Mill Construction on the Rand.—*Min. and Sci. Press*, Vol. XCIV (1907), p. 632.

Berzelius Mill, Bensberg.—*Metallurgie*, Vol. IV (1907), p. 548.

New Blende Concentrator.—*Zeits. f. B., H. u. S.-wesen*, Vol. LV (1907), p. 128.

Milling Lead and Zinc Ores in North Wales.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 871.

Cornish Tin Dressing.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 708. *Revue de Metal.*, Vol. IV (1907), p. 597.

Dressing Tin Ores in Cornwall.—*Min. Wld.*, Vol. XXVI (1907), pp. 598, 626, 691.

Tin Stream Works of the Red River, Cornwall.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 991.

Tin Gathering in South Africa.—*Journal*, Chem., Met. and Min. Soc. S. Africa, Vol. VIII (1907), p. 167.

Tin Ore Concentration in Tasmania.—*Min. Wld.*, Vol. XXVI (1907), pp. 719, 776, 806.

Diamond Dressing in South Africa.—Inst. of Min. and Metal., Vol. XVI (1907), p. 200. *Mines and Minerals*, Vol. XXVII (1907), p. 402. *Min. Journ.*, Vol. LXXXI (1907), p. 45.

Diamond Washery of the DeBeers Company.—*Ann. des Mines*, Ser. X, Vol. XI (1907), p. 607.

Dressing Manganese Ore.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 478.

Dressing Graphite Ores in Canada.—Canadian Min. Inst. (1907). *Can. Min. Journ.*, Vol. I (1907), p. 163. *Min. Wld.*, Vol. XXVI (1907), p. 627.

Concentrating Chrome Ore.—Canadian Min. Inst., Vol. IX (1906), p. 35.

Asbestos Dressing.—*Min. Wld.*, Vol. XXVII (1907), p. 799.

Cleaning Monazite in the Carolinas.—"Mineral Resources of the United States'' (1906), p. 1200. *Min. Wld.*, Vol. XXVII (1907), pp. 1017, 1130. *Can. Min. Journ.*, Vol. I (1907), p. 524.

Washing Glass Sand in New Jersey.—Annual Rep. State Geol. of New Jersey for 1906. *Min. Reporter*, Vol. LVI (1907), p. 357.

Preparation of Molding Sand.—*Stahl und Eisen*, Vol. XXVII (1907), Part II, pp. 1485, 1536, 1576.

Corundum Milling.—*Can. Min. Journ.*, Vol. I (1907), p. 291.

Cleaning Barytes.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 996.

Washing Barytes.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 751. *Rev. de Metal.*, Vol. IV (1907), p. 516.

Treatment of Gypsum.—*Mines and Minerals*, Vol. XXVIII (1907), p. 59.

Grinding Scrap Mica.—"Mineral Resources of the United States" (1906), p. 1153.

Treatment of Brass Foundry and Rolling Mill Refuse.—*Brass World*, Vol. III (1907), pp. 55, 219.

Recovery of Tin from Slag.—*Oesterr. Zeits.*, Vol. LV (1907), p. 593.

Testing Ores.—*Compt. Rend.* Soc. de l'Ind. Min. (1907), p. 154.

Concentrating Tests.—Canadian Min. Inst., Vol. IX (1906), p. 143.

Testing Laboratory.—*Can. Min. Journ.*, Vol. I (1907), p. 369.

Ore Testing Plant.—*Min. and Sci. Press*, Vol. XCV (1907), p. 272.

Ore Dressing Laboratory.—Min. Soc. Nova Scotia, Vol. X (1905-06), p. 125. *Can. Min. Journ.*, Vol. I (1907), p. 176. *Min. Wld.*, Vol. XXVII (1907), p. 25.

Detection of Lost Willemite.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 803. *Rev. de Metal.*, Vol. IV (1907), p. 682.

Theory of Flotation Process.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 344.

Elmore Vacuum Oil Process.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 1205. *Oesterr. Zeits.*, Vol. LV (1907), p. 615.

Flotation Processes.—*Metallurgie*, Vol. IV (1907), pp. 522, 543.

Design of Plants.—*Min. and Sci. Press*, Vol. XCIV (1907), p. 149.

Dry Separation.

Hedburg Magnetic Separator.—*Min. Wld.*, Vol. XXVII (1907), p. 103.

Magnetic Scparator.—*Compt. Rend.* de l'Ind. Min. (1907), p. 304.

Magnetic Separation of Iron Ores In Sweden.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 889. *Rev. de Metal.*, Vol. IV (1907), p. 600.

Magnetic Concentration of Iron Ore by the Gröndal Process.—*Can. Min. Journ.*, Vol. I (1907), p. 202. *Min. Wld.*, Vol. XXVI (1907), p. 360. *Min. Reporter*, Vol. LV (1907), p. 473. *Rev. de Metal.*, Vol. IV (1907), p. 437. *Electrochem. and Met. Ind.*, Vol. V (1907), p. 134.

Magnetic Concentration in Norway.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 1206.

Magnetic Concentration at Cañon City, Colo.—*Min. Reporter*, Vol. LVI (1907), p. 268.

Roasting for Magnetic Concentration of Zinc Ores.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 613. *Rev. de Metal.* Vol. IV (1907), p. 559.

Principles of Electrostatic Separation.—*Bull.* Colorado School of Mines, Vol. III (1907), p. 268.

Dry Concentration in the Desert.—*Eng. and Min. Journ.*, Vol. LXXXIII (1907), p. 150.

Coal Washing.

Some Principles of Coal Washing.—*Can. Min. Journ.*, Vol. I (1907), p. 141.

Elliott Coal Washer and Hardy Dust Extractor.—Inst. of Min. Eng., Vol. XXXIII (1907), p. 138.

Coal Washing.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 76.

Coal Washing.—*Iron Age*, Vol. LXXX (1907), p. 96.

Removal of Coal Dust.—*Zeits. f. B., H. u. S.-wesen*, Vol. LV (1907), p. 133.

Separation of Coal Dust.—"Freiberger Jahrbuch'' 1907, p. 27A.

Magnetic Separation of Coal.—*Revue des Mines*, Ser. IV, Vol. XIX (1907), p. 270.

Anthracite Breaker at Alden, Pa.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 1216.

Bituminous Coal Washing.—*Mines and Minerals*, Vol. XXVIII (1907), pp. 7, 62.

Coal Washing in Indiana County, Pa.—*Industrial World*, Vol. XLI (1907), p. 854.

Midland Coal Washing.—*Eng. and Min. Journ.*, Vol. LXXXIV (1907), p. 16.

Coal Washing in England.—*Oesterr. Zeits.*, Vol. LV (1907), p. 455.

Baum Coal Washing Plant.—*Iron and Coal Trades Rev.*, Dec. 14, 1906. *Mines and Minerals*, Vol. XXVII (1907), p. 317.

NOTES ON THE MINING LAWS OF WESTERN AUSTRALIA.*

By A. C. Veatch.

Introduction.

The mining laws of the several Australian States and New Zealand rest on the same general principles. The law of Western Australia represents one of the most carefully matured and well balanced enactments and may well serve to give a general idea of the details of the several mining laws of the Australasian States. Besides differences in rental charges and terms of leases certain distinctions may be specially mentioned. No other State except South Australia has carried the doctrine of mining on private property for the unreserved minerals to the extent it has been carried in Western Australia. In Tasmania the law permits the expression of the development conditions in money per acre per year instead of men per acre per year as is generally required.

The present mining law of Western Australia, with the exception of a special clause in the Mining on Private Property Act of 1898, which still remains unrepealed, the Sluicing and Dredging for Gold Act of 1899, and certain Mines Regulation Acts, is contained in the Mining Act of 1904 and the regulations thereunder.

The Mining Act of 1904 is, in effect, the codification and amplification of the preceding mining law and practice and was passed just after the boom days of the great Coolgardie and Kalgoorlie finds. An amending bill, to be known as the Mines Amendment Act of 1907, is now being prepared by the Minister of Mines. The proposed amendment, however, if passed, will not in any way affect the fundamental provisions of the Mining Act of 1904, which indicates that after three years' trial the law here discussed has as a whole been found quite satisfactory. Indeed, the leasing principle, which is to Americans the most important feature of this law, receives the hearty endorsement of the mining men of Western Australia. During a recent visit to the great gold mining camp of Kalgoorlie I received from the principal mining men, including the President and Secretary of the Chamber of Mines of Western Australia, a most united and emphatic opinion that mining development is undoubtedly promoted more by a leasehold than by a freehold tenure.

* Published by permission of Director, U. S. Geol. Survey.

The proposed amending bill is, for the most part, concerned with the incorporation in the body of the law of certain regulations which have been found effective and with making several changes in details which are described below. A new section is added which has for its object the prevention of gold stealing by employees in the mines and further provisions are made for preventing mining swindles.

Among the factors which make these laws important for comparison with the statutes of other countries are: (1) In Western Australia the population is largely made up of those interested in mining. (2) Western Australia is a country of great mineral wealth, having produced in each of the last eight years between 1,500,000 and 2,000,000 oz. of fine gold or several times that produced by Alaska, and has for the same period had a greater total annual mineral production than any of the other Australian States or New Zealand, except New South Wales, which surpassed it in 1906 and 1907. (3) It contains enormous areas yet undeveloped, the State having an area of almost a million square miles,—more than the combined areas of California, Oregon, Washington, Nevada, Idaho, Utah, Colorado, Wyoming, and Montana—and a total population of less than one-twentieth of that of all these states, or less than that of the single State of Utah. (4) It is a country in which mineral lands were sold outright and its mineral laws have therefore been evolved from a basis similar to that which now is and for many years past has been, commonly accepted as the rule and practice in the United States. (5) The desire of the Government to promote and encourage the development of its mineral wealth in every way is emphatically shown by the policy of Government aid. This policy in the past has involved enormous expenditure in connection with water supplies for the mining districts; the Coolgardie Water System alone (built to pump 5,000,000 gal. a day, 351 miles to an elevation 1200 ft. above the supply point) involved an expenditure on the part of the Government of over $18,000,000.

The policy of the Government in this regard is strikingly shown in the Mining Development Act of 1902, which provides for: (a) Government loans at 5 per cent. to aid in development work. (b) Government loans to miners to aid in prospecting. (c) The erection of public crushing, ore dressing, cyaniding and smelting works and the subsidizing of persons or companies that will erect such works for testing or treating ores for the public. (d) The conduct of exploratory boring operations for water and and minerals either entirely at the cost of the Government or in connection with individuals. (e) The direct expenditure or the loan of money for constructing drainage tunnels, sinking shafts to great depths and transporting miners to undeveloped regions.

The provision for Government loans to aid miners in prospecting is not regarded by the Mines Department as having yielded entirely satisfactory

results. Loans are now made to working miners only on the security of machinery. The money invested in public batteries and exploratory boring is, however, believed to have been well spent.

The Mining Act of 1904 must, therefore, be regarded not as a theoretical attempt of political economists but as the matured law of a State which has had large practical experience in mining matters, in which, in fact, mining is the principal industry, and in which vast areas await settlement and development—a State which has, moreover, in many ways conclusively demonstrated its desire to permit and encourage the development and settlement of its territory.

Underlying Principles.

The Western Australian Mining Law of 1904 rests on two rather closely related and interlocking principles: (1) That land shall be utilized for that purpose for which it is most valuable; and (2) That no man may hold any mineral rights without development, without, in fact, so far as can be reasonably demanded, the constant employment of labor and expenditure of capital.

While encouraging and protecting individual development, there is no indication of the idea that each person who wants it is entitled to a small portion of the public domain, just so much and no more, such as is included in the coal land law of the United States, nor does it endeavor to curb monopoly, if indeed such was ever the intent of the United States coal land law, in any such clumsy fashion. There is no limit, other than that which may be fixed at the discretion of the administrating officers to the amount of land any person or persons may hold, provided they develop it, the provision that no mineral rights can be held without constant development being considered a sufficient guarantee that capital will be expended and the country's prosperity thereby promoted; and as regards injurious monopoly or combination the executive officers' discretionary power to refuse to issue a lease, the requirement of the executive officers' approval to all transfers, mortgages and the like, and the fact that all leases will some day expire give sufficient guarantee or protection.

This law in respect to the ownership of minerals by the Government reasserts the principle that "gold, silver, and other precious metals on or below the surface of all land in Western Australia" whether alienated or not, and if alienated, whensoever alienated, are the property of the Government, and that all other minerals which were not alienated in fee simple before January 1, 1899, are the property of the Government. These lands in effect belong absolutely to the local Government, and no revenue derived from them is paid to England or to the Crown. This assertion of the Government rights to the precious metals is of interest to

Americans because of the still legally undecided point whether the right to all gold and silver mines is not vested by common law in the Government in the United States.

In accordance with the doctrine that all land must be utilized for that for which it is most valuable, this law provides: (1) That after January, 1899, all minerals shall be reserved to the Government in lands alienated in any way. (2) That where minerals are found in lands where the surface rights have been alienated the Government may either (a) resume the land, paying the owner its value other than for the minerals contained or (b) permit mining provided the surface owner is indemnified for any damage resulting from such operations. (3) That as regards unreserved minerals in lands that were alienated before January 1, 1899, the Government may permit mining therefor under certain conditions.

In order to secure the entire enforcement of the doctrine that no man may hold any mineral rights without development, this law provides, in effect, that none of the minerals to which the Government has title shall be sold, but the Government may authorize the working of the mineral deposits by those willing to develop them continuously.

The enforcement of this requirement for continuous development is incompatible with the state of freehold. This is but the logical carrying out of the fundamental principle of American mining law—the possessory right. In the early history of mining in the United States, in the absence of any general law, the local mining customs and regulations recognized the right of a man to a mining claim so long as he worked it. If he abandoned his claim any other person could assert his rights thereto. This principle in a considerably distorted form has been incorporated in that makeshift composition, the Mining Law of the United States, in which the possessory right, instead of being based on continuous development, is allowed to rest on "annual assessment work" of the most meagre description, necessitating in practice only a few days' work a year, though the expenditure is supposed to amount to $100 per annum, and requiring this only until a total expenditure of $500 has been made when freehold title is given on payment of a nominal price per acre.

The Western Australian mining law for some time previous to 1904 held in respect to minerals on public lands and reserved minerals on private land that only those who would continuously develop could hold a mining tenement of any description; the law of 1904 goes a step further by applying this doctrine to all the lands in the State. It now declares, as regards private lands, where the title to certain minerals under some previous grant or law is vested in the owner of the surface, that if the owner will not work the minerals the Government will permit anyone who desires to work them to do so, will assess and collect rent or royalty or both and

pay the proceeds to the owner of the fee simple less 10 per cent. for administrative expenses.

Mining Tenements.

The various kinds of mining tenements which are now possible in Western Australia are: 1. Claims; 2. Prospecting areas; 3. Mining leases: (a) for gold; (b) for minerals other than gold or coal; (c) for coal; (d) for minerals on private lands; 4. Holdings in connection with mining: (a) residence and building areas; (b) machinery, tailings and washing areas; (c) market garden areas; 5. Water rights; 6. Miner's homestead leases; 7. Reward claims and leases.

Claims.—The miner's claim, like the miner's claim under the United States law, may be taken up on any public land not otherwise appropriated or reserved. It is essentially the old possessory right mining claim which may be held only on the condition of continuous development; the Government, however, in this case guarantees possession by registering the claim and goes further in the protection of the miner by giving the executive officers power to relieve, for limited periods, the holder of a claim from the required development work, for any good and sufficient reason. The condition of continuous development which is interpreted to mean the work of one man every working day in the year for each two men's ground contained in a lode claim until payable and for each one man's ground in all other cases, is a much more exacting requirement than the "$100 assessment work" required each year by the United States law. In practice this would mean, if applied to the United States, that the continuous work of eight men (because eight men's ground is contained therein) would be required to hold a 160-acre placer petroleum claim, or an expenditure, estimating labor at only $2 per day (in some of the States the value of a day's assessment work has been fixed by law at $4), of between $4000 and $5000 per year. A claim can, moreover, be held only by the holder of a miner's right, which may be obtained at a cost of 5s. and is good for one year, but is indefinitely renewable at that same cost per year.

As originally drafted this law contained a specific provision that only one claim could be held for each miner's right (any person or company can obtain any number of miner's rights) and so involved an automatic declaration at the end of each year of the condition of each claim. If the miner desired to hold a claim he would take out a new miner's right in respect thereto; if not, the claim lapsed and the records clearly showed that it had lapsed. This specific provision was, however, opposed in Parliament by the Labor members and was omitted, but the act as passed contains several clauses which would permit the enforcement of this sort

of a provision by regulations. The Mines Department, however, has not availed itself of the powers conferred by these clauses. In order to clear the matter up, provision for only one claim for one miner's right is to be incorporated in the Amending Act now being prepared.

Claims must be rectangular except where existing boundaries interfere. The areas considered one man's ground in the different kinds of ordinary alluvial claims are: For gold, silver and platinum, 25 by 100 yards; for non-metallic minerals, 125 by 100 yards; for precious stones, 50 by 50 yards. One man's ground in the case of lode claims may be of the following dimensions: Gold, silver, and platinum, 20 by 130 yards; all other metallic minerals, 50 by 130 yards; non-metallic minerals, 75 by 130 yards; precious stones, 50 by 130 yards. Provision is made for alluvial claims called "Extended alluvial claims" and for "river claims," two to three times the size of ordinary alluvial claims, where unusual conditions are encountered. Any number of men, not exceeding 10, may take up the proportionate multiple of one man's ground. By a supplementary regulation issued July 25, 1905, ground which has been worked and abandoned, or is suitable only for dredging, may, with the consent of the Warden, be taken up as dredging claims. A single dredging claim may not exceed 300 acres, but a union of claims under certain conditions is permitted. Claims must ordinarily be not less than 15 chains in width, but the only condition as to river claims is that they shall not exceed 6 miles in length. In a dredging claim one man must be employed for every 100 acres, or machinery whose value is not less than £1000 per 100 acres must be kept constantly in operation and in no case shall the total value of the machinery be less than £3000. This provision regarding dredging is to be incorporated in the proposed Amending Act under a special clause providing for dredging leases. The only changes made are that the area allowed is 320 instead of 300 acres and a rental of 2s. 6d. per acre per year is to be charged. The existing dredging enactment (The Sluicing and Dredging Act of 1899) referred to below is to be repealed.

Neither claim nor any other form of mineral tenements gives any extra lateral rights in Australia in the case of lode deposits, and the numerous expensive law suits which have hindered the development of the West have thus been avoided.

There is no provision compelling the holder of a claim to take out a lease in case he develops a paying mine, but in practice it is found that under such conditions most miners prefer to convert their claims into leases. For minerals other than coal the rent is greater per acre under a lease than the annual cost of a miner's right per acre of claim, even where only one claim is taken out, but the labor requirements on a lease are relatively much less than on a claim, and hence in case of misfortune it offers a more secure tenure. Moreover, the Government may at any

time by published notice exempt from further occupation by the holder of a miner's right any specified portion of the Government lands, in which case the holder of the claim is entitled to damages only for the values of any substantial buildings he may have erected on the claim. H. S. King, the Under-Secretary for Mines, states: "Though this power is given I know no case in which it has been exercised." It is opposed to the general tone of the legislation, and it is doubtful if it would ever be enforced. On the whole, the preference for a lease rather than a claim seems to be more imaginary than real. The opportunities of getting exemption for cause are just as good in a claim as in a lease, the rent is much less, and there is no limit to the period of tenure so long as the conditions are complied with. While under this legislation it is possible to take out coal claims there is no advantage in doing so in projects involving extensive development and in practice, coal operators will take out either prospecting areas or leases.

Such a system of mining claims largely prevents the staking out and holding of large areas, such as is now possible in the United States, by one or more local speculators who have no intention of developing, but who stake out and hold these claims only for the purpose of levying tribute on the bona fide developer. As these speculators can in the United States hold the claims for a year without any expenditure of capital, the practice must be regarded as permitting a serious retardation of development. It certainly is not in line with a Government policy which has for its object the encouragement of actual development and settlement.

Prospecting Areas.—The prospecting area seems to have been particularly designed to meet the needs of the prospector for coal and oil, for which the mining claim is in no wise suited. The holder of a miner's right may, with the approval of the Warden, acquire exclusive right to prospect for coal or oil over an area not exceeding 3000 acres for 12 months from date of registration of his area. This period may be extended six months but no longer. The holder of a prospecting area may not remove any mineral from the area, except samples not exceeding 50 lb. without the consent of the Warden or Mining Registrar. On discovery of mineral in paying quantities, he can obtain, according to circumstances, either an ordinary or a reward lease.

As applied to gold and minerals other than coal the prospecting area offers a tenement intermediate between a claim and a lease. It permits an area varying from 18 to 48 acres, according to circumstances, and as a rule somewhat smaller than a lease but very much larger than a claim, to be held for no more labor requirements than a claim and at very much less cost than a lease, but the holder has no right to mine and sell minerals as on a claim or lease and his rights expires absolutely in 18 months. As a means of holding a piece of land until by careful exploration an operator

can determine whether he desires a lease, this tenement, however, offers several decided advantages and has a very definite place in the scheme mining development.

Mining Leases.—No miner's right is required in taking out a lease, and this form of tenement may therefore be acquired without prospecting, but it is specially provided that no land held as a claim can be included in a lease without the claimant's consent. The maximum term for mining leases for all substances is 21 years, but provision is made for renewals for further periods of the same length "subject to the provisions of the acts and regulations in force at the time of such renewal." The area allowed in any lease varies according to the substance and conditions, but the dimensions must be such that the length shall not exceed twice the width except in the gold dredging leases provided for by the Dredging Act of 1899.

MAXIMUM AREAS WHICH MAY BE COVERED BY A SINGLE LEASE.

	Acres.
Gold mining lease.[1]	24
Gold mining lease, where ground has already been worked and abandoned or where cost of development is likely to be excessive.	48
Gold dredging or sluicing lease in swamps, lakes, and lagoons, (any part of such a lease is, however, open to entry as a claim and subject to lease for ordinary mining)[2]	5,000
Lease for minerals other than gold or coal[3]	48
Lease for minerals other than gold or coal where ground has already been worked and abandoned and where cost of development is likely to be excessive	96
Lease for precious stones	24
Coal lease ("Coal" for the purposes of this act is defined as including "Ironstone, shale, and fire clay")[4]	320
Coal leases, reward for discovery	640

Leases, the property of the same lessee, may be amalgamated under certain conditions. As to minerals other than coal the Minister may in general permit amalgamation up to 96 acres, but in special cases he may permit amalgamation to any extent he may deem necessary for the proper working of a reef or lode to a depth of 3000 ft.; such exceptional amalgamation is, however, subject to any conditions which the Minister may from time to time impose and is also subject to such restrictions of areas as he may decide is advisable if subsequent development shows

[1] Under the proposed legislation, silver, platinum, and precious stones are classed with gold and dealt with under the provisions which now apply to gold alone.

[2] This was a special enactment (The Sluicing and Dredging Act of 1899) passed under the supposition that certain alkali lagoons and swamps in the goldfields would prove dredging propositions. There has been no development under this Act and in the proposed amendment it is to be repealed and provisions similar to those now in the regulation regarding dredging claims enacted therefor. The proposed rental charge is 2s. 6d. per acre per annum and the area of a single lease 320 acres.

[3] The conditions which now apply to all minerals except gold and coal are under the proposed act made to apply to tin, copper, lead, zinc, mercury, bismuth, arsenic, antimony, nickel, cobalt, wolfram, scheelite, chromite, molybdenite, tantalite, stibiotantalite, monazite, bauxite, kaolin, asbestos, mica, minerals containing earths used in the manufacture of incandescent light mantles, minerals containing radium, phosphorite, gypsum, marble, ornamental stone, roofing slate, infusorial earth, graphite, iron pyrites. In this class amalgamation up to 192 acres is to be permitted.

[4] Under the amending act the following minerals are classed with coal: carbonaceous shale, oil shale, petroleum, iron and manganese ores, building stone, ironstone, clay, fireclay, and common salt. But provision is made for a rent of 2s. 6d. in all cases except for coal, and in substances other than coal the maximum amalgamation permitted is to be 1,280 acres. Prospecting for minerals of the first two classes in the land covered by leases of this class are to be permitted under certain conditions.

that the separate working of any lease included in the amalgamation is desirable. The Minister may likewise allow the amalgamation of coal leases provided the aggregate does not exceed 2560 acres when the seam is of ordinary depth and 5120 acres when the seam is at a depth of over 1000 ft. Amalgamation permits the satisfaction of the labor conditions for a group of adjacent leases at one point and is hence subject to cancellation on the transfer, surrender or forfeiture of any lease included in the amalgamation.

The rent and royalty assessed are as follows: In ordinary gold leases, 5s. or £1 per acre per year for the first year, as may be determined by the Governor, and £1 per acre per year thereafter; in a gold dredging lease, 6d. per acre per year and a royalty of 1s. per ounce of gold won; in an ordinary mineral lease, 5s. per acre per year, and in mineral lease of the second character, 2s. to 5s., as may be determined; in an ordinary coal lease 6d. per acre per year and a royalty on every ton of coal raised of 3d. for the first 10 years and 6d. thereafter; in a reward coal lease at 6d. per acre per year, no royalty for the first 10 years and then a royalty of 1d. per ton. There is no limit expressly fixed to the number of leases any person or company can hold, but the law provides that the Governor may refuse to issue any leases and that no lease shall be transferred, sublet, mortgaged, encumbered, or otherwise dealt with without the written approval of the Warden or Minister.

Under the conditions of continuous development it is required that a lease granted for coal or oil shall be worked every working day by not less than one man for every 60 acres or fraction thereof for the first 12 months, two men for every 60 acres for the second 12 months, and three men for every 60 acres thereafter; gold leases and mineral leases in which double area is allowed must be worked by not less than two men for every 12 acres thereafter. Ordinary gold and mineral leases must be worked by one man for every six acres after the first 12 months. Provision is made, however, for exemption from labor conditions if the holder has made reasonable efforts to work and develop the mine and is prevented from doing so by conditions beyond his control. Exemption may also be demanded as a right for certain periods by the expenditure of a given amount of capital in a certain length of time. The executive officers are in this respect given very wide discretionary powers and are able fully to protect any bona fide developer from loss through forfeiture due to no real fault of his own. In some of the earlier Australian enactments in which no provision was made for exemption from the development conditions, the labor covenant was used by labor organizations as a lever to accomplish their ends. Specific provision for exemption in case of general strikes is made in the last enactment. It also provides for exemption for six months *as a right* on the expenditure of £1500 and in the same

direction of rendering the tenure more secure, permits the warden in cases of breach of labor covenants either to recommend forfeiture or to impose a fine not exceeding £500 from which the complainant may be compensated for expenses and loss of time. In the proposed mining bill a new and very far-reaching cause for exemption is proposed in the clause which allows exemption when "owing to existing conditions it is impossible to dispose of the product of the leasehold to a profit."

The Chamber of Mines of Western Australia has for several years strongly recommended, and in this recommendation they have the concurrence of the present Minister for Mines and the Mines Department, that the condition of continuous development be expressed in money instead of men. They feel that while they have no complaint to make on the administration of the law thus far and that while under existing law the executive officers have sufficient discretionary power to enable them fully to protect the bona fide developer, there is always the danger that the Labor Party, which is now greatly in the minority, may become more powerful, and of the country having a Labor Minister for mines who might be arbitrary and unreasonable in the granting of labor exemption. This change, it is thought, would render the tenure more secure, without in any way affecting the underlying principle that all mining tenure must rest on development.

Leases authorizing the removal of any of the reserved minerals on or under private lands and mining claims to any portions of such lands may be obtained at the discretion of the Minister of Mines and subject to the payment of compensation for damages to the owner and occupier of the surface, the amount or amounts to be fixed by agreement, or, if the parties can not agree, by the Warden. The area, rent, and royalty of such leases are the same as those on the public domain. If the Government desires it may resume any such private lands on payment of fair compensation, no allowance to be made for the value of the reserved minerals. When land is resumed in this manner the rent and royalty are not subject to the general provisions of the law but may be fixed at will by the Governor.

In respect to the unreserved minerals on alienated lands, this law, carrying out the doctrine that all land should be utilized for that for which it is most valuable and that no one can hold mineral rights to the exclusion of another without development, declares that any person may petition that such land be declared open to mineral development. If on investigation the Government decides that there is a reasonable probability that the land contains minerals in paying quantities, the Minister may, in his discretion, by the publication of a formal notice, declare that at the expiration of six months from the date of such notice the land specified will be considered mineral land. If within this six months' period the owner

registers with the Department a declaration that he desires exclusive right to mine on such land or any portion thereof, the area indicated by the owner will be surveyed into lease areas of the regulation size, and the owner shall, so far as development is concerned, be held to hold the land subject to a mineral lease or leases and it shall be obligatory for such registered owner to work the land in accordance with the requirements of the mineral acts and regulations, but no rent or royalty shall be payable. If this owner does not within the six months' period register his exclusive right to mine on the area, the Government proceeds to lease the same in the usual manner, but during the currency of the lease pays all rent and royalty to the owner less 10 per cent. Thus while giving the owner a preference right in respect to mining on his own property it effectually prevents him from hindering the general development of the region.

Holdings in Connection With Mining.—In a mining region certain tenements are required which do not necessitate anything more than surface rights. The proper conservation of the interests of all parties concerned demands that special provisions be made for such holdings, which will at the same time guarantee the occupier thereof against damage to his improvements and yet will not allow such holdings to interfere with the removal of minerals, on which the prosperity of the whole settlement ultimately depends. The holder of a miner's right is therefore authorized to take up the following: A *residence area*, not exceeding ¼ acre; a *business area*, not exceeding one acre; a *market garden* area, not exceeding five acres; a *machinery area* for erecting machinery for extracting gold or minerals, not exceeding five acres; a *washing area*, for washing any earth containing gold or minerals, not exceeding five acres; a *tailings area*, for stocking and treating tailings, not exceeding five acres. For all these, except the market garden area, for which the rent is 5s. per acre per year, a rental of £1 per acre per year is charged. These areas are registered and may be held so long as they are actually used for the purpose for which they are registered. Any portion of such holding may be granted as a lease or claim, but only subject to payment of compensation for all damages In the proposed legislation all these holdings are treated as leases for a period of 21 years instead of registered holdings under a miner's right. The rentals range from 2½s. to £5 per acre per year. A new form of lease, "the tramway lease," is added.

Water Rights.—In a semi-arid region such as that containing the most important of the Western Australian mineral fields the matter of water supply largely controls the possibility of development and the mineral law and regulations practically provide for Government control of this subject. In the first place, practically all permanent water is reserved by the Government. Water may be obtained (except from one of the public or private systems) only by the holder of a water right.

Water rights are divided into (1) stream water rights; (2) lagoon, lake, spring or swamp water rights; (3) watershed or stormwater rights; (4) dam, tank or reservoir water rights; (5) subterranean water rights; (6) race or pipe track water rights. In the first two cases the amount of water that can be used is limited. Provision is made in all cases for the sale of water by the holder of a water right but the Minister reserves the right to fix prices. For a stream water right there is no rental; for a watershed water right the rental is 6d. per acre per year of the area of the watershed; for a dam, tank, or subterranean water right the rental is £1 per acre per year; for a lagoon, lake, swamp or spring water right, the rental is 6d. per 1000 gallons.

Miners' Homestead Leases.—The Governor by proclamation may create gold or mineral fields with such boundaries as he may decide, may alter the boundaries of such fields, or may abolish the fields entirely. No land in any such gold or mineral field may be disposed of under the general land laws except with the consent of the Minister of Mines. The provision in the mining law for miners' homestead leases seems to indicate that in a gold or mineral field the ordinary homestead laws are generally not operative, for it provides for a kind of tenure which is very similar to the conditional homestead purchase, and which is evidently intended as a substitute for it. The holder of a miners' homestead lease, however, acquires no title to the soil, although after paying rent for 20 years he acquires the right to hold the same indefinitely at a rental of 1s. per year for the whole area, *if the rental is demanded.* This form of tenure is evidently intended to provide for the acquisition of larger areas than are authorized for the holdings above described.

Any holder of a miner's right (either an individual or a company) may take up any number of homestead leases provided that the aggregate area taken up in any goldfield shall not exceed 20 acres, if the land is within two miles of the nearest boundary of any town site and 500 acres if beyond that distance. The rent for the first 20 years is 2s. per acre per year if the area is less than 20 acres and 6d. per acre per year if the area exceeds 20 acres. The lessee must within three years from date of the lease fence the whole of the land and within five years make improvements to the value of 10s. per acre. Such a homestead is, however, open to mineral lease or claim and is in part or whole subject to resumption on six months' notice, but in any of these cases the holder of the homestead lease is compensated for the improvements effected.

Reward Areas and Leases.—As regards coal, the law provides that the discoverer of payable coal more than 16 miles from the nearest known payable coal or at a depth exceeding 600 ft. shall be entitled to a reward lease of 640 acres free of royalty for 10 years, after which the royalty shall be but 1d. per ton. In regard to the other mineral discoveries there

is a general provision that the Minister may grant reward areas by way of lease or otherwise to the discoverer of minerals. Under this provision the discoverer is at present allowed by the regulations either to take a reward claim, which varies in area from 1 to 16 acres according to the minerals found and the distance of the discovery from other mines, or to take a reward lease, which is of the same size as an ordinary lease, but in which the rent is omitted for a period not exceeding five years, the period varying according to the distance of discovery from other mines.

Administrative and Judicial System.

The administration of the provisions of the mineral law is vested in a Minister, under whom are Wardens for each gold or mineral field, Mining Registrars, Mining Surveyors, Inspectors, Geologists, and such other officers as the Governor may deem necessary. The Wardens are both executive and judicial officers and command salaries of from $2600 to $4200 per year and quarters. It is this judicial portion of this system that is of the greatest interest to Americans because of the evident expedition which is possible thereunder. The Warden's court has in effect jurisdiction over all matters relating to mining tenements and mining; its proceedings are similar to the local courts and its judgments enforced in a like manner; it may order mines or minerals seized by bailiff or other officer until further order of court; it may inspect any mine or mining tenement and "may take judicial notice of anything observed" or it may order such inspection, may issue injunctions, may procure witnesses by means of subpoena, may punish for contempt by fine or imprisonment, may order sale under writ of execution, etc. Appeals, from the decisions of the Warden's court lie to the Supreme Court, but no appeal is permitted: (1) If the parties agree that the Warden's decision shall be final. (2) If the value of the matter or interest in dispute does not exceed £200, except by permission of the Supreme Court or a Judge. (3) From any decision, order, or recommendation of the Warden upon any application for a mining tenement, the forfeiture thereof, or exemption from labor or other conditions. (Except to the Minister of Mines in case the final decision rests with him, as it usually does.)

This gives ample and effective judicial powers to the officers who are charged with administration of the law and who are on the ground. Under the American system a limited amount of judicial power is vested in the Land Office under the Secretary of the Interior in respect to lands so long as they remain in the hands of the United States, but this is very different from the judicial powers here vested in the officers of the Department of Mines. The intricate relation of State and Nation, of state courts and federal courts, of course, greatly complicate the situation in this respect

in the United States, but it must be conceded that the effective and rapid judicial administration of any extensive mineral leasing system in the United States is certainly not one of the least matters deserving earnest consideration in connection with any adequate mineral land legislation.

Conclusion.

The Western Australian mining law is, in short, a wonderfully symmetrical and carefully balanced enactment; and while one may not regard it as applicable in all its features to American conditions, it contains many suggestive provisions, all of which merit careful consideration. They can not in any way be considered the idle visions of the theorist, but are the mature enactments of a legislature whose members are entirely chosen by voters of a great democratic mining State—a State which ranks among the great mining states of the World, and which has, as recently as 1904, reorganized and revised its mining laws to meet the practical workaday conditions of that region.

AUSTRALASIA.

In the following tables the production of minerals and metals in each of the Australian States and New Zealand is separately itemized. In the tables relating to foreign commerce, however, the states are not separately treated, the combined statistics of the Commonwealth now being officially reported.

MINERAL PRODUCTION OF NEW SOUTH WALES. (a)
(In metric tons or dollars; £1=$5) (b)

Year.	Alunite.	Antimony and Ore.	Bismuth Ore.	Chrome Ore.	Coal.	Coke.	Cobalt Ore.	Copper Ore.	Copper Matte, Ingot and Regulus.
1896	1,394	134	42	3,914	3,972,069	26,774	...	15	4,464
1897	736	172	3	3,433	4,453,729	65,229	...	169	6,458
1898	2,988	83	29	2,145	4,781,551	83,538	119	181	5,577
1899	935	332	16	5,327	4,670,580	98,074	193	445	5,574
1900	1,946	252	11	3,338	5,595,879	128,238	145	867	6,243
1901	3,196	90	21	2,523	6,063,921	130,944	112	655	6,184
1902	3,702	58	10	508	6,037,083	128,902	35	3,190	5,560
1903	2,524	13	23	1,982	6,456,523	163,161	155	1,750	8,094
1904	376	111	41	404	6,116,126	173,742	6	2,470	6,654
1905	2,745	394	56	53	6,738,252	165,568	*Nil.*	487	7,899
1906	1,886	2,490	25	15	7,748,384	189,038	*Nil.*	(h)	9,911
1907	2,021	1,780	17	30	8,796,451	258,683	*Nil.*	(h)	10,260

Year.	Diamonds. *Karats.*	Gold. (b)	Lead. Argentiferous. (g) Ore.	Lead. Argentiferous. (g) Metal. (f)	Lead. Pig. (g)	Molybdenite.	Opal.
1896	8,000	$5,222,971	271,641	19,886	24	..	$225,000
1897	9,189	5,373,596	275,249	18,395	32	..	375,000
1898	16,493	5,847,680	394,676	10,270	1,745	..	400,000
1899	25,874	7,899,075	431,126	20,614	(e)4,896	..	675,000
1900	9,828	5,211,097	426,480	19,400	(e)6,807	..	400,000
1901	9,322	3,587,040	406,560	17,191	(e)3,394	..	600,000
1902	11,995	3,333,064	371,496	15,660	(e)4,685	16	700,000
1903	12,239	5,255,421	335,870	18,779	(e)3,561	31	500,000
1904	14,296	5,576,966	373,362	30,212	(e)5,977	26	285,000
1905	6,354	5,669,099	420,266	28,244	214	20	295,000
1906	2,827	5,249,762	377,890	22,573	60	34	282,500
1907	2,539	5,112,852	441,024	(i)	20,084	22	395,000

Year.	Platinum. Kg.	Shale Oil.	Silver—Kg. (g)	Stone, Limestone Flux.	Tin. Ore.	Tin. Block.	Tungsten Ore.	Zinc. (d) (g)
1896	75.8	32,348	6,307	90,347	98	1,147	..	.. ...
1897	61.2	34,635	4,666	68,671	14	799	..	29,303
1898	38.9	30,164	16,580	9,401	1	639	..	39,564
1899	19.8	37,307	21,525	1,016	5	749	.	50,677
1900	15.6	23,229	24,080	17,273	15	1,087	..	20,594
1901	12.1	55,650	13,950	26,995	11	659	..	642
1902	11.6	63,886	33,195	17,630	23	502	.	1,281
1903	16.5	35,332	34,195	24,205	556	949	9	21,086
1904	16.6	38,477	34,880	25,374	586	1,084	106	58,523
1905	12.4	38,838	12,987	15,180	726	817	228	105,189
1906	6.4	32,965	8,865	12,993	(h)	1,698	245	105,325
1907	8.6	48,088	63,573	42,334	(h)	1,945	409	241,015

(a) From the Annual Report of the Department of Mines, New South Wales. (b) Where gold is reported £1=$4.866. (d) Spelter and concentrate. (e) Includes minor quantities of lead carbonate and chloride, the product of the leaching plant at Broken Hill. (f) Includes a small quantity of silver-sulphide. (g) Exported. (h) Included with metal. (i) Included with ore.

MINERAL PRODUCTION OF NEW ZEALAND. (a) (b)
(In metric tons or dollars; £1=$5.) (c)

Year.	Antimony Ore.	Chrome Ore.	Coal.	Coke.	Copper Ore.	Gold. (c)	Kauri-gum.	Manganese Ore.	Silver—Kg.
1896	21	....	805,537	107	..	$5,067,589	7,240	63	2,933.3
1897	10	....	854,164	..	..	4,769,673	6,748	183	5,719.8
1898	..	....	921,546	9	2	5,258,642	10,063	220	9,140.0
1899	..	....	990,838	18	..	7,363,100	11,294	137	10,865.6
1900	3	28	1,111,860	..	12	7,005,103	10,322	166	10,202.0
1901	30	....	1,259,521	..	3	8,533,908	7,662	211	17,762.0
1902	..	128	1,386,881	..	..	9,495,673	7,549	...	20,970.3
1903	..	....	1,542,953	..	6	9,916,086	9,507	71	28,364.3
1904	..	....	1,537,838	..	..	9,671,180	9,203	196	34,042.3
1905	..	....	1,585,756	15	4	10,189,093	10,883	55	36,695.0
1906	..	....	1,613,301	5	...	11,050,219	9,300	16	43,251.5

(a) From New Zealand Mines Statement, by the Hon. James McGowan, Minister of Mines, Wellington. (b) The exports are stated to be identical with the production, with the exception of coal, the exports of which were as follows: In 1896, 80,796 tons; in 1897, 77,280 tons; in 1898, 57,333 tons; in 1899, 90,912 tons; in 1900, 116,216 tons; in 1901, 162,197 tons; in 1902, 191,696 tons; in 1903, 154,769 tons; in 1904, 165,220 tons; in 1905, 122,817 tons; in 1906, 141,641 tons. (c) Where gold is reported £1 =$4.866.

MINERAL PRODUCTION OF QUEENSLAND. (a)
(In metric tons or dollars; £1=$5.)

Year.	Bismuth Ore.	Coal.	Copper Ore.	Gems other than Opal.	Gold. (d)	Lead.	Manganese Ore.
1895	60	328,237	441	(b)$29,575	$13,056,414	369	361
1896	..	377,332	589	(c)	13,235,842	628	305
1897	1	364,142	293	(c)	16,699,477	391	403
1898	8	414,461	63	(c)	19,016,763	252	68
1899	2	501,913	164	(c)	19,571,662	57	747
1900	8	505,252	386	4,500	20,002,290	207	77
1901	20	548,104	3,110	30,000	12,367,276	570	221
1902	1	509,579	3,845	25,000	13,238,500	271	4,674
1903	11	515,950	4,995	35,000	13,818,653	3,856	1,341
1904	20	520,232	4,440	52,875	13,210,869	2,079	843
1905	15	537,795	7,337	26,275	12,249,157	2,464	1,541
1906	7	610,480	10,238	90,550	11,257,316	2,854	1,131
1907	6	694,204	12,959	202,500	9,641,789	5,240	1,134

Year.	Molybdenite.	Opal.	Silver *Kg.*	Stone, Building. (*b*).	Tin Ore.	Tungsten Ore.
1895	....	$163,750	6,999	52,206	2,148	25
1896	....	116,500	8,687	(*c*)	1,579	3
1897	....	51,250	7,280	(*c*)	1,222	13
1898	....	33,225	3,235	(*c*)	1,041	79
1899	....	45,000	4,521	164.939	1,322	263
1900	....	37,500	3,514	152,484	1,133	193
1901	....	37,000	17,777	(*c*)	1,638	73
1902	(*e*) 42	35,000	21,813	139,338	2,118	56
1903	(*e*) 24	36,500	19,972	107,780	3,768	200
1904	(*e*) 22	17,750	20,370	72,841	3,986	1,564
1905	64	15,000	18.716	177.912	4,008	1,434
1906	108	15,000	24.357	(*f*)	4,900	785
1907	68	15,000	28,662	(*f*)	5,222	627

(*a*) From *Annual Reports of the Under Secretary of Mines*, Queensland, when not otherwise stated. (*b*) From *Mineral Statistics of the United Kingdom.* (*c*) Not reported. (*d*) Where gold values are reported, £1=4.866. (*e*) Includes bismuth and tungsten. (*f*) Returns not available.

MINERAL PRODUCTION OF SOUTH AUSTRALIA. (*a*)

(In metric tons or dollars: £1=$5.) (*b*)

Year.	Copper.		Gold. (*b*)	Iron Ore.	Lead.	Limestone.	Salt.	Other Metals and Minerals.
	Ore.	Metal.						
1896	354	4,176	$69,827	....	45	....	....	$3,775
1897	554	4,267	189,871	....	74	....	....	14,340
1898	545	4,327	51,949	....	286	....	....	7,550
1899	2,938	4,985	75,822	....	330	....	....	6,785
1900	2,405	4,432	70,528	....	347	....	....	89,835
1901	1,896	6,140	80,839	....	....	....	....	67,830
1902	2,620	6,210	121,056	....	1,973	....	....	102,160
1903	7,182	5,886	139,411	86,291	653	....	40,640	10,855
1904	3,100	5,694	369,938	47.434	....	44,135	40,640	990
1905	2,604	5.939	223,121	85.335	47	45,210	33,020	6,305
1906	535	8,540	284,432	76,430	45	32,451	55,880	6,315

(*a*) From *Review of Mining Operation* by Hon. L. O'Loughlin, Adelaide, 1906. (*b*) Where gold is reported £1=$4.866.

MINERAL PRODUCTION OF TASMANIA (*a*)

(In metric tons or dollars; £1=$5.)

Year	Coal.	Copper Ore and Matte.	Gold. (*e*)	Iron Ore.	Lead-Silver Ore.	Stone.			Tin and Tin Ore.
						Limestone.	Freestone, Flagstone, and Building Stones. *Cubic Feet.*	Rubble or Metal.	
1896	44,286	52	$1,156,035	203	21,150	2,621	13,575	(*g*)4,556	3,867
1897	43,210	113,261	1,407,447	999	17,806	1,702	82,197	13,274	3,282
1898	49,902	(*b*)	1,369,706	1,296	196,707	45,324	19,560	70,701	2,882
1899	43,803	(*c*)60,985	1,593,834	6,726	424,552	71,747	14,400	12,060	3,333
1900	51.549	(*d*) 4,221	1,538,727	5,141	453,579	47,671	51,509	4,291	2,693
1901	49,963	(*d*)11,401	1,436,326	1,422	804,463	26,545	37,579	244,721	2,516
1902	49,647	(*d*) 8,630	1,467,454	2,424	47,226	(*f*)	(*f*)	(*f*)	1,989
1903	49,856	(*d*) 3,891	1,237,925	6,076	43,103	(*f*)	(*f*)	(*f*)	2,414
1904	62,090	8,826	1,362,587	6,950	51,959	(*f*)	(*f*)	(*f*)	2,104
1905	52,825	9,919	1,520,101	6,401	76,424	(*f*)	(*f*)	(*f*)	3,953
1906	53.742	11,114	1,240,650	2,642	88.513	(*f*)	(*f*)	(*f*)	4,545
1907	59,833	9,180	1,350,836	3,048	91,199	(*f*)	(*f*)	(*f*)	4,412

(*a*) From *Statistics of the Colony of Tasmania.* (*b*) Included with lead-silver ore. (*c*) In addition there were produced 43 tons of copper bullion. (*d*) In 1900 there were produced 9343 tons of blister copper; in 1901, 10,141 tons; in 1902, 7,869 tons and in 1903, 6,791 tons. (*e*) Where gold values are reported £1 =$4.866. (*f*) Not reported. (*g*) Represents cart-loads.

MINERAL PRODUCTION OF WESTERN AUSTRALIA. (*a*)
(In metric tons or dollars.)

Year.	Anti-mony.	Coal.	Copper Ore.	Gold. (*b*) (*c*)	Iron Ore.	Lead Ore.	Lime-stone.	Silver. *Kg.*	Tin Ore.
1900	..	120,305	6,282	$27,461,865	12,448	272	16,183	894	836
1901	..	119,721	10,319	32,698,941	20,898	(*d*)21	18,501	1,893	746
1902	..	143,145	2,298	37,026,119	4,877	(*d*)36	5,162	2,590	630
1903...........	22	135,568	20,854	40,560,927	224	*Nil.*	1,301	5,229	830
1904...........	..	140,773	4,033	39,557,933	1,465	*Nil.*	13,612	12,416	869
1905......	..	129,402	2,389	38,045,366	3,264	*Nil.*	9,291	11,189	1,096
1906	*Nil.*	152,151	7,548	35,888,278	1,300	*Nil.*	9,624	9,071	1,518
1907...........	*Nil.*	144,651	19,282	34,579,849	1,112	10	3,660	.....	1,650

(*a*) From the *Report of the Department of Mines of Western Australia.* (*b*) £1=$4.866. (*c*) The value of gold produced in 1895 was $4,280,855; in 1896, $5,200,821; in 1897, $12,481,176; in 1898, $19,418,735. (*d*) Silver-lead ore.

MINERAL PRODUCTION OF VICTORIA. (*a*)
(In metric tons or dollars.)

Year.	Coal.	Lignite.	Gold. (*c*)	Stone, Building, etc.	Tin Ore.
1896.	230,187	5,908	$16,640,997	$485	47
1897	240,057	4,894	16,799,824	(*e*)125,000	48
1898...........	246,845	2,915	17,305,547	100,000	87
1899..........	266,578	(*b*)	17,662,410	(*b*)	158
1900	215,052	(*b*)	16,767,261	175,000	71
1901	212,678	152	16,320,029	225,000	78
1902	228,777	(*b*)	14,899,876	266,975	10
1903..........	65,230	5,752	15,860,815	213,245	34
1904.........	123,695	*Nil.*	15,824,952	1,488,075	72
1905..........	157,648	*Nil.*	15,443,438	(*b*)	126
1906...........	163,201	*Nil.*	15,962,804	(*b*)	108

(*a*) From Annual Reports of the Secretary for Mines of the Colony. (*b*) Not reported. (*c*) Where gold values are reported, £1=$4.866. (*e*) Estimated value.

MINERAL IMPORTS OF AUSTRALIA. (*a*)
(In metric tons, cwts. of 112 lb. or dollars; £1=$5.) (*b*)

Year.	Alkalies.	Brass Manufactures.	Bricks, Fire and Glazed.	Britannia, Yellow Metal, Etc.	Cement. *Cwts.*	Chemicals.	Chinaware and Earthenware.	Coal.	Coke.
1900...	$395,870	$322,230	$25,890	$217,765	1,182,442	$4,457,060	$1,347,070	7,714	44,169
1901...	418,955	336,170	49,755	142,240	1,422,647	4,495,020	1,616,645	10,141	36,814
1902...	401,225	218,980	50,115	162,510	1,074,482	4,237,505	1,358,275	5,149	9,846
1903...	427,540	99,600	22,645	135,425	954,606	3,926,610	940,170	389	4,294
1904...	450,085	72,425	35,525	120,110	561,237	4,126,245	1,151,520	398	4,270
1905...	557,370	109,735	28,900	63,920	700,245	2,571,755	1,068,090	7,866	5,553
1906...	712,475	169,310	44,770	150,450	793,928	1,762,700	1,216,945	706	6,202

Year.	Copper.		Glass and Glassware.	Gold. (*b*)				
	Ore. *Cwts.*	Manufactures.		Ore.	Bullion.	Specie.	Foil. (*c*)	Total Value.
1900...	31,386	$249,735	$1,900,875	$ 14,880	$4,556,007	$ 78,888	$51,224	$4,700,999
1901...	14,520	294,870	1,807,805	37,473	3,709,848	18,053	34,704	3,800,078
1902...	29,236	335,105	1,508,530	2,375,513	3,834,510	505,899	30,028	6,740,950
1903...	5	338,915	1,257,860	66,908	5,935,800	6,530	38,680	6,047,918
1904...	12	325,560	1,398,735	68,309	5,684,164	6,297	43,215	5,801,985
1905...	80	226,625	1,379,490	103,709	7,067,534	422,127	52,144	7,645,514
1906...	873	441,955	1,611,485	93,116	10,053,463	397,990	53,356	10,597,925

Year.	Graphite. *Cwts.*	Iron. Bars, Rods, Girders, Sheets etc. *Cwts.*	Iron. Galvanized Plates and Sheets. *Cwts.*	Iron. Pig and Scrap. *Cwts.*	Iron and Steel. Tin Plate.	Iron and Steel. Manufactures.	Iron and Steel. Pipe and Tubes	Iron and Steel. Railway Material.
1900	3,020	2,223,731	983,399	985,265	$1,520,585	$5,182,730	$1,500,850	$3,888,325
1901	3,419	2,081,423	905,709	732,512	1,262,720	7,865,560	1,284,830	6,395,705
1902	2,659	....	1,104,701	766,725	1,393,045	7,967,025	1,005,975	6,318,475
1903	5,557	1,211,437	886,570	989,998	805,810	3,802,190	1,193,540	2,472,940
1904	4,263	1,399,783	1,027,859	883,397	879,755	2,526,750	1,130,185	920,180
1905	4,386	1,482,334	1,112,467	940,757	748,835	7,407,215	1,293,390	1,030,455
1906	6,531	1,878,851	1,245,211	1,220,236	1,292,415	10,836,850	1,764,255	1,702,175

Year.	Jewelry and Precious Stones.	Lead Mfrs. *Cwts.*	Paints and Colors.	Petroleum Products. Kerosene—*gal.*	Petroleum Products. Naptha—*gal.*	Petroleum Products. Paraffin.	Potassium Nitrate. *Cwts.*	Platinum.
1900........	$1,570,560	....	$1,570,880	11,125,905	48,863	1,275	8,142	....
1901........	1,990,570	....	1,551,270	20,924,640	114,092	1,040	6,559	....
1902........	1,756,450	8,300	1,293,960	10,399,931	116,170	1,913	7,955	....
1903........	2,024,275	9,525	1,092,420	15,009,609	127,445	2,163	4,659	$9,255
1904........	2,234,375	6,243	1,299,875	14,791,319	277,737	530	7,812	910
1905........	2,115,955	8,859	1,417,670	16,416,734	292,670	....	9,010	1,875
1906........	2,581,500	14,830	1,403,605	15,473,570	488,961	....	8,112	3,980

Year.	Quick-silver.	Salt. *Cwts.*	Silver. (*b*) Ore. *Cwts.*	Silver. (*b*) Bullion. *Kg.*	Silver. (*b*) Specie.	Stone, including slate and marble.	Sulphur. cwts.	Zinc. Bar and Old.	Zinc. Spelter. *Cwts.*
1900....	63.2	486,457	190	190.4	$1,226,208	$364,705	109,647	213,800	13,582
1901....	91.0	560,560	16,385	14.9	772,020	330,185	99,270	154,555	14,291
1902....	92.6	571,548	5,562	13.6	439,186	403,185	173,176	132,725	20,965
1903....	87.5	312,681	....	14.2	160,111	353,465	180,719	158,940	14,197
1904....	92.6	355,599	....	39.8	154,534	393,135	252,744	134,665	23,316
1905....	82.1	492,727	....	3908.0	261,397	301,350	177,304	154,775	26,211
1906....	78.6	326,042	380	9756.4	703,820	323,765	269,704	175,710	24,233

(*a*) From Trade and Customs Returns, Commonwealth of Australia. Previous to 1900 each Colony reported its own imports and exports. (*b*) Where gold, silver or platinum values are reported, £1=$4.866. (*c*) Includes a small quantity of silver foil.

MINERAL EXPORTS OF AUSTRALIA (*a*).
(In metric tons, cwts. of 112 lb., or dollars; £=1$5).

Year.	Alunite. *Cwts.*	Antimony Ore. *Cwts.*	Bismuth Ore. *Cwts*	Cement *Cwts.*	Chrome Ore. *Cwts.*	Coal.	Coke.	Cobalt. Ore. *Cwts.*	Copper Ore. *Cwts.*	Copper Ingot and Matte. (*Cwts.*)	Glass and Glass ware.
1900..	38,300	5,197	194	....	48,300	1,774,980	6,005	2,865	90,589	350,362	$28,650
1901..	62,920	2,206	993	....	41,035	1,750,066	4,465	2,212	231,644	389,041	30,855
1902..	72,880	1,428	136	....	10,000	1,687,621	6,080	748	165,149	464,715	58,325
1903..	49,690	947	832	11,168	39,022	2,063,016	27,345	3,060	61,569	616,277	61,360
1904..	7,400	2,177	1,918	26,305	7,941	1,637,113	2,771	167	90,098	540,998	44,245
1905..	54,040	7,811	2,222	17,283	(*f*)	2,058,190	2,316	1,320	17,380	632,183	117,680
1906..	37,120	66,188	1,574	39,737	(*f*)	2,094,793	11,382	......	33,476	744,357	86,715

Year.	Gold. (c)				Iron and Steel.		Jewelry and Precious Stones.	Lead.		
	Ore.	Bullion.	Specie.	Total Value.	Bars, Rods, Scrap etc. *Cwts.*	Manufactures.		Pig and Matte *Cwts.*	Argentiferous. *Cwts.*	Manufactures. *Cwts.*
1900..	$2,379	$19,604,657	$41,898,304	$61,505,340	6,263	$16,965	$290,675	379,259	655,129	21,797
1901..	65,341	22,416,198	43,233,515	65,715,054	4,396	59,150	323,750	281,391	668,955	22,611
1902..	1,214,208	20,736,800	41,954,939	63,905,947	3,182	29,305	345,195	365,830	638,359	17,429
1902..	80,591	29,691,889	53,634,629	83,407,109	5,753	40,290	371,375	633,816	553,308	28,783
1904..	46,894	27,073,767	49,284,833	76,405,494	4,952	57,285	380,345	1,626,292	790 435	20,552
1905..	49,507	25,788,574	27,523,288	53,361,369	4,821	80,635	1,013,595	1,302,428	753,008	34,629
1906..	20,296	24,113,950	47,937,681	72,071,927	11,560	22,375	1,727,860	1,031,605	781,426	20,358

Year.	Molybdenum Ore. *Cwts.*	Paints and Colors.	Platinum.	Salt. *Cwts.*	Shale Oil.	Silver.		Stone including Marble & Slate	Tin.		Zinc.	
						Ore (g) *Cwts.*	Bullion, *Cwts.*		Ore. *Cwts.*	Block. *Cwts.*	Bar. and Old	Spelter. *Cwts.*
1900..	......	$2,990	......	100,893	16,792	1,598,789	192,328	$5,470	6,815	72,172	$3,440	37,352
1901..	......	3,100	......	156,760	19,587	1,630,252	196,136	24,920	5,012	60,129	1,690	1,732
1902..	160	4,870	......	238,192	27,896	1,439,374	189,703	35,545	10,291	63,424	2,980	4,461
1903..	783	12,940	$5,163	155,613	14,483	1,653,794	202,730	10,130	26,900	82,473	20,465	60,206
1904..	1,100	6,725	5,207	141,553	8,202	2,235,385	227,972	10,450	40,339	99,476	7,420	309,422
1905..	1,381	58,120	7,545	174,987	11,818	581,651	208,134	8,140	55,153	108,963	15,565	3,006,372
1906..	1,867	9,830	4,800	198,851	7,203	1,010,707	174,457	11,400	51,793	130,120	19,355	2,592,018

(*a*) From "Trade and Customs Returns," Commonwealth of Australia, 1907.—Note. Previous to 1900 each Colony reported its own exports separately. (*b*) Included with ingots and matte. (*c*) Where gold, platinum or silver values are reported £1 = $4.866. (*d*) Includes a small quantity of scrap. (*e*) Included under silver ore. (*f*) Included with iron ore. (*g*) Includes lead ore.

AUSTRIA-HUNGARY.

In the following tables the mineral and metal productions of the two Kingdoms are reported separately, together with that of Bosnia and Herzegovina.

MINERAL AND METALLURGICAL PRODUCTION OF AUSTRIA. (a)
(In metric tons.)

Year.	Alum.	Alum and Pyritic Shale.	Antimony.		Asphaltic Rock.	Bismuth Ore.	Coal.	
			Ore.	Metal.			Bituminous.	Lignitic.
1895...	885	5,716	695	296	404	185.0	9,722,679	18,389,147
1896...	919	25,184	905	422	390	*Nil*	9,899,522	18,882,537
1897...	851	21,585	864	425	300	1.0	10,492,771	20,458,093
1898...	1,037	28,914	679	343	643	*Nil*	10,947,522	21,083,361
1899...	604	19,879	410	271	2,635	0.3	11,455,139	21,751,794
1900...	620	3,004	201	153	887	4.0	10,992,545	21,539,917
1901...	442	2,551	126	114	541	16.0	11,738,840	22,473,510
1902...	62	2,866	18	24	897	8.0	11,045,039	22,139,683
1903...	*Nil*	2,978	41	14	1,273	10.0	11,498,111	22,157,521
1904...	*Nil*	2,337	103	36	1,435	1.7	11,868,245	21,987,651
1905	*Nil*	1,657	1,673	90	4,363	1.7	12,585,263	22,692,076
1906	*Nil*	1,020	1,071	*Nil*	2,840	2.7	13,473,307	24,167,714

Year.	Copper			Cop-peras.	Gold.		Graphite.	Iron.	
	Ore.	Metal.	Sulphate.		Ore.	Bullion.		Ore.	Pig & Cast
1895	7,435	865	246	160	104	$49,841	28,443	1,384,911	660,549
1896	6,823	1,001	265	170	416	46,386	35,972	1,448,615	693,188
1897	7,405	1,083	276	125	647	44,924	38,504	1 613,876	762,685
1898	6,791	1,041	209	360	448	47,515	33,062	1,733,649	837,767
1899	6,731	1,123	235	475	387	50,306	31,819	1,725,143	872,352
1900	5,825	881	234	474	227	47,183	33,663	1,894,458	879,132
1901	7,406	776	256	472	143	31,234	29,992	1,963,246	884.844
1902	8,455	914	248	271	74	4,652	29,527	1,742,498	991,827
1903	12,688	961	310	298	2,148	5,316	29,590	1,715,984	970,832
1904	10,701	889	808	414	12,653	47,183	28,620	1,719,219	988,364
1905	10,677	870	540	116	35,937	133,218	34,416	1,913,782	1,119,614
1906	20,255	877	578	154	33,033	83,401	38,117	2,253,662	1,222,230

Year.	Lead.			Manganese Ore.	Mineral Paint.	Petroleum.	Quicksilver.		Salt.
	Ore.	Pig.	Litharge.				Ore.	Metal.	
1895	12,919	8,085	2,034	4,352	3,164	188,634	86,683	535	278,875
1896	14,563	9,769	1,738	3,950	3,979	262,356	83,305	564	308,933
1897	14,145	9,680	1,626	6,012	3,653	275,204	88,238	532	331,084
1898	14,363	10,340	1,520	6,132	3,213	323,142	88,519	491	341,959
1899	12,820	9,736	1,526	5,411	2,055	309,590	92,323	536	342,059
1900	14,314	10,650	1,288	8,804	2,828	347,213	94,747	510	330,277
1901	16,688	10,161	1,317	7,796	1,701	404,662	97,360	525	333,238
1902	19,055	11,264	1,023	5,646	1,486	520,845	90,040	511	311,806
1903	22,196	12,162	923	6,179	1,691	672,508	83,321	523	359,015
1904	22,514	12,645	783	10,189	1,829	....	88,279	536	369,877
1905	23,339	12,968	865	13,788	798	....	86,856	520	343,375
1906	19,683	14,846	1,059	13,402	943	....	91,494	526	378,912

Year.	Silver.		Sulphuric Acid.	Sulphur Ore.	Tin.		Tungsten Ore.	Uranium.		Zinc.	
	Ore.	Bullion. (*Kg*)			Ore.	Block.		Ore.	Salts.	Ore.	Spelter.
1895	18,113	40,081	7,431	830	24	60	35	31	4.5	25,862	6,456
1896	18,701	39,904	7,972	643	15	54	22	30	4.2	26,887	6,888
1897	20,628	40,026	8,515	530	16	48	31	44	4.4	27,463	6,236
1898	20,886	40,304	7,003	496	13	48	36	51	4.3	27,395	7,302
1899	21,554	39,564	7,814	555	54	41	50	49	7.6	37,100	7,192
1900	21,641	39,572	7,067	862	51	40	50	52	11.3	38,243	6,742
1901	21,363	40,205	7,073	4,911	42	49	45	48	13.0	36,072	7,558
1902	22,288	39,544	8,781	3,721	47	50	45	46	10.0	31,927	8,309
1903	21,958	39,812	9,105	4,475	57	34	49	45	6.0	29,544	8,949
1904	21,949	39,032	8,742	6,288	77	38	52	17	11.0	29,226	9,159
1905	21,047	38,453	1,007	8,407	52	53	55	16	13.9	29,983	9,326
1906	21,944	38,940	745	15,125	55	42	56	16	16.1	32,037	10,804

(*a*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.*

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*)

(In metric tons or dollars; 1 crown=$0.203.)

Year.	Antimony.		Asphalt.	Asphaltic Rock.	Bismuth	Carbon Bisulphide.	Coal.			
	Ore.	Regulus.					Bituminous. (*e*)	Lignite. (*e*)	Coke.	Briquets.
1895	1,240	465	2,285	....	....	237	1,068,046	3,517,901	12,033	29,421
1896	1,361	500	2,740	....	....	352	1,132,625	3,761,728	25,550	31,179
1897	1,800	523	3,057	....	4.7	432	1,118,024	3,870,530	(*d*)	27,022
1898	2,201	855	3,125	....	3.1	771	1,239,498	4,516,581	(*d*)	31,781
1899	1,965	940	3,060	....	3.0	1,120	1,238,855	4,292,584	10,336	31,137
1900	2,373	846	2,700	....	2.0	1,250	1,447,047	5,128,277	12,973	69,353
1901	(*b*) 323	706	2,878	25,161	1.6	2,087	1,365,270	5,179,829	10,975	40,182
1902	(*b*) 748	683	2,774	24,873	0.9	2,320	1,162,785	5,132,053	8,204	88,069
1903	(*b*) 205	732	2,422	21,552	1.5	2,357	1,233,410	5,271,781	9,442	101,197
1904	1,080	1,007	2,221	17,660	0.9	2,512	1,155,320	5,519,349	5,103	103,481
1905	949	756	173	19,372	1.4	2,760	1,088,087	6,088,578	69,303	144,697
1906	1807	954	4,111	34,664	2.0	2,756	1,237,730	6,365,214	79,930	151,657

Year.	Copper.	Copperas.	Gold.	Iron.			Lead.		Litharge.	Manganese Ore.
				Ore. (*e*)	Pig.	Cast.	Ore.	Pig.		
1895	286	521	$2,118,100	9,955,262	322,206	....	.....	2.277	615	3,525
1896	159	595	2,131,876	1,269,680	383,698	....	.....	1,911	465	2,101
1897	213	592	2,038,839	1,421,130	402,503	....	525	2,527	155	4,030
1898	153	745	1,839,474	1,666,837	448,621	20,784	771	2,305	188	8,087
1899	165	771	2,039,504	1,587,600	451,637	19,631	526	2,166	213	5,073
1900	181	700	2,173,079	1,666,363	432,817	22,738	612	2,030	201	5,746
1901	162	805	2,189,692	1,557,300	430,686	20,640	(*b*) 10	2,029	238	4,591
1902	89	909	2,260,135	1,562,238	416,835	18,569	(*b*) 20	2,244	219	7,237
1903	45	982	2,243,521	1,439,132	396,674	18,875	(*e*)3,698	2,057	257	5,311
1904	63	1,277	2,437,998	1,524,036	370,297	17,203	(*e*)3,922	2,104	710	11,527
1905	73	920	2,439,451	1,661,358	403,719	17,563	686	2,146	209	5,943
1906	69	1,306	2,487,156	1,698,291	402,527	17,164	564	1,925	698	10,895

Year.	Mineral Paints.	Petroleum	Pyrites.	Quicksilver *Kg.*	Salt.	Silver. *Kg.*	Sulphur	Sulphuric Acid.	Zinc.	
									Ore. (*b*)	Spelter
1895	371	2,083	69,195	1,129	169,395	20,432	102	4,223	(*d*)	..
1896.....	334	2,168	52,697	1,100	180,133	19,916	138	3,550	(*d*)	..
1897.....	460	2,229	44,454	700	193,463	26,790	112	3,397	30	..
1898.....	247	2,471	58,079	6,800	197,593	18,799	93	1,318	30	..
1899.....	394	2,125	79,519	27,000	200,525	20,991	116	1,463	1,197	..
1900.....	370	2,199	87,000	31,800	212,957	20,202	123	1,371	326	..
1901.....	305	3,296	93,907	33,003	215,581	23,636	137	1,464	693	14
1902.....	283	4,347	106,490	44,600	217,079	23,020	105	1,193	364	..
1903.....	263	3,010	96,619	43,700	214,536	19,281	135	1,543	46	26
1904.....	273	2,134	97,148	45,169	230,943	16,352	143	1,329	203	..
1905.....	196	471	106,848	36,000	238,642	15,946	135	1,410	173	..
1906.....	221	2,692	112,623	50,100	245,402	13,642	133	1,457	243	146

(*a*) From the *Annuaire Statistique Hongrois.* (*b*) Includes only that part of the crude output that was not smelted into a refined product. (*d*) Not reported. (*e*) Total production

MINERAL AND METALLURGICAL PRODUCTION OF BOSNIA AND HERZEGOVINA. (*a*)
(In metric tons.)

Year	Chrome Ore.	Copper.		Iron.		Lignite.	Manganese Ore	Pyrites.	Quicksilver.	Salt.
		Ore.	Metal.	Ore.	Pig					
1895....	707	(*b*)	105	(*b*)	2,569	195,422	8,145	....	(*b*)	12,758
1896....	443	(*b*)	206	(*b*)	10,120	222,724	6,821	....	(*b*)	13,720
1897 ...	396	3,847	135	37,095	15,606	229,643	5,344	....	(*b*)	13,919
1898....	458	3,760	156	57,935	15,263	270,752	5,320	3,670	4.0	14,496
1899....	200	3,980	180	67,030	13,730	303,000	5,270	....	3.3	15,030
1900....	100	3,008	141	133,454	38,960	394,516	7,939	1,700	6.7	15,791
1901....	505	3,696	199	122,569	39,296	445,007	6,346	4,570	9.3	16,865
1902....	270	3,657	166	133,348	43,992	424,753	5,760	5,170	7.2	17,348
1903....	147	1,073	191	114,059	39,833	467,962	4,538	6,589	8.1	18,459
1904....	279	640	115	127,297	47,678	483,617	1,114	10,421	8.1	18,021
1905....	186	670	39	122,540	43,074	540,237	4,129	19,045	10.0	(*b*)
1906....	320	765	25	136,513	45,660	594,172	7,651	11,347	5.1	(*b*)

(*a*) From *Oestr. Zeit. f. B.- u. H.* (*b*) Not reported.

BELGIUM.

The mining and metallurgical production in Belgium, and the imports and exports, according to the latest official statistics are as follows:

MINERAL, METALLURGICAL AND QUARRY PRODUCTION OF BELGIUM. (*a*)
(In metric tons except where otherwise noted.)

Year.	Barytes.	Chalk, Marl. *Cubic Meters.*	Clay.	Coal.		Coke.	Flint. *Cubic Meters.*		Iron Ore.
				Bituminous.	Briquets.		For Earthen ware.	For Ballast. (*c*)	
1896....	25,000	191,100	83,020	21,252,370	1,213,760	2,004,430	23,450	244,050	307,031
1897....	23,000	204,600	270,715	21,492,446	1,245,114	2,207,840	23,050	235,495	240,774
1898....	21,700	287,805	287,805	22,088,335	1,351,884	2,161,162	22,150	360,960	217,370
1899..	25,900	351,800	291,125	22,072,068	1,276,050	2,304,607	25,185	258,835	201,445
1900....	38,800	377,550	313,205	23,462,817	1,395,910	2,434,678	25,700	263,850	247,890
1901....	22,800	449,000	298,340	22,213,410	1,587,800	1,847,780	17,700	7,860	218,780
1902....	33,000	390,700	299,820	22,877,470	1,616,520	2,048,070	17,430	7,705	166,480
1903....	21,000	501,920	292,855	23,796,680	1,686,415	2,203,020	16,250	8,935	184,400
1904....	60,000	450,400	347,135	22,761,430	1,735,480	2,211,820	18,070	12,500	206,730
1905....	26,000	372,000	274,550	21,775,280	1,711,920	2,238,920	12,8[illegible]0	26,895	176,940
1906....	22,365	568,170	430,860	23,569,860	1,887,090	2,414,490	15,100	14,360	232,570

Year.	Iron, Crude.					Iron, Manufactures of.			
	Forge Pig.	Foundry Pig.	Bessemer Pig.	Basic Pig.	Total Pig.	Merchant Bars.	Sheet and Plate.	Wrought	Other Mfres.
1896........	362,451	84,275	193,518	307,779	959,414	81,394	112,597	851	298,163
1897........	426,332	78,410	183,701	333,958	1,035,037	108,608	100,252	872	263,644
1898........	308,875	93,645	173,085	397,891	979,755	123,993	91,686	993	267,521
1899........	317,029	84,165	169,664	453,718	1,024,576	93,601	97,604	662	283,331
1900........	305,344	88,335	176,557	447,271	1,018,561	61,458	73,572	1,411	284,591
1901........	178,250	86,170	166,820	332,940	764,180	249,380	65,760	550	64,900
1902........	104,540	254,710	199,170	510,630	1,069,050	260,290	62,740	450	58,150
1903........	91,600	256,890	229,160	638,430	1,216,080	274,520	56,550	390	60,920
1904........	99,350	224,410	217,390	742,040	1,287,597	246,240	41,000	370	67,580
1905........	98,170	206,390	220,210	784,850	1,311,120	270,840	39,250	40	67,490
1906........	96,090	218,225	177,900	870,860	1,375,775	265,010	37,540	20	35,680

Year.	Steel.					Lead.		Manganese Ore.
	Ingots, Blooms and Billets.	Rails.	Tires.	Wrought.	Plates.	Ore.	Pig.	
1896......................	598,947	147,183	10,497	6,702	64,653	70	17,222	23,265
1897......................	616,541	136,911	10,870	23,104	64,366	108	17,023	28,372
1898......	653,523	117,751	10,953	17,902	87,219	133	19,330	16,440
1899......................	731,249	123,119	11,212	32,180	68,051	137	15,727	12,120
1900......................	655,199	134,428	11,934	25,985	55,307	230	16,365	10,820
1901...	529,840	132,260	12,380	3,310	83,810	220	18,760	8,510
1902......................	786,980	(*e*)268,220	12,790	2,910	94,360	164	73,357	14,440
1903......................	988,160	(*e*)351,540	17,810	2,920	118,200	90	68,700	6,100
1904......................	1,065,870	(*e*)266,900	23,540	4,300	149,270	91	23,470	485
1905......................	1,227,110	241,640	25,810	6,080	179,470	126	22,885	*Nil.*
1906......................	1,440,860	274,920	32,070	6,835	223,580	121	23,765	120

Year.	Mineral Paints. Ochers. *Cubic Meters.*	Phosphate of lime. *Cubic Meters.*	Pyrites.	Sand. *Cubic Meters*	Slate. *Pieces.*	Silver. *Kg*	Stone. *Cubic Meters.* Dolomite.	Flagstones. *Sq. Meters.*	Freestone.
1896..	700	297,470	2,560	418,720	35,980,000	28,509	21,500	131,400	152,420
1897..	350	350,056	1,828	559,141	41,422,000	30,073	52,720	107,572	181,746
1898..	290	156,920	147	287,805	42,311,000	116,035	37,100	170,672	215,417
1899..	300	190,090	283	627,770	44,167,000	134,854	56,400	144,330	139,294
1900..	300	215,670	400	653,780	43,941,000	146,548	45,000	153,217	157,294
1901..	(*d*)2,100	(*d*) 222,520	560	626,020	39,030,000	169,450	31,500	106,470	167,310
1902..	(*d*) 200	(*d*) 135,850	710	722,775	37,120,000	212,249	39,140	101,945	185,319
1903..	(*d*) 200	(*d*) 184,120	720	724,495	38,953,000	232,740	43,600	117,165	245,184
1904..	(*d*) 450	(*d*) 202,480	1,075	807,715	41,240,000	252,920	48,600	71,630	216,717
1905..	(*d*) 300	(*d*) 193,305	976	775,385	41,435,000	201,935	78,860	83,455	325,000
1906..	(*d*) 250	(*d*) 152,140	908	926,355	43,801,000	173,535	78,670	81,805	227,748

Year.	Stone. *Cubic Meters.* (*Continued*). Limestone.	Limestone for Flux.	Marble.	Paving Stones. *Pieces.*	Whetstones and Hones. *Pieces.*	Zinc. Ore. (Blende)	Ore. (Calamine)	Spelter	Sheets.
1896.........	2,646,305	164,900	16,315	102,295,950	45,850	7,070	4,560	113,361	36,238
1897.........	3,010,877	225,300	17,797	95,542,700	43,150	6,804	4,150	116,067	37,011
1898.........	2,968,997	212,685	16,610	108,025,000	89,150	7,350	4,125	119,671	35,587
1899.........	3,238,875	195,505	17,740	114,103,900	82,100	5,736	3,730	122,843	34,289
1900.........	3,228,205	229,250	15,990	107,294,600	105,000	5,715	3,000	119,317	38,825
1901.........	3,751,880	193,370	15,390	110,920,000	160,150	4,445	2,200	127,170	37,380
1902.........	1,626,670	226,220	15,490	110,103,000	122,300	3,568	284	124,780	37,070
1903.........	1,580,330	210,250	16,735	111,318,000	134,620	3,565	65	131,740	42,280
1904.........	1,645,655	213,320	17,740	117,412,000	135,700	3,698	4	137,323	41,490
1905.........	1,493,745	250,500	17,254	115,440,000	154,820	3,929	*Nil.*	142,555	45,320
1906.........	1,521,660	316,870	15,335	110,143,000	192,460	3,858	*Nil.*	148,035	44,525

(*a*) From *Statistique des Industries Extractives et Metallurgiques et des Appareils à vapeur en Belgique.* (*c*) Includes some gravel. (*d*) Metric tons (*e*) Includes beams.

MINERAL IMPORTS OF BELGIUM. (*a*)
(In metric tons or dollars; 5 f.=$1.)

Year.	Ashes.	Cement.	Clay Products. Terra Cotta.	Common Pottery.	Coal.	Coal Briquets.	Coke.
1896..............	6,747	30,565	85,486	2,065	1,693,376	1,561	260,273
1897..............	10,870	17,681	86,493	2,115	2,017,344	632	269,606
1898..............	8,199	34,039	92,149	2,007	2,202,517	1,756	180,590
1899..............	15,818	18,649	99,156	2,856	2,844,274	10,722	296,508
1900..............	15,428	12,773	90,852	4,281	3,288,513	21,814	289,673
1901..............	14,802	13,558	81,359	2,223	2,930,874	17,160	154,247
1902..............	16,708	13,269	91,795	2,132	3,232,510	33,235	230,612
1903..............	37,178	19,698	107,457	2,095	3,554,807	43,835	308,877
1904..............	56,052	37,593	111,394	6,401	3,701,240	45,600	338,127
1905..............	39,210	34,610	105,619	5,605	4,230,313	72,643	356,136
1906..............	39,102	29,003	93,537	3,203	5,358,789	147,302	352,316

Year.	Copper and Nickel. Crude.	Hammered, Drawn or Rolled.	Wrought	Diamonds Crude and Uncut.	Fertilizers (All).	Glass and Glassware Common (Bottles, Broken Glass, Etc)	Gold (Including Platinum). Ore. *Kg.*	Unwr'ght. *Kg.*	Specie.	Jewelry.
1896...	15,506	1,109	$188,931	....	(*c*) 25,946	6,980	93	4,923	$ 599,540	$ 757,507
1897...	14,821	1,418	193,242	....	(*c*) 5,162	4,699	....	3,824	1,726,700	701,535
1898...	14,947	1,821	205,705	....	(*c*) 10,657	4,247	8,390	1,282	372,000	840,593
1899...	8,327	2,174	226,853	....	(*c*) 15,072	3,757	51	1,136	744,000	965,170
1900...	13,768	2,087	231,800	$8,051,200	163,229	5,671	1,250	1,728	459,420	921,600
1901...	11,381	1,780	272,600	8,463,200	149,984	5,446	....	......	....	1,332,800
1902...	14,197	1,998	251,400	8,537,600	158,924	6,129	.....	.....	....	1,221,800
1903...	13,602	2,035	313,400	15,557,800	188,389	5,209	...	....	....	1,190,000
1904...	13,422	2,267	490,400	16,059,400	157,402	5,267	8,516	2,385	256,680	1,009,843
1905...	12,379	2,421	439,017	18,674,270	174,435	4,287	49	1,483	571,600	1,502,107
1906...	22,007	2,687	419,265	18,683,536	211,067	10,738	11,749	6,934	1,288,360	1,065,944

Year	Iron. Ore.	Iron. Pig and Scrap.	Iron. Tin Plate.	Steel. Ingots, Blooms, and Billets.	Lead. Pig.	Lead. Manufactures.
1896	2,069,676	378,191	3,203	28,435	35,221	$ 17,231
1897	2,544,377	354,178	3,875	25,370	43,840	91,580
1898	2,252,553	370,117	3,848	25,142	54,867	50,728
1899	2,621,152	423,968	3,900	11,666	60,649	191,508
1900	2,528,615	371,726	5,036	19,705	58,141	143,000
1901	1,768,441	222,230	(*d*)4,705	68,228	54,720	123,200
1902	2,550,347	348,337	(*d*)6,608	103,286	71,085	164,800
1903	3,054,808	387,884	(*d*)9,776	144,370	63,386	126,600
1904	3,359,430	388,732	(*d*)7,899	182,336	63,813	234,400
1905	3,382,832	558,414	(*d*)7,426	167,513	61,668	397,400
1906	3,549,391	778,260	(*d*)9,605	115,038	54,690	531,317

Year.	Lime.	Petroleum. Crude.	Petroleum. Refined.	Resins and Bitumens, Not Specified.	Salt. Crude.	Salt. Refined.
1896	11,522	95	158,979	216,278	92,408	38,785
1897	13,184	988	149,501	237,570	96,805	39,193
1898	12,674	382	161,281	269,914	92,300	50,136
1899	12,811	2,479	166,404	264,718	81,324	50,647
1900	11,448	1,751	158,064	253,788	97,812	59,375
1901	11,288	305	160,327	244,866	93,043	48,974
1902	13,820	247	183,592	305,597	102,110	50,704
1903	13,525	237	210,905	285,305	108,886	52,272
1904	16,896	1,893	192,805	305,136	110,516	54,615
1905	17,220	143	194,584	324,298	111,317	59,575
1906	12,035	*Nil.*	196,458	332,297	113,035	60,928

Year	Silver. Ore.	Silver. Bullion. *Kg.*	Silver. Specie.	Silver. Jewelry.	Sodium Salts. Carbonate.	Sodium Salts. Nitrate.	Sodium Salts. Sulphate and Sulphite.
1896	1,477	8,980	$ 6,461,840	$415,967	(*e*)	194,202	(*e*)
1897	2,533	467,851	2,083,040	460,244	(*e*)	181,676	(*e*)
1898	461	299,369	7,655,200	449,244	(*e*)	152,164	(*e*)
1899	2,523	105,723	14,272,080	520,070	(*e*)	249,756	(*e*)
1900	922	11,366	7,324,560	513,800	20,603	153,318	49,020
1901	....	....	....	550,000	19,465	167,489	47,558
1902	....	....	....	546,200	13,584	135,937	60,721
1903	64	....	....	597,800	19,459	149,942	54,926
1904	310	....	....	635,800	28,006	157,005	44,122
1905	499	12,876	4,916,480	388,971	26,525	202,241	50.667
1906	313	32,396	16,509,760	444,668	19,438	175,180	64,706

Year.	Stone. Roofing Slate. *1000 Pieces.*	Stone. Building Stone, Including Marble and Alabaster.	Stone. Cut, Polished, Etc.	Stone. Paving.	All Other Kinds.	Sulphur.	Tin. Block	Zinc. Spelter.	Zinc. Manufactures.
1896	38,209	40,511	$ 61,769	6,163	81,360	14,399	4,617	20,182	$11,230
1897	38,754	47,929	111,313	13,197	182,950	13,261	1,609	16,320	10,661
1898	38,216	45,544	63,665	8,926	239,281	13,322	1,208	17,441	11,575
1899	35,888	49,498	83,372	7,835	216,231	8,449	1,113	11,058	11,436
1900	34,331	60,057	145,800	3,761	135,245	17,516	1,653	11,478	12,000
1901	35,516	65,509	120,800	5,367	81,006	14,775	1,841	13,896	11,200
1902	36,950	69,745	109,000	7,124	72,384	12,367	1,416	17,830	14,600
1903	34,068	83,165	153,800	5,371	103,239	21,637	2,677	20,586	15,000
1904	34,139	86,013	143,800	4,545	115,275	24,788	3,416	17,424	13,800
1905	33,501	100,856	197,196	15,759	116,119	19,617	3,270	15,692	14,599
1906	31,015	96,866	228,655	10,060	133,379	18,627	2,975	15,553	10,205

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers, Brussels.* (*b*) Pieces. (*c*) Guano only is reported. (*d*) Includes wrought tin-plate. (*e*) Included under nitrate.

MINERAL EXPORTS OF BELGIUM. (a)
(In metric tons or dollars; 5 fr.=$1.)

Year	Ashes.	Cement.	Clay Products.		Coal.	Coal Briquets.	Coke.	Copper and Nickel.		
			Terra-cotta.	Common Pottery.				Crude.	Hammered Drawn or Rolled.	Wrought.
1896	1,084	277,615	302,526	2,628	4,649,799	459,974	863,067	11,700	2,073	$168,524
1897	2,675	322,024	294,815	3,197	4,448,544	615,074	909,486	9,994	1,996	198,665
1898	615	419,132	247,970	3,186	4,579,955	666,265	878,435	8,511	1,770	161,726
1899	400	445,602	328,733	3,216	4,568,938	525,625	1,008,470	4,665	2,111	261,060
1900	2,148	408,284	289,759	5,915	5,260,993	604,864	1,073,315	8,411	2,097	276,600
1901	3,401	492,882	220,326	6,661	4,820,300	714,455	829,421	6,309	1,988	198,200
1902	4,438	542,547	256,555	3,815	5,078,278	671,700	824,256	7,320	1,656	235,400
1903	3,202	599,092	342,149	5,826	4,923,368	623,691	841,142	7,638	1,702	256,400
1904	864	588,295	329,616	3,450	5,067,037	539,364	879,883	7,255	2,243	182,200
1905	3,713	679,426	249,707	3,047	4,704,063	480,247	977,095	6,805	2,056	315,648
1906	4,477	813,329	265,282	3,952	4,972,340	459,753	856,475	14,661	2,032	287,400

Year.	Diamonds. Crude and Uncut.	Fertilizers. (All Kinds.)	Glass and Glassware.			Gold (Including Platinum).		
			Common (Bottles, Broken Glass, etc.)	Plate.	All Other Kinds.	Unwrought, Kg.	Specie.	Jewelry.
1896.......	....	(c)14,633	3,647	$3,468,668	178,611	3,713	$2,666,620	$ 75,570
1897.......	....	(c)14,044	3,546	3,761,219	174,232	2,547	605,120	126,385
1898.......	....	(c)21,626	2,747	3,007,865	25,226	1,231	578,120	118,897
1899.......	....	(c)18,213	3,850	4,942,298	196,842	504	998,200	118,916
1900.......	$ 8,601,000	258,366	3,993	4,413,000	159,557	549	613,180	161,200
1901.......	8,814,200	299,709	5,486	5,030,400	154,610	....	....	225,800
1902.......	8,855,400	359,607	6,338	4,799,800	198,647	....	....	193,800
1903.......	16,708,200	497,355	5,156	4,660,800	197,246	....	....	159,400
1904.......	17,028,400	555,956	6,781	4,707,200	146,637	671	726,020	137,400
1905.......	18,674,270	511,395	6,183	5,410,600	182,314	608	1,860,000	210,600
1906.......	19,993,394	639,326	7,697	5,210,640	249.552	13,040	8,361,940	133,935

Year.	Iron.			Steel. Ingots, Blooms and Billets.	Lead.		Lime.
	Ore.	Pig and Scrap. (d)	Tin Plate.		Pig.	Manufactures.	
1896........................	389,235	63,906	3,952	1,145	31,366	$36,821	477,213
1897........................	410,817	60,678	1,191	1,201	35,988	33,286	520,588
1898........................	384,047	40,522	973	1,018	40,302	16,450	546,199
1899........................	318,415	76,047	1,436	1,259	41,618	37,022	537,357
1900........................	420,180	79,172	940	975	46,566	9,000	617,666
1901........................	327,499	70,027	(e) 642	290	47,971	6,600	579,123
1902........................	368,560	96,302	(e)1,023	1,463	58,495	9,400	623,617
1903........................	400,972	101,549	(e)3,577	3,047	7,765	8,200	659,125
1904........................	441,059	85,489	(e)2,378	5,250	59,334	13,400	706,351
1905........................	443,511	80,896	(e)1,322	19,075	53,378	53,669	720,123
1906........................	436,465	99,983	(e)3,478	31,860	54,207	67,939	699,300

Year.	Petroleum.		Resins and Bitumens, not specified.	Salt.		Silver.			
	Crude.	Refined.		Crude.	Refined.	Ore.	Bullion, Kg.	Specie.	Jewelry, etc.
1896............	2	29,321	86,906	1,434	129	19	40,118	$ 667,840	$137,137
1897............	1	18,088	92,591	493	231	423	57,933	20,851,320	189,959
1898............	782	19,556	107,806	298	386	......	107,385	13,083,640	127,194
1899............	2,146	25,970	112,392	506	885	......	54,358	13,483,160	214,175
1900............	1,759	21,812	97,970	2,345	799	......	38,331	1,304,840	103,200
1901............	*Nil.*	22,091	86,908	1,611	2,454	......	....	....	85,000
1902............	*Nil.*	23,344	101,950	1,378	1 077	......	....	....	88,000
1903............	*Nil.*	29,155	93,533	928	618	......	....	....	94,400
1904............	*Nil.*	30,209	97,462	1,899	1,955	......	....	6,870,000	108,800
1905............	*Nil.*	36,491	115,212	305	2,186	1,789	44 682	723,720	99,971
1906............	*Nil.*	31,908	123,275	17	1,724	2	46,732	4,086,680	27,881

Year.	Sodium Salts.			Stone.		
	Carbonate.	Nitrate.	Sulphate and Sulphite.	Roofing Slate, 1000 *Pieces.*	Building Stone Including Marble and Alabaster.	Cut, Polished, etc.
1896	(*f*)	42,857	(*f*)	15,435	161,298	$ 922,147
1897	(*f*)	9,054	(*f*)	17,304	187,180	934,286
1898	(*f*)	106,252	(*f*)	16,948	178,249	861,015
1899	(*f*)	109,253	(*f*)	15,316	164,952	948,090
1900	25,569	39,346	10,577	12,836	171,126	872,400
1901	22,520	44,303	15,833	12,947	168,824	735,800
1902	12,831	33,319	14,316	13,715	163,210	848,800
1903	9,454	36,057	17,434	11,984	160,874	887,800
1904	15,049	36,970	18,715	11,362	160,421	822,800
1905	16,571	58,815	15,367	11,934	147,934	867,905
1906	20,619	60,548	7,707	11,596	147,062	1,032,543

Year.	Stone. (*Continued.*)		Sulphur.	Tin. Block.	Zinc.	
	Paving.	All Other Kinds.			Spelter.	Manufactures.
1896	154,737	796,231	5,335	1,055	100,369	$ 56,349
1897	153,504	773,531	6,041	347	100,228	90,749
1898	159,455	917,654	6,355	508	108,507	102,583
1899	150,993	834,528	6,769	659	101,244	109,762
1900	178,057	1,022,781	7,363	495	969,233	98,400
1901	148,176	729,196	6,722	299	106,656	98,600
1902	145,019	685,567	7,349	234	118,118	95,800
1903	128,715	696,338	10,324	838	119,988	156,200
1904	119,557	964,255	9,020	815	116,289	165,800
1905	118,185	916,504	5,925	201	125,423	169,910
1906	114,444	1,070,918	5,841	294	132,774	167,646

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays Etrangers.* (*c*) Guano only is reported. (*d*) Includes iron and steel filings. (*e*) Includes wrought tin plate. (*f*) Included under nitrate.

CANADA.

The statistics of mineral production in the Dominion of Canada as reported by the Geological Survey are summarized in the following tables.

The statement of imports and exports for 1907 is for the nine months ending March 31 in consequence of a change in the law whereby the fiscal year was changed from June 30:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (a)
(In metric tons or dollars.)

Year.	Arsenic.	Asbestos and Asbestic.	Barytes.	Cement—*Barrels.*		Chromite.	Clay Products.		
				Natural Rock.	Portland.		Fire Clay.	Pottery.	Terra-Cotta.
1896......	*Nil.*	11,113	132	70,705	78,385	2,124	764	$163,427	$83,855
1897	*Nil.*	27,617	518	85,450	119,763	2,392	1,921	129,629	155,595
1898......	*Nil.*	21,577	971	87,125	163,084	1,833	608	135,000	167,902
1899......	52	22,938	653	131,387	255,366	1,796	543	185,000	220,258
1900..	275	27,797	1,213	125,428	292,124	2,335	1,129	200,000	259,450
1901......	630	36,477	592	133,328	317,066	1,274	3,609	200,000	278,671
1902......	726	36,657	994	127,931	594,594	900	2,486	200,000	276,241
1903......	725	37,902	1,055	92,252	627,741	3,509	2,394	200,000	405,796
1904	(*d*) 66	44,131	1,253	51,555	771,650	5,511	(*b*)	200,000	400,000
1905	*Nil.*	61,928	3,049	14,184	1,346,547	7,781	1,437	120,000	64,892
1906(*g*)...	*Nil.*	72,025	3,628	8,610	2,139,164	7,936	(*b*)	(*b*)	(*b*)
1907(*g*)...	317	82,117	1,829	5,775	2,368,593	6,527	(*b*)	(*b*)	(*b*)

Year.	Clay Products (*Continued*)	Coal.	Coke.	Copper. In Ore, etc.	Corundum.	Feldspar.	Gold. (*e*)	Graphite.	Grindstones.
	Tiles and Sewer Pipe.								
1896...	$378,875	3,398,091	45,004	4,260	(*b*)	882	$2,754,774	126	3,368
1897...	389,250	3,434,756	55,042	6,032	(*b*)	1,270	6,027,016	395	4,147
1898...	406,717	3,784,532	79,453	8,048	(*b*)	2,268	13,775,420	1,107	4,476
1899...	386,546	4,467,021	91,444	6,838	(*b*)	2,721	21,261,584	1,188	4,091
1900...	456,525	5,087,060	142,521	8,588	3	288	27,908,153	1,743	5,024
1901...	498,115	5,648,208	331,537	17,155	403	4,852	24,128,503	2,004	4,155
1902...	551,965	6,524,180	455,353	17,598	697	6,871	21,336,667	993	5,835
1903...	592,970	6,933,107	509,115	19,357	880	12,633	18,843,590	660	5,023
1904...	653,894	6,812,834	493,107	19,497	834	10,057	16,400,000	410	4,091
1905...	642,000	7,961,397	622,154	21,596	1,492	10,617	14,486,833	491	4,693
1906(*g*)	(*f*)446,790	9,033,973	(*b*)	25,863	2,063	14,397	12,023,932	405	5,029
1907(*g*).	1,211,000	9,533,442	(*b*)	26,025	1,716	11,414	8,264,765	525	4,881

Year.	Gypsum.	Iron Ore.	Iron, Pig. All kinds.	Iron and Steel, Rolled.	Lead (In ore, etc.)	Lime.	Limestone for Flux.	Manganese Ore.	Mica.
1896.......	187,778	83,359	61,012	76,244	10,975	$650,000	33,978	(*d*)112	$60,000
1897.......	217,340	45,989	52,612	78,253	17,695	650,000	28,365	(*d*) 14	76,000
1898.......	198,864	52,917	69,853	101,748	14,469	650,000	30,759	45	118,375
1899.......	221,821	67,678	93,367	112,412	9,914	800,000	47,006	1,434	163,000
1900.......	228,656	110,654	87,594	102,301	28,648	800,000	48,040	27	166,000
1901.......	266,476	284,477	248,859	113,799	23,537	830,000	153,645	(*d*) 399	160,000
1902.......	301,165	366,431	324,617	164,069	10,411	892,000	266,290	(*d*) 156	135,904
1903.......	285,242	239,715	270,182	131,588	8,226	860,000	251,649	83	177,857
1904.......	309,133	317,387	275,367	(*b*)	17,241	(*h*)	182,023	(*d*)112	152,919
1905.......	395,341	263,113	475,491	(*b*)	25,391	750,000	309,907	(*d*) 20	168,170
1906(*g*) ..	378,904	269,842	550,628	(*b*)	24,580	(*h*)	331,976	(*d*) 84	(*d*) 581,043
1907(*g*)....	431,286	......	590,444	(*b*)	21,570	.....	326,069	......	333,022

Year.	Mineral Paints. (Ochers)	NaturalGas.	Nickel. (In ore, etc.)	Petroleum, Crude. *Barrels.* (e)	Phosphate (Apatite).	Pyrites.	Salt.	Sand and Gravel Exports.
1896	2,142	$276,301	1,541	726,822	517	30,580	39,872	203,865
1897	3,542	325,873	1,813	709,857	824	35,291	46,574	138,737
1898	2,019	322,123	2,502	758,391	665	29,223	51,828	150,520
1899	3,555	387,271	2,605	808,570	2,721	25,112	53,820	219,902
1900	1,783	417,094	3,211	710,498	1,283	36,308	56,284	179,185
1901	2,025	339,476	4,167	623,392	937	31,982	53,901	178,953
1902	4,494	195,992	4,849	530,624	776	32,304	58,462	144,932
1903	5,683	202,210	5,671	486,637	1,205	30,822	56,644	322,703
1904	3,562	247,370	4,786	552,575	832	29,980	62,411	362,701
1905	4,632	314,249	8,565	634,095	1,180	29,713	41,159	332,879
1906(g)	6,201	528,868	9,745	569,753	(b)	35,927	69,283	232,691
1907 (g)	7,115	748,541	9,610	788,872	680	35,494	65,936	270,372

Year.	Sand (Molding).	Silver—*Kg.* (In ore, etc.)	Slate.	Soapstone and Talc.	Stone.			
					Building Bricks.	Building Stones.	Flagstones.	Granite.
1896	5,205	99,699	$53,370	372	$1,600,000	$1,000,000	$6,710	$106,709
1897	4,975	172,891	42,800	142	1,600,000	1,000,000	7,190	61,934
1898	9,589	138,486	40,791	367	1,900,000	1,300,000	4,250	81,073
1899	12,448	106,116	33,406	408	2,195,000	1,500,000	7,600	90,542
1900	5,606	138,980	12,100	1,288	2,275,000	1,520,000	5,250	80,000
1901	13,337	172,292	9,980	235	2,400,000	1,650,000	4,575	155,000
1902	12,110	133,478	19,200	625	2,593,000	1,900,000	7,760	210,000
1903	3,318	99,489	22,040	898	2,832,000	1,975,000	6,688	200,000
1904	3,105	115,666	23,247	762	(h)	5,667,000	6,720	100,000
1905	(b)	185,839	21,568	454	3,933,925	6,095,000	7,650	209,955
1906(g)	(b)	266,521	24,446	1,119	(h)	7,200,000	(b)	(b)
1907(g)	(b)	390,359	20,056	1,391	(h)	7,500,000	(h)	(h)

(a) From *Reports Compiled by the Geological Survey of Canada.* (b) Not reported. (c) Gold values are calculated at the rate of $20.67 per oz. (d) Export. (e) One barrel contains 35 imp. gal. (f) Sewer pipe only reported. (g) From preliminary unrevised reports. (h) Included under building stone.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (a.)

(In metric tons or dollars.)

Year. (b)	Abrasives.				Aluminum.		Antimony. (c)	Arsenic.	Asbestos. (d)
	Buhrstones Number.	Emery, (Wheels and Bulk)	Grindstones.	Pumicestone.	Manufactures.	Ingots, Sheets, Etc.			
1897	1,499	$11,231	$25,547	$2,903	$5,717	....	61	68	$19,032
1898	889	15,478	22,217	3,829	7,102	....	71	132	26,389
1899	1,116	22,343	27,476	5,973	9,275	....	131	264	32,607
1900	1,250	44,882	34,382	5,604	12,543	....	90	105	43,455
1901	3,641	39,116	39,068	5,516	16,202	....	159	72	50,829
1902	1,854	23,946	40,838	7,254	30,496	....	229	48	52,464
1903	35	40,235	53,388	6,152	14,201	$ 13,930	393	135	75,465
1904	2	50,899	46,039	6,537	16,065	101,427	190	188	83,827
1905	1,931	55,230	49,247	8,447	28,418	154,569	85	122	116,836
1906	1,746	63,876	59,627	9,053	23,565	168,405	183	202	138,000
1907(o)	31	41,080	40,780	5,745	20,656	218,399	146	158	127,509

Year.	Asphalt.	Brass. (e)	Cement.	Chalk and Whiting.	Clay Products.		
					Bricks and Tiles.	Clays.	Earthenware and China.
1897	342	$ 457,342	$ 260,842	$29,973	$ 44,622	$ 59,386	$ 595,822
1898	6,006	560,014	365,624	35,099	36,263	72,795	675,874
1899	8,196	747,557	477,617	44,771	55,204	88,517	916,727
1900	2,825	853,599	513,770	46,787	58,454	122,965	959,526
1901	2,849	985,776	666,350	72,507	76,760	141,251	1,114,677
1902	3,426	1,017,294	863,646	53,473	89,332	140,521	1,275,093
1903	3,037	1,197,012	890,745	56,364	157,783	176,416	1,406,610
1904	7,093	1,257,117	1,014,713	65,840	259,421	144,706	1,611,356
1905	5,096	1,375,907	1,263,828	77,387	369,561	197,609	1,636,214
1906	7,178	1,786,596	1,003,022	77,750	460,447	220,504	1,674,817
1907(o)	11,929	942,741	540,006	50,899	421,501	178,240	1,422,880

Year.	Coal.		Coal Tar. *Barrels.*	Coke.	Copper.		Copper. Sulphate	Explo-sives.	Flint and Stones.
	Anthracite. (*f*)	Bituminous. (*f*)			Ingots, Pig and Scrap.	Manu-factures.			
1897	1,321,767	1,604,517	23,661	75,580	22	$ 264,587	516	$131,562	475
1898	1,324,856	1,735,576	26,702	122,499	476	786,529	738	141,731	389
1899	1,583,132	2,220,250	39,296	128,145	751	551,586	726	212,968	243
1900	1,500,542	2,512,334	50,484	170,405	519	1,090,280	752	247,511	280
1901	1,753,488	2,658,257	54,928	280,069	432	951,045	673	306,067	222
1902	1,498,773	3,208,005	55,376	242,298	801	1,281,522	711	423,982	186
1903	1,320,239	3,684,502	29,325	232,848	924	1,291,635	1,010	347,020	403
1904	2,064,444	4,230,436	55,172	200,590	960	1,094,183	795	418,916	554
1905	2,361,952	4,377,667	77,856	337,035	882	1,666,955	934	369,311	844
1906	1,996,183	5,003,029	66,540	435,561	1,191	2,660,882	844	647,272	1,410
1907(*o*)	1,260,723	4,022,843	43,017	363,286	1,186	2,424,201	897	519,368	2,220

Year.	Fuller's Earth.	Glass.	Gold and Silver.		Gravel and Sand.	Graphite.	
			Coin and Bullion. (*g*)	Manu-factures.		Crude.	Manu-factures. (*h*)
1897	$1,552	$1,139,764	$ 4,676,094	$296,143	19,330	$1,406	$38,537
1898	3,330	1,024,706	4,390,844	297,242	29,164	1,862	52,291
1899	3,418	1,343,058	4,705,134	342,320	27,477	4,979	57,824
1900	2,661	1,658,694	8,297,438	339,145	32,399	4,437	60,518
1901	3,147	1,584,922	3,537,294	367,857	35,744	2,357	75,536
1902	3,909	1,932,539	6,311,405	352,224	42,995	3,649	64,123
1903	4,169	2,084,451	8,976,797	434,273	83,047	2,870	69,676
1904	5,554	1,983,781	7,874,313	444,154	100,394	1,802	67,563
1905	4,967	1,948,969	10,308,435	502,305	77,402	2,499	75,288
1906	4,644	2,671,398	7,078,603	579,378	105,666	2,791	86,028
1907(*o*)	4,483	2,108,571	7,029,047	431,102	155,732	3,176	57,430

Year.	Gypsum.		Iron and Steel.				Kainite
	Crude and Ground.	Plaster of Paris.	Pig and Scrap.	Slabs, Blooms, Bars, Etc.	Alloys of Iron.	Manufactures.	
1897	482	440	33,442	2,566	387	$10,133,379	206
1898	1,057	150	81,577	7,391	1,287	15,458,355	49
1899	310	225	69,819	5,640	1,053	18,874,092	30
1900	72	385	94,489	11,576	1,043	27,259,134	143
1901	289	228	59,033	10,659	1,372	25,686,154	88
1902	516	215	71,882	18,208	5,910	31,257,412	85
1903	1,007	286	129,641	17,896	5,762	39,350,068	259
1904	626	291	86,087	9,088	2,700	31,014,946	339
1905	2,972	3,595	90,698	14,420	11,738	29,847,298	306
1906	5,743	6,579	112,937	29,520	13,626	35,698,630	306
1907(*o*)	8,334	9,730	137,654	17,369	17,785	34,963,816	511

Year.	Lead					Lime.		Mineral Paints. (Ochers)
	Pig and Scrap.	Bars and Sheets.	Litharge.	Manu-factures.	Pigments and Zinc White.	Burned. *Barrels.*	Chloride of.	
1897	2,962	477	546	$60,735	4,678	16,108	1,361	682
1898	4,012	1,008	519	63,179	5,754	12,850	1,765	965
1899	5,202	2,032	432	91,497	6,583	15,720	1,857	1,110
1900	2,829	703	415	104,736	6,661	12,865	1,967	1,122
1901	(*i*)3,871	739	505	107,260	4,647	19,657	1,605	1,031
1902	(*i*)5,548	844	590	120,020	7,071	24,602	1,806	1,148
1903	(*i*)4,471	523	632	134,151	8,715	31,108	2,104	1,459
1904	4,292	......	......	133,639	7,679	54,359	2,080	1,256
1905	2,589	800	811	147,177	9,695	98,676	2,507	1,417
1906	3,751	730	461	163,666	6,947	134,334	2,645	809
1907 (*o*)	3,811	622	513	82,693	2,215	88,919	2,302	570

Year.	Mineral Waters.	Nickel.	Petroleum Products—*Gallons.* Illuminating oil, Etc. Crude or Refined.	Paraffin wax and Candles.	Platinum.	Potassium Salts. Except Saltpeter.	Saltpeter.
1897	$ 22,142	$ 4,737	8,415,302	74	$9,031	265	456
1898	33,314	5,882	9,074,311	75	9,781	244	627
1899	38,046	9,446	10,394,208	70	9,671	472	930
1900	30,343	6,988	9,633,647	35	57,910	733	602
1901	40,802	12,029	11,082,822	74	20,263	476	581
1902	91,871	15,448	13,220,005	123	19,357	771	690
1903	108,130	26,177	18,799,312	307	21,251	1,060	916
1904	136,583	14,682	24,521,115	228	28,112	1,151	898
1905	161,790	19,076	13,229,855	98	61,719	945	1,048
1906	179,837	15,976	10,981,611	375	54,494	1,317	1,141
1907 (*o*)......	142,179	19,461	8,066,403	189	113,967	1,074	638

Year.	Precious Stones and Jewelry.	Quick-Silver.	Sal-Ammoniac	Salt.	Silex.	Sodium Salts except Chloride.	Stone, Building.
1897	$ 506,728	35	69	103,337	116	13,938	$ 38,714
1898	743,607	27	38	96,962	141	16,026	28,495
1899	923,837	47	53	88,397	179	20,742	48,040
1900	732,675	39	60	92,823	182	16,748	64,533
1901	1,279,617	64	76	103,402	162	18,631	46,078
1902	1,497,321	44	78	114,629	199	17,133	99,074
1903	1,977,359	75	114	112,188	159	18,887	87,866
1904	2,075,675	69	93	103,635	252	25,118	93,778
1905	2,315,889	47	143	97,723	405	26,219	102,817
1906............	2,489.364	68	209	99,788	338	30,401	189,261
1907 (*o*).........	2,143,718	44	130	73,156	542	25,068	137,365

Year.	Stone. (*Continued.*) Lithographic Stones.	Marble.	Manu-factures.	Slate.	Sulphur.	Tin and Tinware.	Zinc. Spelter.	Manu-factures.
1897.......	$ 6,360	$ 77,150	$ 34,026	$ 21,615	3,932	$1,274,108	542	$ 5,145
1898.......	7,791	95,894	41,240	24,907	17,248	1,550,851	1,595	10,503
1899.......	6,223	101,879	60,148	33,100	11,121	1,372,813	852	14,661
1900.......	6,294	94,017	57,039	53,707	9,584	2,418,455	1,304	1,475
1901.......	9,584	96,159	66,639	72,187	10,827	2,339,109	931	6,882
1902.......	12,272	130,424	72,397	72,601	11,180	2,293,958	1,582	6,683
1903.......	8,461	153,481	78,629	84,437	11,077	2,712,186	1,209	9,754
1904.......	17,981	181,511	102,829	86,057	8,786	2,389,557	1,540	14,092
1905.......	13,683	145,466	150,160	93,228	10,633	2,791,757	1,721	11,912
1906.......	6,772	190 044	121,302	113,151	19,512	3,105,876	3,383	12,921
1907 (*o*)....	8,698	176,450	104,848	95,670	11,725	2,473,572	2,761	12,556

EXPORTS OF DOMESTIC MINERAL PRODUCE FROM THE DOMINION OF CANADA. (*a*)

(In metric tons or dollars.)

Year (*b*).	Antimony Ore.	Asbestos.	Clay Products. Bricks. *Thousands*	Clay. Mfres. of	Cement.	Chromite.	Coal.	Coke.
1897...	....	9,954	906	$ 796	$1,332	(*k*)1,911	1,000,061	1,692
1898...	1,118	16,718	276	343	609	(*k*)1,527	981,963	3,275
1899...	....	13,176	93	339	2,789	(*k*)1,369	1,035,245	4,024
1900...	6	16,483	342	215	2,274	(*k*) 334	1,489,139	12,558
1901...	219	24,242	728	761	3,554	(*k*)2,049	1,713,737	60,129
1902...	13	30,011	669	414	1,359	(*k*) 672	1,649,278	52,873
1903...	128	27,823	2,083	109	9,735	658	1,796,689	39,616
1904...	87	31,444	971	36	5,467	2,103	1,494,106	61,750
1905...	340	37,320	670	2,755	5,430	3,702	1,465,809	116,387
1906...	388	40,367	706	....	8,913	1,640	1,651,203	50,004
1907 (*o*)	832	37,194	568	125	5,229	604	1,165,809	44,669

Year.	Coin and Bullion. (*m*)	Copper (*e*)	Explosives.	Fertilizer.	Glass and Glassware	Gold. Quartz, Dust, etc.	Graphite	Grindstones
1897...	$ 327,298	4,596	$ 76,578	$ 36,584	$ 7,208	$ 2,804,101	78	$15,760
1898...	1,045,923	6,319	74,305	46,864	7,494	3,387,953	348	18,785
1899..	1,101,245	3,843	115,065	51,224	11,788	3,272,702	662	18,619
1900...	1,670,068	6,274	155,764	51,410	11,016	14,148,543	1,742	22,196
1901...	1,978,489	11,954	240,535	37,706	13,574	24,445,156	1,246	38,304
1902...	1,669,420	13,789	248,434	61,831	11,587	19,668,015	783	21,878
1903...	619,963	13,445	254,605	110,474	14,065	16,437 528	530	14,169
1904...	2,465,557	20,279	212,124	177,193	21,452	18,715,539	269	12,676
1905...	1,844,811	17,431	184,531	229.212	16,163	15.208.380	201	27 985
1906 ..	9.928.828	20.082	205.856	236.114	10,558	12,991,916	180	15,793
1907(*o*)	13,187,467	11,845	150,519	198,443	8,670	7,226,954	3	33,929

Year.	Gypsum.		Iron and Steel.			Lead. (*p*)	Lime.	Manganese Ore.
	Crude.	Ground.	Iron Ore.	Pig and Scrap.	Manufactures.			
1897 ..	163,829	$ 18,710	(*n*) 3,056	....	....	13,636	$ 56,720	74
1898 ..	163,660	2,587	(*n*) 1,975	....	....	19,944	48,307	7
1899 ..	148,565	7,611	(*n*) 2,881	$ 90,505	$ 615,906	15,445	64,112	24
1900 ..	211,792	2,622	(*n*) 5,012	411,491	1,013,672	8,998	77,325	57
1901 ..	156,080	25,472	(*n*) 54,208	256,250	1,176,711	29,747	83,439	33
1902 ..	243,629	10,150	(*n*) 478,503	1,262,285	1,198,476	13,890	111,910	500
1903 ..	271,899	7,947	(*n*) 267,000	335,958	2,927,982	7,386	127,792	137
1904 ..	247,741	10,154	(*n*) 214,309	229,824	1,761,997	7,329	104,044	62
1905 ..	290,574	2,801	204,091	172,720	950,634	23,094	75,498	84
9106 ..	367.203	1.603	134.270	346,030	1,251,289	6,158	73,534	15
1907(*o*)	249,780	2,653	31,011	146,802	962,944	8,330	46,662	84

Year.	Mica.	Nickel in Ore, Matte, etc.	Petroleum, Crude and Refined.	Pyrites.	Salt. *Bushels.*	Silver—*Kg.* (In Ore, Matte, etc.)	Stone. *All Kinds.*	Tin. Mfres.
1897.......	217	3,415	1,831	14,219	4,702	127,440	23,700	$ 2,764
1898.......	231	6,697	9,530	18,752	5,559	211,012	85,671	5,578
1899.......	538	6,546	4,268	11,707	5,209	137,400	110,290	3,159
1900.	490	6,122	6,758	13,507	15,151	71,015	217,880	3,472
1901.......	444	4,327	19,942	22,146	56,461	125,110	182,342	14,481
1902.......	452	1,762	2,478	24,088	21,778	114,610	230,798	26,524
1903..	632	4,098	413	16,762	7,959	100,861	212,097	90,953
1904.......	393	6,456	1,208	15,582	42,662	99,472	114,192	76,796
1905.......	461	5,431	6,441	20,473	5,663	112,076	78,791	37,535
1906.......	603	10,866	1,741	18,398	23,168	203,323	.....	22,106
1907 (*o*) ...	631	7,355	(*q*) 3,167	20,148	5,113	274,178	......	8,388

(*a*) From Tables of the *Trade and Navigation of the Dominion of Canada.* (*b*) Fiscal year ending June 30. (*c*) Includes regulus and salts of antimony. (*d*) Asbestos in any form except crude, and all manufactures of. (*e*) Includes manufactures. (*f*) Includes coal dust. (*g*) Coin, gold and silver, except U. S. silver coin. (*h*) Includes black lead, and crucibles (clay or graphite). (*i*) Includes Canadian lead ore refined in the United States. (*k*) Calendar year. (*l*) Fine copper contained in ore, matte, regulus, etc. (*m*) Of foreign production. (*n*) Includes chromic iron ore. (*o*) Returns for the 9 months of the fiscal year ending March 31. (*p*) Lead contained in ore, etc. (*q*) Gallons.

FRANCE.

In the following tables are given the statistics of mineral and metal production in France and the French colonies—Algeria, New Caledonia and Tunis—together with the foreign commerce of France in mineral and metal products:

MINERAL AND METALLURGICAL PRODUCTION OF FRANCE. (*a*)
(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic Ore.	Asphaltum.	Barytes.	Bauxite.	Bitumen. (*c*)
		Ore.	Metal.					
1896	370	5,675	969	....	17,717	2,791	33,820	225,784
1897	470	4,685	1,033	....	17,982	3,209	41,740	233,328
1898	565	4,433	1,226	....	18,832	2,763	36,723	229,108
1899	763	7,392	1,499	2,600	22,100	4,058	48,215	258,449
1900	1,026	7,843	1,573	4,705	25,228	3,635	58,530	266,474
1901	1,200	9,867	1,786	7,491	20,391	4,145	76,620	249,655
1902	1,355	9,715	1,725	5,372	....	4,323	96,900	258,295
1903	1,570	12,380	2,748	6,658	....	5,731	133,890	243,295
1904	1,650	9,065	2,116	3,117	22,000	6,944	75,640	227,177
1905	1,905	12,543	2,396	3,627	20,000	5,504	103,207	188,403
1906	3,396	18,567	3,433	6,534	38,231	11,680	117,781	196,375

Year.	Cement.	Clay Products.		Coal.			Copper.	
		Potter's Clay.	Fire Clay.	Bituminous.	Lignitic.	Peat.	Ore.	Metal.
1896	934,624	246,677	291,690	28,750,452	439,448	130,207	106	6,544
1897	976,813	270,292	318,185	30,337,207	460,422	98,067	956	7,376
1898	1,072,025	260,362	295,913	31,826,127	529,977	104,265	382	7,834
1899	1,144,271	310,220	367,432	32,256,148	606,564	99 230	2,021	6,640
1900	1,147,670	331,396	329,561	32,721,562	682,736	95,630	3,031	6,446
1901	1,127,206	341,407	293,208	31,633,300	691,700	118,433	3,413	7,000
1902	962,930	(*h*)4,541,359	295.341	29,365,047	632,423	109,941	828	6,300
1903	898,393	(*h*)4,734,924	253,460	34,217,661	688,757	100,348	10,892	6,921
1904	903,632	(*h*)4,968,936	220,409	33,502,394	665,572	95,716	2,756	6,900
1905	922,531	(*h*)5,129,393	215,587	35,218,000	709,000	98,500	5,068	7,576
1906	1,257,861	(*h*) 5,961,564	283,505	33,458,000	738,000	92,469	2,547	5,770

Year.	Gold.	Gypsum.		Iron.				Lead.	
		Crude.	Calcined.	Ore.	Pig.	Wrought Iron.	Wrought Steel.	Ore. (*d*)	Pig. (*e*)
1896	$217,308	264,187	1,429,550	4,069,390	2,339,537	828,758	916,817	19,042	8,232
1897	183,416	292,753	1,369,269	4,582,236	2,484,191	584,540	994,891	21,212	9,916
1898	177,435	303,531	1,449,384	4,731,394	2,525,100	766,000	1,174,000	23,342	10,920
1899	179,429	263,879	1,372,067	4,985,702	2,578,400	834,000	1,240,000	17,505	15,981
1900	134,904	192,916	1,405,845	4,676,740	2,714,298	672,172	1,226,537	24,276	15,210
1901	85,727	355,995	1,623,710	4,260,747	2,388,823	612,362	1,175,454	20,644	21,000
1902	(*b*)	219,487	1,572,687	5,003,782	2,405,000	639,600	1,245,800	22,634	19,000
1903	(*b*)	162,766	1,468,830	6,219,541	2,840,517	598,910	1,305,709	23,080	23,258
1904	(*b*)	106,173	1,481,303	7,022,841	2,999,787	554,632	1,482,708	14,173	18,800
1905	8,260	78,832	1,299,313	7,395,409	3,077,000	670,000	1,442,000	12,118	24,100
1906	9,600	79,568	1,297,861	8 ,4 81,423	3,314,100	747,900	1,683,500	11,795	25,614

Year.	Lime.	Manganese Ore.	Millstones.	Mineral Paints. (Ochers.)	Nickel.	Phosphate Rock.	Pyrites.	Salt.
1896	2,224,847	31,318	28,237	27,499	1,545	582,667	282,064	1,042,614
1897	2,201,428	37,212	32,175	32,299	1,245	535,390	303,488	948,003
1898	2,339,850	31,935	(f)38,929	33,780	1,540	568,558	310,972	999,283
1899	2,343,377	39,897	41,535	32,750	1,740	645,868	318,832	1,193,532
1900	2,377,110	28,992	41,103	33.080	1,700	587,919	305,073	1,088,634
1901	2,443,062	22,304	33,286	35,704	1,800	535,676	307,447	910,000
1902	4,796,807	12,536	34,504	34,770	1,600	543,900	318,235	863,927
1903	4,727,543	11,583	35,031	34,042	1,500	475,783	322,118	967,531
1904	4,583,522	11,254	37,409	34,945	1,500	423,521	271,544	1,153,754
1905	3,694,725	6,751	33,468	37,800	1,800	476,720	267,114	1,130,088
1906	3,869,772	11,189	32,407	35,550	1,750	469,408	265,261	1,335,420

Year.	Slate.		Stone.				Sulphur Ore. (g)	Zinc.	
	Roofing.	Slabs.	Building.	Limestone. (Flux)	Marble.	Paving Blocks.		Ore.	Metal.
1896	283,352	1,148	10,089,845	721,296	119,168	677,213	9,720	81,346	35,585
1897	310 820	1,143	10,105,438	709,562	118,675	568,677	10,723	83,044	38,067
1898	311,911	1,318	9,989,416	695,501	124,161	568,483	9,818	85,550	37,155
1899	299,307	1,162	10,587,789	924,945	191,030	621,799	11,744	84,813	39,274
1900	290,204	1,325	9,974,347	1,040,805	154,414	659,125	11,551	67,059	36,305
1901	288,508	1,304	10,277,098	1,083,372	123,506	604,464	7,000	61,539	37,600
1902	320,098	1,410	10,725,607	628,272	118,894	554,854	8,021	57,982	36,300
1903	382,461	1,404	10,713,356	704,736	136,615	577,554	7,375	66,922	37,416
1904	382,435	2,136	10,515,909	734,502	118,654	568,943	5,447	52,842	41,600
1905	375,874	1,435	10,152,679	730,119	115,222	608,258	4,637	62,150	43,200
1906	385,365	1,690	9,386,836	770,335	119,488	592,331	2,713	53,466	46,536

(a) From *Statistique de l'Industrie Minerale.* (b) Not reported. (c) Includes pure bitumen, bituminous schist and sand, and asphaltic limestone. (d) Argentiferous lead ore. (e) Lead produced from native ores only. (f) Finished product. (g) Sulphur and limestone impregnated with sulphur. (h) Includes potter's clay, white clay for stucco, kaolin and clay for bricks and tiles.

MINERAL PRODUCTION OF ALGERIA. (a)

(In metric tons.)

Year.	Antimony Ore.	Clays.	Copper Ore.	Gypsum.		Iron Ore.	Lead-silver Ore.
				Crude.	Plaster.		
1896	658	48,297	427	300	29,870	374,476	117
1897	781	67,180	289	350	29,120	441,467	145
1898	138	78,690	488	150	29,750	473,569	120
1899	200	88,600	472	200	31,800	550,921	389
1900	93	94,000	....	500	37,100	174,000	222
1901	....	119,195	7,267	600	34,740	161,303	1,614
1902	39	122,850	1,955	600	35,500	525,012	26
1903	490	125,800	100	300	33,000	588,893	499
1904	160	125,410	1,804	350	38,420	468,737	511
1905	....	133,100	1,784	....	34,743	568 609	7,470
1906	50	131,700	2,786	....	27,950	779,826	11,246

Year.	Lime.		Marble.	Onyx.	Phosphate Rock	Salt.	Sand and Gravel.	Zinc Ore.
	Hydraulic.	White.						
1896	20,000	9,450	900	900	165,738	19,658	41,400	17,587
1897	20,425	9,215	1,660	364	228,141	23,222	80,860	32,269
1898	13,000	12,975	985	219	269,500	21,300	72,185	29,800
1899	12,000	13,645	225	217	324,983	17,378	72,760	42,970
1900	12,000	13,700	....	228	319,422	18,325	71,860	30,281
1901	12,000	15,000	....	294	265,000	18,518	86,727	26,913
1902	....	....	375	150	305,174	27,263	72,180	33,139
1903	....	....	700	67	320,834	26,329	46,720	43,313
1904	....	....	530	121	343,317	18,563	51,020	47,192
1905	28,990	15,220	451	270	334,784	26,986	61,900	67,922
1906	50,400	21,000	460	216	333,531	22,615	74,320	74,351

(a) From *Statistique de l'Industrie Minerale.*

MINERAL PRODUCTION OF NEW CALEDONIA. (a)
(In metric tons.)

Year.	Chrome Iron Ore.	Cobalt Ore.	Copper Ore.	Nickel Ore.
1897	3,949	3,200	2,200	26,464
1898	7,712	2,373	*Nil.*	74,614
1899	12,634	3,294	6,349	103,908
1900	10,474	2,438	2	100,319
1901	17,451	3,123	6,349	132,814
1902	10,281	7,512	3,720	129,653
1903	21,437	8,292	10	77,360
1904	42,197	8,964	*Nil.*	98,655
1905	51,374	7,920	*Nil.*	125,289
1906	84,241	2,600	207	118,890
1907	31,552	3,943	....	101,708

(a) From *Statistique de l'Industrie Minerale.*

MINERAL PRODUCTION OF TUNIS. (a)
(In metric tons.)

Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.
1896	5,500	(b)	1,000	12,100
1897	8,100	2,123	(b)	11,830
1898	7,300	2,375	(b)	21,477
1899	8,850	2,263	70,000	20,079
1900	9,160	6,864	178,000	16,596
1901	16,900	8,158	172,000	17,879
1902	21,600	12,892	264,930	18,400
1903	18,846	12,752	352,088	21,262
1904	23,600	16,800	455,197	27,200
1905	52,900	15,200	522,000	37,100
1906	62,600	14,800	796,000	32,400

(a) From *Statistique de l'Industrie Minerale.* (b) Not reported.

MINERAL IMPORTS OF FRANCE. (a)
(In metric tons or dollars. 5 f.=$1.)

Year.	Alum.	Bitumen. (f)	Borax.	Bromides.	Cement.	Coal and Coke.	Copper. Ore.	Copper. Ingot and Mfres.
1895	109	43,975	442	12	13,441	10,261,069	10,450	38,196
1896	41	30,954	255	13	14,395	10,180,449	8,584	46,830
1897	54	29,931	264	18	15,141	10,457,255	11,960	54,460
1898	27	20,385	139	30	11,290	10,445,090	8,779	52,976
1899	34	30,770	123	46	13,640	11,896,030	8,517	58,419
1900	23	39,598	111	10	13,612	14,601,981	9,766	61,638
1901	39	28,888	128	3	16,232	13,925,623	13,383	47,035
1902	36	26,053	141	3	15,720	13,137,720	17,862	54,484
1903	138	27,573	312	9	21,152	14,029,687	9,796	59,126
1904	370	17,178	3,113	17	21,702	13,936,475	9,942	69,183
1905	63	24,606	1,736	31	21,954	13,910,523	14,252	70,101
1906	105	99,336	189	93	24,974	17,848,284	11,932	71,151

Year.	Copper. Sulphate.	Copper. Oxide.	Cobalt Oxide.	Gold. Bullion and Specie.	Iron. Ore.	Iron. Pig.	Iron. Iron and Steel, Mfres. of.	Iron. Sulphate.	Iron. Oxide.
1895	24,404	24	5	$50,775,039	1,651,369	36,247	66,240	3,882	855
1896	33,803	22	5	60,167,745	1,362,043	18,323	48,423	3,086	897
1897	30,132	29	9	58,143,077	2,137,860	35,633	60,804	1,353	1,125
1898	30,897	52	9	39,881,575	2,032,240	(b)	47,325	896	1,021
1899	21,733	36	9	63,697,020	1,950,665	(b)	64,178	1,698	1,037
1900	22,820	84	9	90,408,723	2,119,003	149,755	118,152	1,589	1,022
1901	15,313	162	8	85,485,000	1,662,875	61,085	77,742	45	1,001
1902	22,273	111	10	88,091,400	1,563,334	38,521	60,697	17	1,051
1903	25,428	129	11	62,154,022	1,832,820	121,726	119,799	36	1,207
1904	30,856	142	69	133,737,561	1,738,514	135,252	125,709	319	1,151
1905	23,805	57	35	158,388,076	2,151,954	122,102	150,480	709	1,330
1906	15,358	97	41	88,166,800	2,015,550	156,618	342,411	132	1,311

Year.	Kaolin.	Lead.			Lime.		Manganese Ore.
		Ore.	Carbonate.	Pig, Scrap and Mfres.	Common and Hydraulic.	Chloride of.	
1895	....	5,032	1,077	66,241	246,677	1,047	41,400
1896	38,703	5,569	892	79,752	283,707	2,033	61,600
1897	42,384	13,981	1,327	86,589	321,047	1,713	85,500
1898	40,352	14,377	1,376	74,902	346,000	1,288	100,243
1899	36,904	12,637	2,029	67,149	321,610	1,887	106,630
1900	39,842	19,772	1,739	70,857	399,092	1,215	120,790
1901	41,972	15,430	1,789	59,051	374,281	1,400	94,365
1902	41,165	13,121	2,223	58,694	359,210	2,130	85,629
1903	47,534	20,172	2,040	75,416	386,612	919	109,930
1904	50,465	25,731	2,221	76,198	403,679	1,679	105,652
1905	52,603	35,103	2,306	73,938	409,447	406	140,871
1906	54,660	43,137	2,072	67,651	437,447	593	127,703

Year.	Nickel.		Petroleum.	Phosphate Rock.	Plaster.	Platinum. *Kg.*	Potassium.	
	Ore.	Metal.					Chloride.	Chromate (*h*)
1895	10,303	252	258,700	139,600	2,412	926	3,524	2,875
1896	15,756	425	272,693	256,888	1,774	2,117	11,499	2,838
1897	17,441	316	288,671	313,608	1,869	1,069	11,630	2,852
1898	24,935	330	291,961	336,842	2,040	505	10,929	2,890
1899	28,620	286	306,078	242,021	2,260	817	13,335	3,147
1900	17,687	299	302,482	283,921	3,648	2,398	13,524	3,293
1901	39,497	252	225,962	275,285	2,844	1,857	13,299	2,784
1902	58,374	301	148,170	302,898	2,440	2,940	10,802	2,861
1903	13,933	427	(*g*)476,230	343,012	2,664	3,764	12,275	2,760
1904	20,698	313	(*g*)435,730	419,720	2,674	5,650	14,734	2,618
1905	49,698	632	(*g*)512,727	447,738	1,983	4,023	21,819	2,619
1906	44,960	480	(*g*)213,462	533,213	2,572	5,708	26,523	3,024

Year.	Potassium. (*Cont'd*)		Pyrites.	Quicksilver.		Sal-Ammoniac.	Salt.	Sodium.	
	Nitrate.	Carbonate		Ore.	Metal.			Hydrate.	Nitrate.
1895	775	796	67,930	23	178	9,923	17,528	1,021	$8,624,200
1896	2,614	1,526	45,788	25	234	15,256	17,191	1,109	9,025,400
1897	1,309	1,769	69,470	24	248	27,454	32,917	1,378	8,105,400
1898	1,008	2,418	71,569	19	221	20,426	35,863	1,772	8,026,400
1899	1,015	2,779	109,696	21	276	12,210	37,970	1,494	9,341,600
1900	1,928	2,768	156,825	22	161	15,205	32,045	1,062	11,995,820
1901	757	2,520	205,617	23	205	9,268	32,347	869	10,526,400
1902	1,547	1,539	170,783	24	224	15,446	32,505	643	9,372,600
1903	1,530	3,019	205,322	20	220	12,462	48,556	781	10,810,775
1904	2,117	3,781	230,097	22	208	13,744	46,232	1,068	9,074,859
1905	1,022	3,542	271,684	. .	228	11,639	45,241	860	11,336,752
1906	684	2,206	349,514	....	242	18,146	38,361	614	13,678,848

Year.	Sulphur.	Sulphuric Acid.	Superphosphate of Lime.	Tin.		Zinc.	
				Ore.	Metal.	Ore.	Metal.
1895	110,989	3,461	150,758	104	7,691	41,622	25,652
1896	111,515	3,995	185,602	7	8,400	50,899	33,459
1897	136,118	3,147	195,853	149	7,642	58,074	31,211
1898	130,289	4,666	178,569	357	9,247	60,481	32,342
1899	120,062	4,583	171,631	486	6,907	78,192	25,516
1900	133,531	4,254	143,437	512	7,324	66,178	33,144
1901	101,301	5,386	165,361	365	7,314	74,553	29,812
1902	85,839	7,793	116,093	748	8,575	69,451	36,564
1903	109,594	13,241	89,229	1,808	9,873	67,258	39,305
1904	148,547	11,212	72,921	1,344	9,352	88,083	35,737
1905	129,877	10,915	31,729	1,362	9,898	105,069	29,163
1906	131,678	5,268	44,502	1,038	9,759	107,259	32,631

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (*a*)
(In metric tons.)

Year.	Aluminum.	Antimony.		Cement.	Coal.	Copper.		Gold. *Kg.*(*d*)
		Ore.	Metal.			Ore. (*c*)	Metal	
1895	110	832	68	(*b*)	(*b*)	1,772	8,829	1,353
1896	793	736	74	242,247	1,044,820	1,261	10,494	2,193
1897	224	623	61	244,504	1,142,195	2,000	12,667	3,335
1898	192	616	101	241,150	1,320,616	1,783	14,350	1,812
1899	256	304	255	244,480	1,229,090	2,078	17,949	2,622
1900	324	154	336	232,577	1,201,210	9,197	16,791	883
1901	307	645	741	242,010	908,583	16,066	14,776	1,869
1902	748	595	666	210,590	910,760	20,489	14,423	1,517
1903	666	904	1,358	233,835	2,238,735	12,487	11,403	3,139
1904	664	1,191	720	260,686	2,384,928	14,258	12,663	1,537
1905	928	981	815	275,503	(*i*) 3,348,010	13,260	13,800	5,740
1906	1,522	4,094	1,946	329,879	(*i*) 2,732,428	8,098	27,412	11,727

Year.	Iron.				Lead.		Manganese Ore.	Millstones. *Number.*
	Ore.	Pig.	Bars.	Steel.	Ore.	Metal.		
1895	236,923	150,540	....	29,074	8,670	8,037	16,193	....
1896	238,430	195,212	24,721	44,795	8,597	10,856	10,913	196,685
1897	299,589	108,645	39,894	45,809	12,007	10,364	19,464	158,979
1898	236,169	162,991	27,424	47,278	10,216	3,663	12,229	203,584
1899	291,346	153,792	29,112	33,584	3,909	1,163	12,289	112,620
1900	371,799	114,361	18,763	19,535	2,345	958	8,392	65,436
1901	258,925	96,463	25,220	56,347	3,490	718	5,289	52,383
1902	422,677	213,081	23,828	121,932	2,414	648	1,948	45,647
1903	714,173	196,444	40,533	215,737	2,313	13,048	717	11,557
1904	1,219,149	191,819	40,374	246,738	1,860	13,467	1,392	14,479
1905	1,355,932	218,227	67,240	343,612	3,064	12,903	662	13,078
1906	1,759,443	143,142	58,826	236,617	1,846	8,692	4,103	16,169

Year.	Nickel Refined.	Phosphate. Rock.	Plaster.	Pyrites.	Silver. *Kg.* (*e*)	Tin. (metal)	Zinc.	
							Ore.	Spelter, Sheets and Scrap.
1895	408	....	....	37,968	13,567	650	61,291	5,849
1896	490	48,719	89,952	44,232	9,849	744	62,415	10,485
1897	498	69,188	107,823	54,367	5,374	651	79,909	10,977
1898	526	93,742	106,790	60,406	1,886	587	60,664	16,995
1899	280	70,517	112,520	53,395	....	666	76,104	14,958
1900	599	89,135	108,387	64,530	15,470	716	54,663	12,712
1901	1,031	81,405	101,063	52,952	16,745	438	42,995	15,022
1902	397	62,375	110,270	63,920	17,184	654	47,724	16,158
1903	720	72,252	131,245	119,173	43,690	1,994	62,731	12,657
1904	906	78,612	139,551	40,833	23,105	2,300	57,780	19,063
1905	1,583	55 240	124,561	21,257	66,904	2,611	72,512	17,802
1906	1,091	81,660	142,339	26,216	87,952	2,576	68,211	23,033

(*a*) From *L'Economiste Francais* (representing the *Commerce Special*) except for last four years, which are from *Tableau Général du Commerce et de la Navigation.* (*b*) Not reported. (*c*) Includes matte. (*d*) Gold and platinum in ore sheets, leaves or threads. (*e*) Silver in ore, sheets, leaves, wire, etc. (*f*) Includes bitumen, bituminous schist and sands and asphaltic limestone. (*g*) Crude and refined. Transposition from hectoliters to tons was performed by assuming specific gravity of petroleum to be 0.9. (*h*) Includes chromate of soda. (*i*) Includes coke.

GERMANY.

The mineral production and foreign commerce of the German Empire are given in the following tables in metric tons unless otherwise specified, or in dollars, on the basis of four marks to the dollar.

MINERAL PRODUCTION OF GERMANY. (a)

Year.	Alum.	Aluminum Sulphate.	Arsenic.		Asphaltum.	Bora-cite	Cadmium. *Kg.*	Coal.	
			Ore.	Salts.				Bituminous.	Lignitic.
1897.....	2,995	37,053	3,777	2,989	61,645	198	15,531	91,054,982	29,419,503
1898.....	4,069	35,366	3,527	2,679	67,649	230	14,943	96,309,652	31,648,898
1899.....	3,358	37,693	3,834	2,423	74,770	183	13,608	101,639,753	34,204,666
1900.....	4,355	44,372	4,379	2,415	89,685	232	13,553	109,290,237	40,498,019
1901.....	4,145	46,807	4,035	2,549	90,193	184	13,144	108,539,444	44,479,970
1902.....	4,108	47,905	3,959	2,828	88,374	196	......	107,473,933	43,126,281
1903.....	3,934	49,727	4,369	2,768	87,454	159	16,565	116,637,765	45,819,488
1904.....	3,850	55,881	4,390	2,829	91,736	135	25,245	120,815,503	48,635,080
1905.....	4,127	52,892	4,913	2,535	103,006	183	24,568	121,298,607	52,512,062
1906.....	4,494	55,969	6,249	3,052	117,413	161	......	137,117,926	56,419,567
1907.....	4,200	59,473	4,872	2,904	126,649	114	......	143,168,301	62,559,364

Year.	Cobalt, Nickel and Bismuth Ores.	Copper.				Gold.	Graphite.
		Ore.	Matte. (b)	Ingots.	Sulphate.		
1897................	3,355	700,619	315	29,408	5,549	$1,848,114	3,861
1898................	3,157	702,781	62	30,695	4,352	1,891,974	4,593
1899................	1,270	733,619	95	34,634	5,142	1,731,153	5,196
1900................	4,495	747,749	4,207	30,929	5,076	2,030,200	9,248
1901................	10,479	777,339	365	31,317	5,192	1,830,835	4,435
1902................	12,433	761,921	447	30,578	4,997	1,770,361	5,023
1903................	14,607	772,695	583	31,214	5,200	1,709,223	3,720
1904................	14,016	798,214	641	30,264	6,584	1,819,538	3,784
1905................	10,848	793,488	1,635	31,713	6,988	2,611,812	4,921
1906................	......	768,523	771	32,275	6,757	2,931,750	4,055
1907................	......	771,227	......	31,946	5,284	3,267,750	4,033

Year.	Iron and Steel.					Lead.		
	Iron Ore.	Pig Iron. (c)	Castings.	Steel.	Sulphate. (d)	Ore.	Pig.	Litharge
1897..............	15,465,980	6,881,466	1,473,211	6,248,141	10,351	150,178	118,881	3,441
1898..............	15,901,263	7,312,766	1,597,434	6,941.278	10,422	149,311	132,742	3,857
1899..............	17,989,635	8,153,133	1,776,878	7,532,524	10,931	144,370	129,225	3,562
1900..............	18,964,294	8,520,540	1,812,603	7,377,275	10,913	148,257	121,513	3.088
1901..............	16,570,182	7,880,087	1,520,617	7,033,438	11,148	153,341	123,098	4,101
1902..............	17,963,591	8,529,900	1,575,525	8,317,231	......	167,855	140,331	4,197
1903..............	21,230,650	10,017,901	1,721,781	9,226,898	12,243	165,991	145,319	4,428
1904..............	22,047,393	10,058,273	1,879,879	9.239,302	13,585	164,440	137,580	4,332
1905..............	23,444,073	10,875,061	2,045,477	10,309,690	12,949	152,725	152,590	3,786
1906..............	26,734,570	12,292,819	....	11,135,085	13,376	140,914	150,741	4,137
1907..............	27,697,127	12,875,159	....	....	14,033	147,272	142,271	.....

Year.	Magnesium Salts. Chloride.	Magnesium Salts. Sulphate.	Manganese Ore.	Nickel. (e).	Petroleum.	Potassium Salts. Chloride.	Potassium Salts. Kainite. (f)	Potassium Salts. Sulphate.	Potassium Salts. Potassium and Magnesium Sulphate.	Potassium Salts. Unspecified.
1897........	18,014	35,072	46,427	1,464	23,303	168,001	992,389	13,774	7,812	953,798
1898........	19,819	30,295	43,354	1,691	25,989	191,347	1,103,643	18,853	13,982	1,105,212
1899........	21,370	39,540	61,329	1,747	27,027	207,506	1,108,159	26,103	9,765	1,384.972
1900........	19,397	48,591	59,204	1,989	50,375	271,512	1,227,873	30,853	15,368	1,822,758
1901........	21,018	46,714	56,691	2,207	44,095	294,666	1,498,569	37,394	15,612	2,036,325
1902........	19,658	39,262	49,812	2,196	49,725	267,512	1,322,623	28,278	18,147	1,962,384
1903........	22,990	37,844	47,994	2,637	62,680	280,248	1,557,243	36,674	23,631	2,073,720
1904........	25,730	39,412	52,886	3,063	89,620	297,238	1,905,893	43,959	29,285	2,179,471
1905........	29,017	58,568	51,463	3,317	78,869	373,177	2,387,643	47,994	34,222	2,655,845
1906......	38,468	43,041	52,485	.....	81,350	403,387	2,720,594	54,490	35,211	2,821,073
1907........	32,891	41,105	74,683	.....	106,379	473,138	2,624,412	60,292	33,368	3,124,955

Year	Pyrites.	Salt. Rock.	Salt. Evaporated.	Silver and Gold Ore.	Silver. *Kg.*	Sodium Sulphate.	Sulphur.	Sulphuric Acid.
1897............	133,302	763,412	543,272	9,708	448,068	68,822	2,317	702,445
1898............	136,849	807,792	565,683	14,702	480,578	69,111	1,954	754,151
1899............	144,623	861,123	571,058	13,506	467,590	79,062	1,663	813,141
1900............	169,447	926,563	587,464	12,593	415,735	90,468	1,445	829,376
1901............	157,433	985,050	578,751	11,577	403,796	76,066	963	835,000
1902............	165,225	1,010,412	572,846	11,724	430,610	90,742	.....	894,409
1903............	170,867	1,095,541	598.394	11,467	396,253	83,087	219	928,190
1904............	174,782	1,079,868	621,064	10,405	389,827	75,171	209	963,384
1905............	185,368	1,165,495	612,062	10,286	399,775	68,454	205	1,228,211
1906............	196,971	1,235.041	635,171	8,066	393,442	81,175	178	1,335,128
1907............	196,320	1,285,137	665,552	8,280	386,933	80,347	176	1,380,016

Year.	Tin. Ore.	Tin. Block.	Tin. Chloride.	Uranium and Tungsten Ores.	Zinc. Ore.	Zinc. Spelter.	Zinc. Sulphate.
1897........	55	929	.....	38	663,850	150,739	5,488
1898........	51	993	......	50	641,706	154,867	6,104
1899........	72	1,481	.....	50	664,536	153,155	7,117
1900	80	2,031	(*g*) 143	43	639,215	155,790	6,027
1901........	82	1,464	(*g*) 135	43	647,496	166,283	5,552
1902........	104	2,779	.....	31	702,504	174,927	.....
1903........	110	3,065	1,064	35	682,853	182,548	5,994
1904........	99	4,216	816	23	715,732	193,058	6,185
1905........	123	5,233	811	26	731,271	198,208	5,896
1906	...	6,597	987	..	704,590	205,691	6,092
1907........	...	5,864	1,812	..	698,425	208,195	5,145

(*a*) From the *Vierteljahrshefte zur Statistik des Deutschen Reichs*. (*b*) Includes black copper. (*c*) Includes ferromanganese and spiegeleisen. (*d*) Contains a small quantity of copper and iron sulphate mixed. (*e*) Includes nickeliferous by-products, metallic bismuth, and uranium compounds. (*f*) Compound of potassium chloride and magnesium sulphate. (*g*) Includes nickel sulphate.

MINERAL PRODUCTION OF BADEN. (*a*)
(In metric tons and dollars; 4 marks=$1.)

Year.	Alumina Sulphate.	Barytes.	Coal.	Fire Clay.	Gypsum.	Iron. Cast, Foundry.
1896..	1,824	130	4,001	6,819	32,801	31,356
1897..	1,824	400	4,752	11,450	40,702	36,235
1898..	2,051	1,100	4,133	5,112	28,037	39,988
1899..	2,153	2,430	4,700	4,775	29,419	53,608
1900..	2,286	2,970	4,930	3,096	26,381	50,102
1901..	2,260	3 991	3,650	2,530	28,183	40,100
1902..	2,374	6,234	2,078	3,188	33,150	40,973
1903..	2,498	8,857	1,990	3,870	29,423	45,233
1904..	2,392	9,078	1,485	3,420	26,984	64,320
1905..	2,581	11,094	668	3,964	28,823	74,128
1906..	2,583	11,984	1,000	4,480	25,643	81,387
1907..	2,644	9,303	2,075	992	29,153	98,430

Year.	Iron.—*Continued.*		Lead Ores.	Limestone.	Salt.
	Pig.	Wrought.			
1896	3,418	1,118	(*b*)	116,913	29,227
1897	3,875	1,167	(*b*)	137,670	31,445
1898	3,875	1,167	(*b*)	164,979	31,445
1899	3,830	1,402	(*b*)	201,015	31,197
1900	3,532	1,364	67	221,097	32,699
1901	8,739	1,158	369	168,245	32,835
1902	12,663	1,052	450	168,848	32,192
1903	7,666	863	350	222,908	32,383
1904	7,687	783	265	240,951	32,148
1905	8,053	842	264	240,700	31,393
1906	11,068	466	246	263,992	31,288
1907	10,818	533	278	310,568	32,078

Year.	Quartz Sand.	Stone Porphyry.	Sulphuric Acid.	Tripoli.	Zinc Ore.
1896	1,198	(*b*)	14,226	9.0	(*b*)
1897	1,648	28,000	13,365	9.0	(*b*)
1898	1,694	7,650	13,365	6.0	(*b*)
1899	1,461	22,261	13,660	11.8	357
1900	2,668	23,421	15,938	9.4	3,004
1901	1,923	18,880	17,081	8.0	2,870
1902	1,292	22,168	19,265	10.5	2,958
1903	2,072	13,735	19,755	10.6	3,171
1904	2,700	21,952	35,517	11.7	5,063
1905	3,643	10,454	40,781	11.8	4,046
1906	1,461	8,602	38,655	15.0	1,466
1907	5,141	5,370	42,831	24.5	2,198

(*a*) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (*b*) Not reported.

MINERAL PRODUCTION OF BAVARIA. (*a*)
(In metric tons; 4 marks=$1.)

Year.	Barytes.	Clay		Coal.	Coal. (Lignite).
		Fire Clay.	Kaolin.		
1896	3,397	110,174	19,080	900,080	35,934
1897	3,365	144,425	24,086	917,022	39,043
1898	4,339	282,994	29,196	964,611	38,663
1899	6,214	271,792	25,822	1,004,421	35,736
1900	10,515	187,501	58,795	1,185,296	39,165
1901	8,711	143,028	35,450	1,203,792	25,224
1902	8,034	198,882	92,073	1,233,568	27,337
1903	8,642	173,919	88,140	1,356,556	25,189
1904	9,411	173,126	95,160	1,341,925	53,517
1905	10,030	210,968	99,910	1,317,951	154,128
1906	19,817	277,008	98,138	1,381,175	140,290
1907	21,500	309,120	115,387	1,495,895	286,256

Year.	Copperas and other Sulphate.	Emery.	Feldspar.	Fluorspar.	Graphite.	Gypsum.	Iron.
							Ore.
1896	601	249	1,315	5,218	5,248	28,799	161,279
1897	981	217	1,689	4,904	3,861	26,153	172,699
1898	886	280	1,949	4,440	4,593	25,688	171,987
1899	900	399	287	3,631	5,196	29,727	181,981
1900	916	414	460	7,456	9,248	35,484	178,441
1901	590	366	788	5,220	4,435	3,581	158,820
1902	691	225	447	5,460	5,023	31,701	157,375
1903	814	220	1,060	3,410	3,719	30,894	162,500
1904	893	265	1,866	4,770	3,784	22,766	180,342
1905	844	255	1,710	4,413	4,921	46,247	182,389
1906	836	320	1,740	5,570	4,055	50,763	203,596
1907	850	326	2,125	4,780	4,033	48,975	277,280

Year.	Iron—*Continued.*				
	Bar.	Cast, 1st Fusion	Cast, 2d Fusion	Pig.	Wire.
1896	53,573	114	71,006	79,621	243
1897	58,200	138	78,008	83,418	252
1898	58,342	97	84,227	84,144	323
1899	61,415	(*b*)	92,459	83,821	111
1900	49,727	29	89,692	82,327	221
1901	29,978	76	76,191	72,071	12,661
1902	38,429	56	81,874	83,123	17,664
1903	36,853	41	89,804	90,168	21,064
1904	37,780	40	108,025	92,200	17,829
1905	36,459	24	112,875	94,242	17,375
1906	38,508	....	122,115	97,812	21,068
1907	36,883	....	138,659	98,143	18,944

Year.	Iron.—*Continued.* Steel.	Marl. (For Cement)	Mineral Paints.	Pyrites.	Rock Salt.
1896	101,954	94,481	8,667	1,997	708
1897	115,530	97,831	8,673	2,211	1,161
1898	120,623	110,757	8,748	2,304	736
1899	134,007	220,716	9,287	2,516	802
1900	135,411	180,032	11,507	2,120	1,298
1901	109,464	76,663	84,929	2,649	1,319
1902	115,354	178,301	13,947	2,635	832
1903	127,141	200,407	19,486	2,324	879
1904	125,483	170,698	19,107	3,427	1,139
1905	134,755	231,310	18,285	3,301	911
1906	150,129	230,271	22,304	3,918	1,053
1907	150,148	230,583	21,219	5,085	1,393

Year.	Salt, Brine.	Silica (Quartz Sand).	Slate.	Soapstone.	Sodium Sulphate.	Stone. Flagstones.
1896	40,400	29,868	1,565	3,051	663	20,559
1897	41,533	31,678	1,496	2,464	2,318	14,647
1898	39,717	45,907	3,956	1,912	2,332	16,720
1899	41,207	39,922	2,066	2,197	1,570	20,195
1900	44,431	42,671	1,904	1,977	1,821	16,268
1901	41,217	37,710	1,024	2,291	1,893	1,550
1902	....	....	....	....	....	....
1903	41,782	155,921	2,074	1,866	....	8,790
1904	43,048	274,346	1,486	1,709	....	12,958
1905	42,591	248,872	1,234	1,872	....	9,070
1906	....	....	....	....	....	....
1907	....	....	....	....	....	....

Year.	Stone—*Continued.*					Sulphuric Acid.
	Granite and other Massive Rocks.	Limestone.	Lithographic.	Sandstone.	Whetstone.	
1896	658,581	238,434	10,868	235,518	88	7,064
1897	663,749	224,550	13,941	242,112	95	7,041
1898	678,171	214,309	12,029	296,139	85	103,385
1899	490,712	267,180	11,962	315,786	81	123,273
1900	523,279	297,635	16,030	314,154	25	123,910
1901	521,288	356,239	9,500	355,850	10	115,775
1902	....	....	....	....	....	....
1903	1,493,677	730,279	9,890	542,110	83	....
1904	899,671	824,971	13,836	576,561	50	....
1905	950,006	890,109	11,360	648,303	25	....
1906	....	....	....	....	....	....
1907	....	....	....	....	....	....

(*a*) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (*b*) Not reported

MINERAL PRODUCTION OF PRUSSIA. (*a*)
(Metric tons; 4 marks=$1.)

Year.	Alum Shale.	Antimony and Alloys.	Arsenic Products.	Arsenic Ore.	Asphalt.
1897	129	1,552	1.924	3,377	11,466
1898	107	2,612	1,624	3,298	12,822
1899	145	3,003	1,469	3,265	16,458
1900	103	3,025	1,585	3,531	23,891
1901	611	2,404	1,446	3,050	26,450
1902	219	3,542	1,514	2,909	28,035
1903	580	3,224	1,583	3,538	23,518
1904	106	2,774	1,573	3,527	26,348
1905	97	2,795	1,493	4,022	28,872
1906	634	2,953	1,551	5,430	32,270

Year.	Boracite.	Cadmium. *Kg.*	Coal.	Coal. (Lignite.)	Cobalt Ore.	Cobalt Products.
1897	185	15,531	84,253,393	24,222,911	121	51
1898	216	14,943	89,593,528	26,035,814	34	44
1899	171	13,608	94,740,829	28,418,598	17	46
1900	217	13,533	101,966,158	34,007,542	4	52
1901	164	13,144	101,203,807	37,491,412	36	66
1902	172	12,625	100.115,315	36,228,285	76	74
1903	135	16,565	108,809,384	38,462,766	65	87
1904	115	25,245	112,755,621	41,153,576	41	85
1905	151	24,568	113,000,657	44,148,751	22	99
1906	124	21,486	128.295.948	47,912,721	7	98

Year.	Copper.	Copper and Iron Sulphate.	Copper Ore.	Copper Matte.	Copper Sulphate.	Epsom Salt.
1897	25,997	225	690,338	274	2,689	2,248
1898	27,216	120	691,866	62	1,701	2,061
1899	20,902	154	722,884	95	1,586	1,793
1900	27,156	113	747,601	4,207	1,660	1,511
1901	28,422	78	765,241	281	1,951	1,952
1902	27,893	119	751,496	346	1,937	761
1903	28.386	110	761,188	488	2,254	421
1904	27,450	95	782,049	601	3,364	289
1905	28,874	102	769,381	1,052	3,065	338
1906	29,166	94	755,812	524	2,724	144

Year.	Gold. *Kg.*	Iron.	Iron Ore.	Iron Sulphate.	Lead.
1897	1,087.1	4,892,059	4,183,536	9,064	108,880
1898	1,036.3	5,176,943	4,020,809	9,144	119,346
1899	1,016.4	5.644,614	4,295,575	10,186	116,995
1900	1,076.6	5,781,892	4,268,069	10,225	112,738
1901	1,087.1	5,315,628	3,831,670	10,239	113,939
1902	1,138.0	5,633,089	3,362,887	11,214	127,283
1903	949.5	6,614,768	3,786,743	11,086	133,405
1904	1,081.9	6,573,507	3,757,651	12,524	128,294
1905	1,034.9	7,106,975	4,130,210	12,075	143,270
1906	750.2	8,154,880	4,713,928	12,473	140,690

Year.	Lead Ore.	Litharge.	Manganese Ore.	Nickel.	Nickel Ore.	Nickel Sulphate.
1897	133,158	1,999	45,254	898	204	167
1898	133,637	2,360	42,232	1,108	79	127
1899	128,942	2,482	60,379	1,115	91	123
1900	133,483	2,366	58,016	1,376	3,896	115
1901	139,285	2,885	55,866	1,660	9,922	120
1902	152,282	2,516	48,882	1,605	11,816	159
1903	151,746	2,710	47,110	1,945	14,058	173
1904	150,328	2,517	52,092	2,333	13,518	207
1905	138,928	2,272	51,048	2,631	10,432	220
1906	127,322	2,744	51,881	2,648	7,472	187

Year.	Ocher and Mineral Paints.	Petroleum.	Potassium Salts.		Pyrites.	Quick-silver. *Kg.*	Salt.	
			Kainite.	All Other.			Common.	Rock.
1897	2,400	2,600	716,348	640,236	121,766	4,867	274,888	310,755
1898	2,376	2,545	744,240	718,957	128,077	4,717	286,051	329,959
1899	2,770	3,405	744,657	941,055	134,564	2,611	288,588	331,943
1900	2,850	27,731	857,271	1,264,993	159,186	1,711	287,005	354,603
1901	2,800	24,098	1,068,237	1,131,703	148,457	1,713	290,869	353,557
1902	2,780	29,520	943,450	1,344,541	155,410	1,828	291,296	359,006
1903	2,850	41,733	1,118,270	1,344,038	159,234	2,145	317,475	409,199
1904	3,200	67,604	1,261,930	1,447,323	163,209	3,030	328,933	394,910
1905	3,170	57,741	1,580,530	1,734,033	174,641	2,597	328,051	436,942
1906	3,635	59,196	1,937,181	1,937,181	186,849	5,084	339,675	492,339

Year.	Silver *Kg.*	Silver and Gold Ores.	Sulphur.	Sulphuric Acid.	Tin.	Zinc.		
						Ore.	Metal.	Sulph-ate.
1897	289,960	6	2,091	484,289	912	663,739	150,739	3,583
1898	291,969	43	1,757	531,838	979	641,671	154,643	4,158
1899	293,858	7	1,419	573,733	1,461	663,763	152,987	4,864
1900	266,577	1	1,207	593,109	2,010	636,068	155,760	3,742
1901	246,286	6	772	609,041	1,443	644,504	166,223	3,369
1902	273,901	17	250	677,798	2,753	699,392	174,892	3,381
1903	255,722	13	16	724,784	3,042	679,320	182,472	3,586
1904	252,020	8	16	868,424	4,193	710,599	192,903	3,696
1905	266,072	4	14	921,219	5,196	727,104	198,179	3,506
1906	264,427	239	16	980,188	6,570	702,933	205,632	3,630

(*a*) From *Zeitschrift für das Berg, Hütten, und Salinenwesen.*

MINERAL IMPORTS OF GERMANY. (*a*)

Year.	Aluminum, Refined and Crude.	Ammonium Sulphate.	Antimony.	Antimony and Arsenic Ores.	Asbestos. Crude.
1897	....	33,113	....	....	....
1898	....	30,254	....	....	....
1899	....	28,868	....	....	....
1900	943	23,105	1,461	1,291	6,850
1901	1,090	44,408	1,494	1,098	5,500
1902	1,100	42,252	1,495	1,231	3,415
1903	1,155	35,168	2,281	1,741	5,727
1904	2,422	35,166	2,003	1,687	5,251
1905	3,252	48,005	1,680	567	7,830
1906	3,886	35,366	2,044	2,417	9,828
1907	3,913	33,522	2,496	4,913	11,096

Year.	Asphalt.	Bituminous Rock.	Barium Chloride.	Barytes. (*b*)	Borax.	Bauxite.	Calcium Carbide.	Cement.
1897	....	....	....	....	....	....	....	42,364
1898	....	....	....	....	....	....	....	53,519
1899	61,534	....	....	....	....	....	....	63,388
1900	80,765	48,986	3,062	7,282	2,403	29,383	7,703	79,303
1901	62,299	41,733	1,768	5,764	2,537	24,113	9,526	87,262
1902	88,536	36,791	2,135	5,040	2,057	26,698	11,287	52,018
1903	94,377	40,873	2,374	5,534	2,567	22,316	14,081	49,870
1904	85,049	38,812	2,428	6,742	2,603	27,849	14,840	60,188
1905	3,461	64,196	2,114	7,981	2,802	39,137	17,256	148,118
1906	15,095	118,238	2,559	17,246	3,044	43,117	22,819	233,119
1907	4,793	128,257	2,781	12,588	3,286	59,989	25,834	241,475

Year.	Chalk (*d*), Crude White.	Chrome Ore.	Coal.		Coke.
			Bituminous.	Lignitic.	
1897	....	....	6,072,029	8,111,076	435,161
1898	....	....	5,820,332	8,450,149	332,579
1899	....	....	6,220,489	8,616,751	462,577
1900	63,929	18,728	7,384,049	7,960,313	512,690
1901	29,611	18,222	6,297,389	8,108,943	400,197
1902	26,408	10,152	6,425,658	7,882,010	382,488
1903	33,362	13,919	6,766,513	7,962,123	432,819
1904	32,581	18,132	7,299,042	7,669,099	550,302
1905	(*f*)35,529	11,998	9,399,693	7,945,261	713,619
1906	18,871	17,124	9,253,711	8,430,441	565,561
1907	16,035	19,508	13,729,849	8,963,103	584,220

Year.	Peat.	Briquettes and Peat Coke.	Cobalt and Nickel Ore.	Copper.			
				Ore.	Ingots.	Bars and Sheets.	Sulphate.
1897	....	....	....	....	67,573	400	....
1898	....	....	....	....	73,291	450	....
1899	....	....	....	....	70,091	610	....
1900	19,807	137,153	13,032	10,930	83,503	906	2,369
1901	15,102	92,037	12,186	4,614	58,620	786	1,211
1902	16,696	81,854	14,630	14,630	76,050	540	2,499
1903	14,640	84,635	36,927	13,714	83,261	568	1,691
1904	9,071	125,477	14,555	7,949	110,231	719	1,735
1905	11,439	191,753	39,590	10,137	102,218	927	2,180
1906	19,428	162,650	22,557	9,941	126,071	409	1,702
1907	15,214	195,403	29,296	19,295	124,072	772	4,519

Year.	Copperas.	Cryolite.	Gold.	
			Bullion.	Specie.
1897	....	....	$23,253,269	$14,350,000
1898	....	....	41,824,783	37,779,250
1899	....	....	34,250,242	31,870,250
1900	752	1,460	24,650,818	32,784,738
1901	501	1,249	28,631,472	32,517,702
1902	807	1,332	17,300,895	15,790,824
1903	778	1,082	39,141,995	27,711,995
1904	765	1,139	52,068,157	56,034,482
1905	666	1,143	31,743,082	30,588,776
1906	621	(*k*)	41,589,580	13,950,240
1907	1,165	(*k*)	29,135,201	19,996,310

Year.	Gold, Silver and Platinum Ores.	Graphite.	Gypsum.	Iodine.	Iron.	
					Ore.	Pig.
1897	8,927	17,366	....	164	3,185,644	423,127
1898	7,841	20,269	....	216	3,516,577	384,561
1899	7,597	23,400	....	191	4,165,372	612,652
1900	9,153	22,495	7,571	236	4,107,840	726,712
1901	8,764	17,374	7,622	266	4,370,022	267,503
1902	6,585	19,392	8,177	220	3,957,403	143,040
1903	4,386	20,953	8,328	320	5,225,336	158,347
1904	5,960	23,533	9,550	272	6,061,127	178,256
1905	6,225	26,143	11,247	377	6,085,196	158,700
1906	4,819	28,175	11,062	297	7,629,730	409,083
1907	3,601	29,405	14,662	147	8,476,076	443,624

Year.	Lead.			Magnesite.	Manganese Ore.	Mineral Pigments.	Nickel.
	Ore.	Pig and Scrap.	Lead White.				
1897	....	35,092	696	....	86,911	....	1,390
1898	....	47,497	822	....	130,711	....	1,467
1899	....	55,635	703	....	196,825	....	1,391
1900	51,338	70,252	698	13,920	204,420	12,107	1,712
1901	100,196	52,886	423	8,897	222,010	9,403	1,947
1902	71,078	39,006	357	12,237	204,647	7,719	1,458
1903	67,573	52,440	442	14,958	223,709	9,888	1,507
1904	83,807	61,388	622	15,877	255,760	10,494	1,712
1905	92,667	78,528	2,488	19,459	262,311	11,473	1,955
1906	90,027	71,191	2,342	25 527	331,171	3,960	3,478
1907	137,861	75,200	3,037	30,857	393,327	2,166	2,182

Year	Ozokerite.	Petroleum Products.		Phosphorus.	Phosphate Rock.	Potassium Salts.						
		Illuminating Oil.	Lubricating Oil.			Chloride.	Cyanide.	Iodide	Nitrate.	Carbonate.	Hydroxide.	Sulphate.
1897	....	946,344	83,957	....	289,234	715	7	18	2,889	1,734	......	912
1898	....	954,646	97,028	....	270,988	422	2	16	1,895	1,486	......	999
1899	....	963,943	106,624	....	407,457	443	3	9	1,785	1,737	......	533
1900	3,457	989,361	124,505	381	320,138	484	2	10	2,047	1,522	283	856
1901	1,981	985,904	118,999	313	351,155	462	2	1,529	1,758	1,529	165	680
1902	1,585	1,006,829	125,667	350	430,043	261	3	10	1,889	2,112	42	266
1903	1,663	1,067,697	147,837	222	461,092	40	3	8	2,163	1,850	52	81
1904	1,300	1,076,324	142,929	220	508,634	47	2	10	2,349	1,955	61	121
1905	1,114	1,070,252	143,926	198	501,048	223	3	30	2,156	1,693	24	131
1906	1,303	984,134	180,989	208	531,195	181	3	18	1,918	2,099	44	257
1907	1,653	1,021,358	226,609	165	579,505	120	1	8	1,815	2,304	89	141

Year.	Pumice-stone.	Pyrites.	Quick-silver.	Salt.	Silica, Sand, Marl, Etc.	Silver.		
						Bullion. *Kg.*	Specie. *Kg.*	Slag and Slag Wool.
1897	....	356,869	(*e*)	(*e*)	228,241	147,034	....	670,224
1898	....	376,817	560	21,957	239,708	104,770	....	685,118
1899	....	437,732	572	22,040	279,089	89,930	....	892,764
1900	2,154	457,679	555	21,738	386,028	167,432	36,857	974,947
1901	2,336	488,633	651	23,901	264,686	197,855	40,885	733,931
1902	2,070	482,095	648	26,404	305,235	282,774	39,936	831,282
1903	2,697	519,317	674	20,118	249,475	293,117	38,675	877,394
1904	3,000	503,503	691	18,743	303,419	338,875	35,189	846,738
1905	3,240	552,184	729	20,726	320,839	428,485	34,721	888,665
1906	5,463	579,355	698	16,997	325,940	235,429	38,776	813,388
1907	5,443	742,526	831	23,109	369,453	271,361	46,170	568,046

Year.	Slag, Basic Slag, Ground.	Sodium Salts.		
		Soda, Calcined.	Nitrate (Chile Saltpeter.)	Sulphate.
1897	110,216	916	465,493	....
1898	88,374	524	425,054	....
1899	68,305	515	526,944	....
1900	103,481	373	484,544	9,450
1901	87,152	178	529,568	7,921
1902	103,107	121	467,024	7,308
1903	132,337	114	467,130	6,058
1904	150,836	179	506,172	9,598
1905	198,763	143	540,916	4,752
1906	193,895	189	593,218	7,405
1907	164,364	257	591,131	10,446

Year.	Strontianite.	Sulphur.	Sulphuric Acid	Superphosphate.
1897	....	25,305	....	....
1898	....	30,269	....	....
1899	....	31,196	....	....
1900	8,701	40,689	20,634	72,062
1901	19,739	32,750	18,502	107,365
1902	34,035	32,798	22,205	109,374
1903	24,183	41,545	13,418	82,740
1904	18,055	41,030	16,087	91,288
1905	13,720	39,989	33,837	109,666
1906	5,212	41,390	74,536	76,384
1907	5,595	44,700	59,753	62,877

Year.	Tin.	Zinc.			
	Crude.	Ore.	Spelter.	Drawn or Rolled.	Zinc-white Zinc-gray Lithophon.
1897	12,395	24,735	19,734	130	3,532
1898	14,623	48,050	24,116	53	3,653
1899	12,253	57,880	23,691	95	4,226
1900	12,454	68,982	24,263	145	4,884
1901	12,910	75,533	21,250	306	3,673
1902	13,760	61,407	25,946	134	3,986
1903	13,925	67,156	25,749	237	4,667
1904	14,352	93,515	26,389	151	6,461
1905	13,501	126,577	29,583	54	7,802
1906	14,098	178,953	39,314	97	9,140
1907	12,814	184,703	28,459	134	10,189

MINERAL EXPORTS OF GERMANY. (*a*)

Year.	Aluminum, Refined and Crude,	Aluminum, Nickel Wares, etc.	Aluminum Sulphate.	Ammonium.		Antimony and Arsenic Ores.	Antimony.	
				Carbonate and Chloride.	Sulphate.		Metallic.	Salts.
1897	....	1,899	....	....	2,623	....	....	....
1898	....	2,045	....	....	4,083	....	....	....
1899	....	2,312	....	....	1,553	....	....	....
1900	269	2,398	29,372	3,196	2,431	284	131	786
1901	282	2,270	31,171	3,196	9,842	283	76	826
1902	410	2,608	34,005	3,351	5,744	410	105	954
1903	353	2,865	28,513	2,778	5,592	427	83	873
1904	407	3,077	29,311	3,106	10,696	486	250	964
1905	1,192	3,476	34,776	3,579	27,589	287	218	1,097
1906	1,111	1,321	25,937	3,555	37,288	548	221	997
1907	1,119	1,142	24,759	3,118	57,439	930	255	1,168

Year.	Arsenic.		Asbestos.	Barytes. (b)	Barium.		Bauxite.	Borax.
	Metallic.	White, etc.	Crude.		Chloride and Salts of.	White.		
1897	....	....	....	....	....	....	....	....
1898	....	....	....	....	....	....	....	....
1899	....	....	....	....	....	....	....	....
1900	14	1,573	496	59,012	5,927	2,717	44	2,894
1901	28	1,534	638	67,526	6,803	2,765	137	2,563
1902	46	2,036	709	56,026	7,358	2,922	32	2,836
1903	32	1,903	513	72,455	8,417	3,187	19	2,779
1904	50	1,956	738	69,564	8,596	3,777	21	2,741
1905	40	1,753	1,173	81,134	9,550	4,382	6	2,720
1906	....	2,282	1,938	90,819	6,541	10,721	398	2,795
1907	....	1,733	1,707	111,209	4,189	8,454	517	3,533

Year.	Bromine.	Bromine Salts.	Calcium.		Cement.	Chalk. (d) Crude White.	Chromium.	
			Carbide.	Chloride.			Ore.	Alum.
1897	....	....	....	....	524,557	....	....	....
1898	....	....	....	....	551,744	....	....	....
1899	....	....	....	....	580,255	....	....	....
1900	191	255	224	1,315	600,386	11,860	427	1,192
1901	228	249	275	888	560,612	14,134	581	1,299
1902	153	357	126	1,346	699,378	8,475	846	1,758
1903	155	435	335	1,831	742,381	12,211	37	1,921
1904	208	411	608	2,381	635,248	11,359	47	2,432
1905	156	634	709	2,831	675,664	13,081	43	2,507
1906	172	643	545	(l)	736,579	4,287	(h)36	2,942
1907	118	655	918	(l)	692,982	2,919	(h)149	3,110

Year.	Coal.		Coke.	Peat.	Briquets and Peat Coke.
	Bituminous.	Lignitic.			
1897	12,389,907	19,112	2,161,886	....	....
1898	13,989,223	22,155	2,133,179	....	....
1899	13,943,174	20,925	2,137,985	....	....
1900	15,275,805	52,795	2,229,188	8,849	550,222
1901	15,266,267	21,718	2,096,931	11,588	529,765
1902	16,101,141	21,766	2,182,383	13,410	697,799
1903	17,389,934	22,499	2,523,351	16,986	895,145
1904	17,996,726	22,135	2,716,855	14,830	917,526
1905	18,156,998	20,118	2,761,080	16,009	936,694
1906	19,550,964	18,759	3,415,347	15,680	1,095,029
1907	20,056,503	22,065	3,792,580	25,746	1,260,135

Year.	Cobalt and Nickel Ores.	Copper.				
		Ore.	Bars, Sheets and Wire.	Ingots.	Sulphate.	Copperas.
1897	....	....	....	7,183	....	....
1898	....	....	....	6,972	....	....
1899	....	....	....	7,061	....	....
1900	186	25,686	9,787	5,505	1,881	3,829
1901	96	26,678	7,700	5,097	1,942	4,125
1902	3	17,031	10,599	4,678	1,366	4,360
1903	1	15,986	10,715	4,333	1,880	3,986
1904	83	19,235	12,594	4,223	2,231	3,514
1905	107	28,908	10,006	5,958	2,180	4,495
1906	(i)	6,414	10,728	7,241	3,018	4,712
1907	(i)	20,950	13,411	6,113	2,016	6.212

Year.	Cryolite.	Fluor-spar.	Gold.	
			Bullion.	Specie.
1897	....	....	$21,472,940	$ 7,150,000
1898	....	....	3,223,732	52,061,000
1899	....	....	3,223,068	30,548,500
1900	315	12,749	3,712,841	24,562,500
1901	367	13,436	5,775,668	6,848,000
1902	486	14,177	14,171,529	11,610,000
1903	349	13,028	15,273,353	6,854,750
1904	310	13,540	7,434,958	8,491,250
1905	286	15,019	3,884,901	13,996,541
1906	(*k*)	15,493	7,981,103	9,562,279
1907	(*k*)	16,624	12,392,864	45,181,425

Year.	Graphite.	Gypsum.	Iodine.	Iron.		
				Ore.	Pig.	Oxide.
1897	2,422	....	26	3,230,391	90,885	....
1898	2,936	....	26	2,933,734	187,375	....
1899	2,703	....	26	3,119,878	182,091	....
1900	2,068	39,933	29	3,247,888	129,409	1,052
1901	1,667	40,397	27	2,389,870	150,448	1,549
1902	1,691	42,859	24	2,868,068	374,256	1,755
1903	1,810	51,874	29	3,343,510	418,072	2,006
1904	1,815	55,043	30	3,440,846	225,897	2,093
1905	1,971	52,886	27	3,698,563	380,824	2,188
1906	2,013	63,516	46	3,851,791	479,772	2,097
1907	2,176	70,737	44	3,904,400	275,170	1,066

Year.	Lead.					Manufac-tures.	Lime, Chloride of	Magnesite.
	Ore.	Pig and Scrap.	Litharge.	White.	Red.			
1897	....	24,075	....	14,786	....	....	....	....
1898	....	24,867	....	16,473	....	....	....	....
1899	....	24,491	....	16,360	....	....	....	....
1900	1,309	18,825	3,577	15,126	6,603	14,594	25,954	2,392
1901	891	20,820	4,876	16,966	7,776	13,496	32,705	2,485
1902	2,024	23,100	4,072	19,070	8,372	13,764	29,694	2,955
1903	1,270	30,243	5,175	20,765	7,617	14.955	28,849	2,812
1904	1,312	23,169	5,410	16,638	7,544	14,569	30,078	1,917
1905	1,496	32,515	4,466	16,478	8,902	17,676	30,667	2,552
1906	1,915	27,067	2,493	14,022	9,450	11,026	29,485	2,843
1907	1,296	27,708	4,470	13,651	9,371	9,371	24,946	3,264

Year.	Magnesium Chloride.	Manganese Ore.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum Products.(*f*)		Phosphor-us.
						Illumin-ating Oil.	Lubrica-ting Oil.	
1897	....	8,615	....	169	....	....	....	....
1898	....	4,810	....	203	....	....	....	....
1899	....	7,040	....	295	....	....	....	....
1900	13,375	2,454	13,958	268	1,592	843	1,455	170
1901	16,102	5,584	12,671	390	1,700	655	963	149
1902	14,757	4,528	14,392	689	1,856	824	1,177	260
1903	17,008	11,138	15,161	700	2.027	701	1,975	286
1904	16,706	5,536	16,395	1,203	2,447	760	1,763	236
1905	21,673	4,116	17,603	1,034	2,757	7,286	1,746	228
1906	26,708	2,555	4,290	954	509	673	9,982	228
1907	29,566	3,490	4,097	931	692	701	10,552	165

Year.	Phosphate Rock.	Potassium Salts.						Potassium and Potassium-Magnesium Sulphate.
		Carbonate.	Cyanide.	Chloride.	Hydroxide.	Iodide.	Nitrate.	
1897	4,000	13,100	1,086	80,389	....	124	8,986	....
1898	5,100	13,456	1,907	96,236	....	135	10,969	....
1899	2,504	11,917	1,645	101,045	....	145	15,146	....
1900	1,123	15,761	1,338	114,469	15,379	138	14,744	38,125
1901	2,260	15,567	2,089	118,959	14,892	145	13,439	37,216
1902	1,103	14,041	3,257	106,925	13,804	152	9,734	40,487
1903	4,342	13,121	2,017	125,302	13,006	154	9,671	56,455
1904	3,222	10,777	3,290	140,765	24,963	174	10,405	64,400
1905	3,720	11,963	4,005	156,440	22,246	170	12,140	67,286
1906	5,484	12,543	5,049	171,994	21,772	168	11,564	54,557
1907	1,494	13,314	5,210	173,747	20,254	146	12,668	128,344

Year.	Pumice Stone.	Pyrites.	Quick-Silver.	Salt.	Silica, Sand, Marl, etc.	Silver.		Slag and Slag Wool.
						Bullion. *Kg.*	Specie. *Kg.*	
1897	....	15,387	(*e*)	....	652,248	371,086	27,687	27,723
1898	....	19,220	97	225,548	910,354	348,733	46,932	29,931
1899	....	16,985	23	241,036	872,292	294,039	....	25,565
1900	561	24,936	23	236,291	822,840	284,853	31,799	32,494
1901	699	23,680	27	286,424	832,335	328.723	31,915	27,269
1902	691	35,370	109	328,324	713.568	372,390	24,049	22,726
1903	794	32,611	62	399,184	783,210	275,259	46,008	14,674
1904	943	30,666	43	347,251	980,673	282,017	43,986	38,587
1905	939	35,195	48	284,203	1,272,339	428,298	30,097	28,032
1906	1,578	35,829	21	297,878	1,018,547	232,273	27,816	49,912
1907	2,590	24,183	26	292,288	1,211,516	254,580	21,576	46,680

Year.	Slag. Basic.	Sodium Salts.						Sodium and Potassium Salts.	
		Bicarbonate.	Carbonate.	Hydroxide.	Nitrate. (Chile Saltpeter)	Soda, Calcined.	Sulphate and Sulphite.	Chromates.	Sulphides
1897	169,336	....	....	....	11,364	45,672	....	....	....
1898	187,598	....	....	....	12,884	37,106	....	....	....
1899	199,382	....	....	....	13,910	40,566	....	....	....
1900	174,563	1,314	1,392	1,913	14,159	44,316	41,572	3,741	2,461
1901	202,738	1,086	1,382	4,926	13,481	45,967	45,462	2,791	2,763
1902	162,062	954	2,449	5,650	14,737	33,109	56,748	2,656	4,565
1903	216,191	1,016	2,982	5,886	17,583	46,086	47,660	2,977	5,845
1904	258,767	1,524	3,050	5,084	21,075	43,590	45,506	2,272	5,489
1905	270,905	1,881	4,113	5,925	20,531	46,768	54,377	2,133	6,569
1906	354,116	2,120	5,860	6,101	22,099	41,598	64,217	2,877	6,730
1907	399,194	1,764	7,462	7,462	22,715	36,802	68,907	3,016	8,103

Year.	Stassfurt Salts.	Strontium.	
		Carbonate.	Salts.
1897	337,577	....	....
1898	370,829	....	....
1899	367,828	....	....
1900	468,277	74	496
1901	592,347	384	1,022
1902	499,220	762	1,546
1903	501,385	819	1,389
1904	631,762	613	1,207
1905	852,454	613	1,386
1906	831,293	1,726	1,578
1907	839,889	1,462	1,671

Year.	Sulphur.	Sulphuric Acid.	Super-Phosphate.	Tin. Crude.	Zinc.				Zinc. White Zinc-Gray, and Lithophon.
					Ore.	Spelter and Scrap.	Drawn or Rolled.	Sulphate.	
1897	....	....	....	861	30,047	51,341	17,453	....	17,631
1898	....	....	....	874	30,408	51,324	14,477	....	18,674
1899	....	....	....	1,121	25,192	46,334	18,281	....	19,489
1900	1,146	37,738	77,118	1,626	34,941	51,899	16,709	382	20,729
1901	621	42,853	79,190	1,683	41,002	54,490	16,517	324	24,201
1902	576	47,666	77,818	2,271	46,965	70,292	17,015	330	28,400
1903	1,052	50,109	99,672	2,581	40,458	67,057	15,715	264	27,527
1904	1,418	52,696	129,925	2,965	40,488	70,063	17,917	332	26,898
1905	1,198	48,701	115,886	3,259	38,972	67,675	18,982	296	27,877
1906	1,582	52,720	104,713	4,845	42,546	69,142	17,794	426	26,296
1907	1,501	49,950	115,049	4,244	34,863	62,238	21,484	425	30,453

(*a*) From *Statistisches Jahrbuch für das Deutsches Reich.* (*b*) Includes witherite. (*d*) Includes precipitated chalk. (*e*) Not reported. (*f*) Of domestic production only. (*h*) Includes nickel ore. (*i*) Included under chromium ore. (*k*) Included under bauxite (*l*) Included under magnesium chloride.

GREECE.

The statistics of mineral production in Greece, according to the latest available reports, are summarized in the following table:

MINERAL PRODUCTION OF GREECE. (*a*)
(In metric tons or dollars; 1 drachma=20 cents.)

Year.	Chrome Ore.	Emery.	Gypsum.	Iron Ore.	Iron Ore. Manganiferous.	Lead. Soft.	Lead Ore. Argentiferous.	Lead. Argentiferous.	Lead. Fume.	Lignite.
1896.......	1,600	3,650	120	225,600	166,850	480	3,200	14,700	1,550	14,000
1897.......	563	3,024	51	260,828	182,850	520	2,815	15,946	2,785	20,018
1898.......	1,367	3,932	83	287,100	213,938	305	(*b*)	18,888	2,655	17,310
1899.......	4,386	4,360	81	331,030	294,320	291	(*b*)	18,768	2,584	12,150
1900.......	5,600	6,328	129	279,880	243,920	245	878	16,150	2,045	12,940
1901.......	4,580	5 691	671	278,640	196,152	(*b*)	(*b*)	17,644	5 292	9,726
1902.......	11 680	4,727	10	364 340	170 040	(*b*)	430	14 048	1 647	6,500
1903.......	8 478	5 586	94	531 804	152,740	(*b*)	(*b*)	12,361	(*b*)	8,687
1904.......	6,530	6,182	393	422,159	108,319	(*b*)	(*b*)	15,186	(*b*)	13,500
1905.......	8,900	6,972	185	465,622	89,687	(*b*)	(*b*)	13.729	(*b*)	11,757
1906.......	11,530	7,565	70	680,620	96,382	(*b*)	(*b*)	12,308	(*b*)	11,582

Year.	Magnesite.			Manganese Ore.	Millstones. *Number.*	Puzzolan.	Sea Salt.	Sulphur.	Zinc Ore.	
	Crude.	Bricks.	Calcined.						Blende.	Calamine, Calcined.
1896...	11,600	892	1,514	15,500	6,757	31,300	22,800	1,540	1,750	20,950
1897...	11,311	826	686	11,868	6,975	42,600	20,421	358	3,118	22,817
1898...	14,829	516	129	14,097	18,500	70,700	25,250	135	1,139	30,906
1899...	17,184	542	3,087	17,600	12,563	46,375	37,125	1,150	1,137	21,770
1900...	17,277	534	807	8,050	13,386	49,426	22,411	891	(*b*)	18,751
1901...	13,410	500	2,009	14,166	16,400	80,223	23,079	3,212	454	17,764
1902...	27,103	935	4,730	14,960	13,564	45,400	25,200	1,391	(*b*)	18,670
1903...	25,657	(*b*)	(*b*)	9,340	11,000	(*c*)11,728	26,000	1,266	(*b*)	12,350
1904...	44,828	(*b*)	(*b*)	8,549	12,744	(*c*)18,888	27,000	1,225	(*b*)	19,913
1905...	43,498	(*b*)	(*b*)	8,171	13,102	(*b*)	25,201	1,126	(*b*)	22,562
1906...	64,424	(*b*)	(*b*)	10,040	12,732	(*b*)	25,167	(*b*)	(*b*)	26,258

(*a*) Statistics up to 1903 communicated by E. Grohmann, Seriphos. (*b*) Not reported. (*c*) Exports.

INDIA.

The official statistics of mineral production in British India are summarized in the subjoined table:

MINERAL PRODUCTION OF INDIA. (*a*)
(In metric tons or dollars; £1 = $5.)

Year.	Amber.	Coal.	Chromite	Corundum.	Gold. (*c*)	Graphite.	Iron Ore.	Jade. (*e*)	Magnesite.
1896...	(*b*)	3,909,764	.....	(*b*)	$7,085,432	(*b*)	50,559	215	(*b*)
1897...	(*b*)	4,128,330	.....	(*b*)	8,041,055	(*b*)	61,697	219	(*b*)
1898...	$5,080	4,681,927	.....	133	7,798,709	(*b*)	(*d*)42,524	196	(*b*)
1899...	755	5,174,752	.....	40	8,357,087	1,548	(*d*)52,832	228	(*b*)
1900...	515	6,222,591	.....	63	9,205,518	1,859	(*d*)57,912	142	(*b*)
1901...	55	6,741,899	.....	74	9,394,723	2,530	(*d*)58,725	206	(*b*)
1902...	2,160	7,543,272	.....	26	9,611,985	4,648	(*d*)77,273	174	3,597
1903...	2,070	7,557,400	260	(*b*)	11,203,926	3,448	(*d*)62,337	99	838
1904...	4,190	8,348,561	3,654	(*b*)	11,513,340	2,955	72,757	130	1,193
1905...	4,725	8,552,422	2,751	(*b*)	11,760,957	2,361	104,174	106	2,645
1906...	3,545	9,939 782	4,445	(*b*)	10,852,546	2,642	75,295	116	1,861
1907...	1,925	11,325,696	7,390	(*b*)	10,251,494	2,472	68,737	249	189

Year.	Manganese Ore.	Mica. (*e*)	Petroleum. *Gallons.*	Rubies	Salt.	Saltpeter (Potassium nitrate.)	Tin Ore.
1896..........	57,782	452	15,057,094	$171,884	1,043,171	21,425	82
1897..........	74,862	652	19,128,828	200,613	937,932	26,845	62
1898..........	61,419	527	22,234,438	289,750	1,043,862	21,224	40
1899..........	88,524	497	32,934,007	454,240	977,269	18,555	64
1900..........	129,865	1,025	37,729,211	486,630	1,071,877	20,189	94
1901..........	122,831	1,505	50,075,117	522,380	1,208,933	17,711	63
1902..........	160,311	806	56,607,688	434,475	1,116,797	17,320	91
1903..........	174,563	1,002	87,859,069	444,095	1,021,581	18,711	100
1904..........	152,708	828	118,491,382	453,060	1,319,535	14,200	63
1905..........	257,969	1,172	144,798,444	....	1,426,066	15,745	68
1906..........	503,684	2,458	140,553,122	465,115	1,371,172	16,822	87
1907..........	(*e*) 557,194	1,977	152,045,677	577,325	1,120,236	......	77

(*a*) Records of the Geological Survey of India. (*b*) Not reported. (*c*) £1 = $4.866. (*d*) Production of iron ore in Bengal only. (*e*) Exports.

ITALY

The following tables itemize the statistics of the production and the foreign commerce of mineral and metallurgical products in Italy:

MINERAL PRODUCTION AND REFINED PRODUCTS OF ITALY. (a)
(In metric tons or dollars; 5 lire=$1.)

Year.	Alum.	Aluminum Sulphate.	Alunite.	Antimony.	Antimony Ore.	Asphalt, Mastic and Bitumen.	Asphaltic Rock.	Barytes.
1895	995	2,950	7,000	423	2,241	14,491	46,713	(b)
1896	850	2,390	6,000	538	5,086	12,490	45,456	(b)
1897	1,030	2,310	6,500	404	2,150	18,644	55,339	(b)
1898	1,165	2,915	7,000	380	1,931	17,813	93,750	12,400
1899	945	2,330	5,800	581	3,791	41,732	81,987	12,545
1900	1,097	2,403	5,200	1,174	7,609	33,127	101,738	14,003
1901	1,075	2,260	4,900	1,721	8,818	31,814	104,111	13,245
1902	....	....	8,200	....	6,116	33,684	64,245	....
1903	....	....	8,100	905	6,927	35,757	89,078	....
1904	2,490	2,210	8,000	836	5,712	34,227	111,390	250
1905	2,975	2,740	8,500	327	5,083	26,838	106,586	590
1906	2,878	2,800	7,500	537	5,704	34,386	130,225	800

Year.	Borax. Refined.	Boric Acid.		Coal. (c)	Coal. (Briquettes).	Coke	Copper.		
		Crude.	Refined.				Ore.	Ingot, etc.	Sulphate
1895	944	2,633	253	305,321	451,470	394,043	83,670	2,375	....
1896	943	2,616	253	276,197	422,409	426,906	90,408	2,842	4,756
1897	990	2,704	260	314,222	549,050	430,617	93,377	2,980	5,337
1898	702	2,650	166	341,327	594,500	469,228	95,128	3,230	6,364
1899	709	2,674	129	388,534	566,000	485,951	94,764	3,032	7,795
1900	858	2,491	283	479,896	703,740	487,831	95,644	2,797	13,191
1901	544	2,558	347	425,614	754,800	490,803	107,750	3,097	15,374
1902	....	2,763	....	414,569	....	....	101,142	....	14,601
1903	....	2,583	....	346,887	724,993	554,559	114,823	....	18,164
1904	569	2,624	314	362,151	903,610	607,297	157,503	2,313	17,237
1905	1,007	2,700	749	412,916	842,250	627,984	140,035	1,175	26,212
1906	1,062	2,561	562	473,293	829,277	672,689	147,137	....	34,270

Year.	Gold.		Graphite.	Iron and Steel.				
	Ore.	Bullion.		Ore.	Pig.	Bar, Sheet, Pipe, Wire, etc.	Tin Plate.	Steel.
1895	7,099	$186,074	2,657	183,371	9,213	163,824	5,860	50,314
1896	7,659	172,552	3,148	203,966	6,987	139,991	2,918	65,955
1897	10,723	209,998	5,650	200,709	8,393	149,944	6,500	63,940
1898	9,549	124,869	6,435	190,110	12,387	167,499	7,200	87,467
1899	11,859	75,294	9,990	236,549	19,218	197,730	8,000	108,501
1900	5,840	38,212	9,720	247,278	23,990	190,518	10,000	115,887
1901	890	2,725	10,313	232,299	15,819	180,729	7,550	123,310
1902	1,215	....	9,210	240,705	....	....	....	....
1903	5,734	41,933	7,920	374,790	90,744	177,392	11,275	154,134
1904	6,746	43,063	9,765	409,460	112,598	181,385	16,655	177,086
1905	1,200	9,169	10,572	366,616	181,248	205,915	18,560	244,793
1906	6,543	47,321	10,805	384,217	180,940	236,946	16,350	332,924

Year.	Lead. Ore.	Lead. Pig.	Manganese Ore.	Manganiferous Iron Ore.	Marble.	Petroleum, Crude.	Petroleum, Benzine, etc.	Pumice-Stone.
1895	30,632	20,353	1,569	5,860	186,900	3,594	4,191	(b)
1896	33,545	20,786	1,890	10,000	209,428	2,524	2,734	(b)
1897	36,200	22,407	1,634	21,262	236,958	1,932	3,392	(b)
1898	33,930	24,543	3,002	11,150	271,725	2,015	5,040	2,766
1899	31,046	20,543	4,356	29,874	313,744	2,242	5,384	7,300
1900	35,103	23,763	6,014	26,800	310,336	1,683	6,077	7,000
1901	43,449	25,796	2,181	24,290	334,146	2,246	4,211	8,300
1902	42,330	....	2,477	23,113	....	2,633	....	....
1903	42,443	22,126	1,930	4,735	....	2,486	4,577	....
1904	42,846	23,475	2,836	2,836	390,118	3,543	6,388	11,600
1905	39,030	19,077	5,384	5,384	389,869	6,122	9,924	11,300
1906	40,945	21,268	3,060	20,500	430,202	7,452	2,322	16,366

Year.	Pyrites. (Cupriferous in part).	Quicksilver. Ore.	Quicksilver. Metal.	Salt. Brine.	Salt. Rock.	Salt. Sea.	Silver. Ore.	Silver. Bullion. *Kg.*
1895	38,586	10,504	199	10,605	18,710	448,335	870	44,189
1896	45,728	14,305	186	11,974	17,300	422,555	640	38,075
1897	58,320	20,659	192	11,725	19,801	429,253	405	45,313
1898	67,191	19,201	173	11,546	18,199	451,426	435	43,437
1899	76,538	29,322	205	11,021	18,721	363,826	540	33,645
1900	71,616	33,930	260	10,890	18,331	338,034	584	31,169
1901	89,376	38,614	278	10,690	23,054	401,443	511	32,464
1902	93,177	44,261	....	10,581	23,677	424,239	421	....
1903	101,455	55,528	312	10,962	25,911	451,633	405	24,388
1904	112,004	60,403	352	11,878	18,638	433,810	143	24,943
1905	117,667	63,378	369	12,756	19,669	405,274	170	20,215
1906	122,364	80,638	417	13,751	19,007	496,872	48	20,362

Year.	Sulphur. Crude (Fused).	Sulphur. Ground.	Sulphur. Refined.	Talc. Ground.	Zinc. Ore.	Zinc. Spelter.
1895	370,766	91,517	75,329	(b)	121,197	*Nil.*
1896	426,353	89,292	71,072	(b)	118,171	*Nil.*
1897	496,658	69,178	85,872	(b)	122,214	250
1898	502,351	146,001	99,494	12,760	132,099	250
1899	563,697	161,509	110,213	11,000	150,629	251
1900	544,119	167,466	157,957	14,415	139,679	547
1901	563,096	171,252	141,431	11,770	135,784	511
1902	510,333	....	....	....	131,965	....
1903	553,751	139,376	139,464	6,300	157,521	126
1904	527,563	189,266	163,695	6,740	148,365	189
1905	568,927	180,676	180,774	6,626	147,834	5
1906	499,814	176,476	170,990	7,894	155,751	69

(a) From *Rivista del Servizio Minerario.* (b) Not reported. (c) Includes anthracite, lignite, fossil wood and bituminous schist.

MINERAL IMPORTS OF ITALY. (a)

(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony	Arsenic. *Kg.*	Asbestos.	Asphaltum.	Barytes.	Borax and Boric acid.	Cement and Hydraulic Lime.	Chalk.
1896	38	(b)	851	11,892	549	166	12,810	15,716
1897	66	2,604	619	1,632	578	253	16,680	28,937
1898	58	700	1,186	1,150	860	147	12,029	18,252
1899	64	600	1,675	1,473	936	123	14,391	13,738
1900	37	900	1,645	1,933	859	122	15,494	18,436
1901	49	1,800	2,019	1,450	825	232	14,872	20,731
1902	80	1,200	1,536	1,020	1,170	516	13,732	15,216
1903	98	4,400	1,691	1,567	1,099	504	15,547	10,063
1904	131	3,700	2,174	2,604	1,875	271	15,260	6,891
1905	117	3,400	1,806	3,252	1,444	112	15,797	5,556
1906	50	5,300	2,171	2,854	1,400	163	18,937	7,714
1907	163	3,100	3,110	3,661	1,540	307	29,024	6,156

Year.	Clay Products.			Coal.	Copper Ore.	Copper Cement.
	Brick, Tile, etc.	Kaolin	Terra-Cotta			
1896	18,504	3,775	2,675	4,081,218	484	1,150
1897	19,086	5,719	2,167	4,259,643	1,611	1,049
1898	21,681	9,079	2,122	4,431,524	5,471	2,040
1899	22,410	12,105	2,200	4,859,556	2,777	1,328
1900	25,702	9,595	2,031	4,947,180	5,290	1,298
1901	35.534	12,809	2,482	4,838,994	11,047	1,987
1902	29,156	14,165	2,537	5,406,069	9,422	2,299
1903	24,586	11,033	2,883	5,546,823	9,459	649
1904	33,655	18,610	3,236	5,904,578	8,104	309
1905	37,762	15,315	4,080	6,437,539	6,879	486
1906	47,667	19,218	4,657	7,673,435	9,363	802
1907	86,351	16,534	4,867	8,300,439	18,023	888

Year.	Copper, Brass and Bronze.	Copper and Iron Sulphates.	Gold.			Graphite.	Iron.		
			Specie. *Kg.*	Unrefined. *Kg.*	Manufactures: *Kg.*		Ore.	Pig.	Wrought.
1896	6,955	24,255	1,004	2,517	1,515	204	594	119,491	4,820
1897	7,999	28,878	444	807	1,375	315	5,831	156,019	3,801
1898	7,433	25,560	154	507	1,844	382	8,723	169,059	4,076
1899	7,334	27,408	181	326	1,390	608	20,799	191,613	4,158
1900	9,249	32,127	188	309	1,348	982	19,205	160,686	7,405
1901	8,659	32,058	1,115	494	1,547	102	4,054	159,972	5,695
1902	10,865	25,107	8,967	479	1,269	60	4,314	155,143	6,603
1903	9,588	24,566	44,218	1,396	1,220	63	5,937	126,756	6,380
1904	15,198	37,298	8,364	1,961	1,640	52	4,390	149,130	6,740
1905	18,188	30,684	46,509	5,768	1,799	107	4,745	136,077	7,616
1906	21,458	25.060	33,960	4,571	2,633	361	6,452	168,985	13,342
1907	28,937	15,989	42,519	4,443	2,015	267	22,046	231,042	11,365

Year.	Iron and Steel Scrap.	Lead.			Lead. Oxide and Carbonate	Mineral Paints.	Nickel Alloys and Manufactures.	Petroleum.
		Ore. (*c*)	Metal and Alloys in Pigs.	Mnfres.				
1896	162,035	9,730	1,166	192	523	852	411	70,217
1897	130,938	14,854	1,178	247	580	888	432	68,973
1898	138,426	10,947	1,431	435	647	692	258	70,654
1899	245,616	7,476	3,990	249	662	958	250	71,391
1900	197,415	9,134	3,248	233	557	958	232	73,089
1901	148,305	9,063	2,926	268	815	865	476	69,298
1902	198,914	1,680	7,563	288	846	670	561	68,781
1903	206,036	689	5,398	273	768	859	525	68,220
1904	246,359	2,187	4,541	247	871	940	652	69,233
1905	276,311	465	6,764	295	686	974	574	66,493
1906	344,977	4,526	10,958	319	984	964	717	64,541
1907	362,567	4,342	9,231	243	953	1,119	725	77,658

Year.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Potassium Sulphate.	Precious Stones. Manufactures.	Quick-Silver.	Silver. *Kg.*			Slag.
						Specie.	Unrefined in Bars.	Manufactures.	
1896	(*b*)	9,841	431	$1,505,461	30	22,751	2,291	6,533	30,275
1897	(*b*)	11,012	562	2,033,714	30	26,008	2,434	5,286	37,201
1898	65,126	11,047	928	1,629,008	39	8,241	991	5,673	51,199
1899	116,283	12,370	1,297	1,323,317	62	20,605	1,782	4,881	56,549
1900	140,281	14,077	1,670	1,801,264	49	29,291	2,678	4,358	32,254
1901	142,108	14,693	1,411	2,967,041	36	35,089	4,391	4,213	7,312
1902	159,341	17,617	1,566	3,885,484	57	28,662	8,768	3,455	5,634
1903	172,328	17,528	1,353	4,866,327	28	81,373	12,541	5,893	8,849
1904	217,162	14,846	1,663	4,896,867	25	67,520	15,885	4,409	3,821
1905	240,144	17,752	1,804	6,333,376	57	51,997	20,697	5,437	72,785
1906	307,762	16,718	1,534	7,281,373	12	122,737	20,410	5,841	88,118
1907	384,896	16,237	3,866	5,506,559	11	123,914	21,829	5,072	134,639

Year.	Sodium Salts. Carbonate.	Sodium Salts. Nitrate (Crude).	Sodium Salts. Sod. and Pot. Nitrates, Refined.	Tin. Block.	Tin. Mnfres.	Zinc. Ore.	Zinc. Oxide.	Zinc. Spelter and Old.	Zinc. Mnfres.
1896.........	18,927	11,685	541	1,763	91	(*b*)	540	2,596	3,482
1897.........	20,721	16,400	917	1,520	81	(*b*)	570	3,278	3,556
1898.........	20,845	19,961	702	1,722	109	216	573	2,813	3,200
1899.........	22,654	22,385	671	1,240	96	(*b*)	804	3,498	3,221
1900.........	23,215	27,706	511	1,643	56	85	1,034	3,627	3,543
1901.........	21,956	40,498	315	1,858	91	23	813	3,991	4,079
1902.........	26,133	24,483	314	2,114	110	131	904	3,805	4,167
1903.........	24,753	43,480	638	2,288	130	46	1,416	4,551	4,461
1904.........	27,747	32,283	613	2,170	150	362	1,124	5,202	4,168
1905.........	29,066	46,517	689	2,304	103	14	1,246	5,997	4,701
1906.........	31,170	32,508	395	3,361	167	2,042	1,920	6,835	4,421
1907.........	35,538	41,457	668	2,771	183	11	1,962	8,152	5,407

MINERAL EXPORTS OF ITALY. (*a*)
(In metric tons or dollars; 5 lire=$1.)

Year.	Antimony.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.
1896..........................	361	130	13,729	66	2,719	3,871	5,593
1897	271	170	15,310	143	1,618	5,330	7,556
1898..........................	338	208	19,465	70	2,167	5,192	6,744
1899..........................	240	245	26,402	45	2,872	5,462	5,386
1900..........................	467	261	24,287	40	2,114	6,860	2,980
1901..........................	765	302	21,856	32	2,190	8,463	3,428
1902..........................	359	144	20,884	91	1,847	7,930	4,215
1903..........................	314	222	24,303	35	901	6,325	3,802
1904..........................	107	163	14,880	70	1,122	7,810	4,089
1905	132	236	23,740	162	2,255	8,445	5,007
1906..........................	208	205	27,176	147	2,777	6,774	4,194
1907..........................	115	142	26,036	152	1,330	4,474	3,118

Year.	Clay Products. Brick, Tile, etc.	Clay Products. Kaolin.	Clay Products. Porcelain.	Clay Products. Terra-Cotta.	Coal.	Copper Ore.	Brass and Bronze.	Copper, and Iron Sulphate.
1896..........	143,648	49	99	2,852	18,924	3,603	643	71
1897	125,925	93	138	2,472	23,191	2,408	641	18
1898..........	125,614	94	86	2,751	17,749	2,356	857	25
1899..........	136,402	94	124	2,796	20,803	1,148	1,734	20
1900..........	122,388	179	96	3,051	23,926	1,179	1,209	60
1901..........	108,057	368	266	3,325	25,594	9	659	20
1902..........	122,482	*Nil*	154	3,378	33,374	11	874	39
1903..........	148,407	133	192	4,482	29,219	15	952	44
1904..........	157,999	18	242	4,581	35,149	43	939	29
1905..........	151,243	86	229	5,194	38,555	77	812	249
1906..........	145,064	1,174	207	5,732	31,666	189	1,244	102
1907..........	111,822	101	160	6,377	40,769	179	1,101	835

Year.	Gold. Specie.	Gold. Unrefined. *Kg.*	Graphite.	Iron. Ore.	Iron. Pig.	Iron. Wrought.
1896............................	$1,539,460	2,517	3,727	187,059	1,378	427
1897............................	996,960	1,381	4,164	207,619	498	1,434
1898............................	1,590,920	1,739	5,145	217,556	840	699
1899............................	1,281,540	1,162	8,114	234,515	378	611
1900............................	1,453,900	2,763	7,820	170,286	329	440
1901............................	1,159,400	2,955	7,169	121,592	311	499
1902............................	1,176,140	733	7,098	209,070	395	1,054
1903............................	507,160	1,291	7,068	98,319	810	1,670
1904............................	1,011,840	1,494	7,433	2,577	229	847
1905............................	655,340	1,731	6,811	11,358	1,395	951
1906............................	744,960	1,476	4,904	1,833	254	463
1907............................	708,660	802	7,474	26,000	121	687

Year.	Lead.				Mineral Paints.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Quick-silver.	Salt.
	Ore.	Lead Alloys in Pigs.	Manu-factures.	Oxide and Carbonate.					
1896	4,731	1,419	1,441	489	2,412	(*b*)	88	155	171,740
1897	4,747	2,790	1,410	461	2,318	(*b*)	66	236	176,520
1898	4,492	5,870	1,764	414	2,884	(*b*)	85	244	126,860
1899	3,129	2,497	910	389	2,784	(*b*)	120	223	114,050
1900	3,741	5,018	1,408	367	2,977	1,726	142	259	112,900
1901	3,977	4,463	2,128	410	2,913	1,290	198	301	114,210
1902	3,354	5,650	2,258	404	2,953	894	136	215	145,190
1903	5,041	2,911	1,934	426	3,305	2,942	233	222	144,910
1904	5,524	1,954	1,887	347	3,231	2,812	162	266	130,940
1905	4,311	976	2,043	310	3,632	3,519	238	243	116,040
1906	8,356	2,005	2,201	315	4,502	1,652	304	278	126,199
1907	3,213	1,548	2,536	240	4,602	4,560	658	350	99,191

Year.	Silver. *Kg.*		Slag.	Sodium Salts.			Stone.			
	Specie.	Unrefined.		Carbon-ate.	Nitrate. (Crude.)	Sod. and Pot. Nitrates, Refined.	Ala-baster. (Crude.)	Building Stone.	Marble. (Crude.)	Marble and Alabaster.
1896....	28,377	26,854	4,753	279	51	306	289	23,580	80,750	68,639
1897....	72,605	50,503	8,847	275	151	344	269	36,229	83,081	62,750
1898....	8,241	68,607	6,861	391	79	256	457	35,945	88,404	68,150
1899....	32,085	32,432	4,898	438	136	124	714	53,904	98,485	84,104
1900....	10,501	25,310	4,222	486	58	129	489	54,051	61,650	72,619
1901....	14,446	42,325	3,261	377	116	59	474	53,668	96,631	73,599
1902....	10,978	20,427	3,615	446	346	259	727	74,036	112,967	83,172
1903....	4,377	9,486	4,929	482	781	492	605	69,473	130,316	87,079
1904....	3,834	24,165	4,458	376	363	230	800	80,337	131,087	82,911
1905....	2,371	25,947	9,844	214	424	159	935	116,110	132,765	94,295
1906....	3,465	18,262	8,990	253	80	133	942	83,001	148,579	99,776
1907....	912	18,164	10,964	200	138	102	1,195	85,045	164,525	113,713

Year.	Sulphur.	Tin.		Zinc.			
		Block.	Manufac-tures.	Ore.	Oxide.	Spelter and Scrap.	Manufac-tures.
1896........................	356,370	10	89	115,454	48	33	8
1897........................	358,932	29	109	133,125	189	309	63
1898........................	405,823	34	177	130,064	110	156	14
1899........................	424,018	69	176	140,107	123	227	21
1900........................	479,139	147	153	111,870	102	359	24
1901........................	414,018	202	187	103,020	140	349	18
1902........................	439,242	236	174	114,894	122	338	66
1903........................	461,289	173	180	116,449	116	591	51
1904........................	437,067	171	151	126,393	483	263	46
1905........................	381,128	285	107	117,810	173	434	48
1906........................	336,339	303	81	144,244	687	639	50
1907........................	297,378	434	59	142,271	727	1,182	39

(*a*) From *Statistica del Commercio speciale di Importazione e di Esportazione*. (*b*) Not reported. (*c*) Includes argentiferous lead ore.

JAPAN.

The total mineral production of the Japanese Empire, according to the latest available returns, is shown in the following table, in metric tons, unless otherwise specified:

MINERAL PRODUCTION OF JAPAN. (*a*)

Year.	Antimony.		Arsenic. *Kg.*	Coal.	Copper.	Gold. *Kg.*	Graphite	Iron. Pig.	Lead.
	Sulphate.	Metal.							
1894	1,172	403	5,387	4,300,370	19,814	806	1,091	19,474	1,577
1895	1,061	641	7,343	4,770,313	19,103	935	77	25,863	1,978
1896	828	517	6,043	5,100,005	20,114	964	215	27,420	1,958
1897	348	823	13,039	5,147,103	20,425	1,037	391	28,040	772
1898	1,005	235	7,129	6,643,047	21,023	1,159	347	23,611	1,703
1899	712	229	5,077	6,668,608	19,421	1,675	53	23,066	1,963
1900	81	349	4,669	7,370,667	24,317	2,124	94	24,841	1,878
1901	118	429	10,312	8,884,812	27,392	2,475	88	29,449	1,803
1902	88	528	12,188	9,588,910	29,034	2,975	97	32,130	1,644
1903	153	434	6,000	10,088,845	33,245	3,140	114	33,870	1,728
1904	104	321	4,000	10,723,796	33,187	2,765	216	38,143	1,803
1905	96	190	8,333	11,955,946	35,944	3,048	209	53,210	2,272
1906	97	627	5,250	13,468,529	36,963	2,873	177	57,373	4,305
1907(*d*)	(*b*)	(*b*)	4,646	13,935,952	38,001	5,090	72	43,599	3,067

Year.	Manganese Ore.	Petroleum *Gallons.*	Phosphates.	Pyrite.	Quicksilver. *Kg.*	Salt. *Hectoliters.*	Silver. *Kg.*	Sulphur.	Tin.
1894	13,368	(*c*)5,426,071	(*b*)	(*b*)	1,547	11,411,275	79,222	18,787	38.7
1895	17,142	7,118,962	(*b*)	(*b*)	481	(*b*)	74,815	15,557	48.3
1896	17,967	(*c*)7,440,206	(*b*)	(*b*)	1,762	(*b*)	64,303	12,540	50.0
1897	15,448	9,179,474	(*b*)	7,626	2,678	(*b*)	54,289	13,582	47.6
1898	11,497	11,145,457	(*b*)	8,726	1,399	11,482,422	60,436	10,321	42·7
1899	11 336	18,844,034	(*b*)	8,376	....	10,483,082	56,161	10,237	18.5
1900	15,831	30,470,068	(*b*)	16,166	270	11,890,361	58,799	14,439	12.3
1901	16,270	39,056,820	(*b*)	17,589	750	12,463,771	54,739	16,548	14.1
1902	10,844	34,850,129	196	18,580	1,418	11,042,192	57,635	18,287	18.6
1903	5,616	50,724,174	191	16,149	206	6,574,890	58,704	22,914	19.0
1904	4,324	51,573,754	13	24,886	*Nil*	7,019,650	61,339	25,587	25.0
1905	14,017	47,132,800	1,519	25,569	349	(*b*)	82,886	24,652	26.0
1906	54,339	60,005,957	3,037	36,038	336	(*b*)	76,247	27,589	77.C
1907(*d*)	10,410	(*e*)70,395,921	(*b*)	36,124	547	(*b*)	88,151	28,381	30.0

(*a*) From *Résumé Statistique de l' Empire du Japon*, Tokio. (*b*) Not reported. (*c*) Crude petroleum. (*d*) Furnished by *Japan Financial and Economic Monthly.* (*e*) Estimated.

MEXICO.

Owing to the incompleteness of the Mexican statistics of production, we are unable to give any satisfactory table. Exports may, however, be taken as indicating the condition of the mining industry.

MINERAL EXPORTS OF MEXICO. (*a*)
(In metric tons or Mexican dollars.)

Year.	Antimony Ore.	Coal.	Copper.		Gold.				
			Ore.	Ingot.	Ore.	Bullion.	Specie.	Cyanide.	Sulphide.
1895	600	61,686	3,006	20,429	$ 103,773	$ 4,920,504	$175,098	$31,231	$3,026
1896	3,231	75,541	144	20,659	206,874	5,533,789	261,078	161,784	44,890
1897	5,873	105,298	1,094	16,858	365,226	6,220,765	202,223	226,986	33,916
1898	5,932	118,553	13,146	10,362	1,037,202	6,493,735	(*b*)	294,730	64,061
1899	10,382	113,192	223	25,293	335,849	7,017,286	183,474	115,961	266,782
1900	2,313	38,676	408	27,970	306,392	7,435,864	192,456	128,675	177,193
1901	5,103	17,281	5,576	33,818	284,722	8,324,681	210,431	178,803	81,744
1902	1,280	3,406	6,101	63,609	303,979	9,079,371	129,899	78,295	40,658
1903	7,302	1,840	10,912	51,716	264,503	9,693,692	54,636	85,465	124,020
1904 (*c*)	(*d*)1,775	125	48,365	57,338	537,290	10,867,272	(*b*)	79,129	176,090
1905	(*e*)3,034	497	92,540	56,634	1,103,237	23,026,547	55,025	288,623	79,388

Year.	Graphite.	Gypsum.	Lead.		Silver.					
			Ore.	Base Bullion.	Ore.	Bullion.	Specie.	Sulphide.	Cyanide.	Slag.
1895.	794	1,340	568	50,122	$10,977,079	$22,178,294	$18,300,553	$555,475	$14,649	$72,590
1896.	795	2,050	167	48,663	9,971,053	28,565,843	18,737,331	1,495,306	38,049	64,121
1897.	759	2,095	2	60,029	11,401,176	35,775,125	21,925,347	1,663,581	123,246	39,800
1898.	(*b*)	1,650	(*b*)	60,918	11,048,358	37,137,599	(*b*)	1,663,501	257,342	46,488
1899.	2,305	1,050	1	67,441	10,766,099	37,585,911	5,580,834	1,929,085	76,942	4,810
1900.	2,561	1,600	468	74,944	12,495,524	41,468,745	22,679,655	1,893,646	67,607	87,883
1901.	762	800	(*b*)	79,097	9,615,939	36,348,374	12,038,158	2,141,685	259,282	93,549
1902.	1,434	(*b*)	118	107,366	4,108,088	45,796,576	17,753,526	1,978,919	108,344	132,093
1903.	1,404	(*b*)	11	100,532	11,781,048	48,276,797	16,167,673	1,642,627	135,561	289,900
1904.	970	(*b*)	1	95,010	11,000,869	45,430,020	....	1,392,356	171,452	202,594
1905.	970	27	1	101,196	8,505,834	63,564,787	20,335,297	736,228	438,094	29,012

(*a*) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. (*b*) Not reported. (*c*) Figures for 1904 were from *Anuario Estadistico de la Republica Mexicana* for 1904. (*d*) Includes 136 metric tons of ore. (*e*) Includes 57 metric tons of ore.

NORWAY.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (*a*)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Apatite (*b*)	Chrome Ore.	Copper.		Feldspar.	Gold.	Iron.		
			Ore.	Ingot.			Ore.	Pig and Cast.	Bars and Steel.
1896......	1,106	....	29,910	1,067	12,223	$9,450	2,000	335	400
1897	872	....	27,606	1,064	17,392	675	3,627	417	452
1898......	3,593	....	37,047	941	11,355	1,539	4,425	231	379
1899	1,500	41	43,358	1,209	19,260	2,700	4,576	406	666
1900......	300	165	46,858	1,280	17,609	2,430	17,925	444	614
1901......	738	85	40,726	1,073	18,323	2,700	42,252	261	376
1902......	2,295	22	40,499	1,347	19,591	36,990	53,675	527	461
1903......	1,795	*Nil*	35,417	1,382	18,590	8,370	53,475	509	442
1904......	1,456	154	36,891	1,342	20,835	*Nil*	45,328	350	395
1905......	2.522	*Nil*	37,045	1,153	22,508	*Nil*	46,582	474	253
1906.....	3,482	*Nil.*	32,203	1,371	23,896	5.400	109,259	257	317

Year.	Molybdenite	Nickel.		Pyrites, Iron and Copper.	Rutile.	Silver.		Zinc Ore (*d*)
		Ore.	Metal.			Ore and NativeSilver.	Metal. *Kg.*	
1896......	......	*Nil*	16	60,507	30	527	4,664	450
1897......	......	*Nil*	*Nil*	94,484	32	642	5,372	908
1898......	......	*Nil*	*Nil*	89,763	35	497	4,802	320
1899......	......	220	5	95,636	30	429	4,600	379
1900......	......	1,888	13	98,945	40	475	4,600	204
1901......	......	2,018	40	101,894	55	519	5,680	90
1902......	20	4,040	60	121,247	*Nil*	471	6,220	30
1903......	31	5,670	75	129,939	25	481	7,269	335
1904......	30	5,352	73	133,603	25	1,297	8,064	42
1905......	46	5.477	77	162,012	35	1,570	7,100	4,241
1906......	1,026	6,081	81	197,886	55	1,570	7,100	3,308

(*a*) *Tabeller vedkommendo Norges Bergværkdsdrift, Statistik Aarbog for Kongeriket Norge.* (*b*) Exports which represent production. (*d*) Includes lead ore.

MINERAL IMPORTS OF NORWAY. (*a*)
(In metric tons.)

Year.	Borax. *Kg.*	Cement and Hydraulic Lime.	Coke, Coal and Cinders. *Hectoliters.*	Copper and Brass.		Glass and Glassware.
				Plates and Bars.	Wares.	
1896............................	38,305	16,028	....	1,074	479	3,729
1897............................	44,495	18,734	15,374,572	1,140	591	4,262
1898............................	71,590	25,403	15,409,902	1,064	807	3,905
1899............................	62,060	33,652	18,475,996	1,000	1,120	3,229
1900............................	71,124	24,511	19,002,026	696	1,164	2,874
1901............................	68,000	20,993	17,665,349	1,018	761	1,793
1902............................	(c)	18,984	19,338.615	1,118	(c)	(c)
1903............................	(c)	17,906	20,086,974	899	309	690
1904............................	54,953	12,845	21.049.128	688	866	1,158
1905............................	(c)	13,797	20,973,608	882	1,146	1,106
1906............................	(*c*)	11,676	21,478,000	787	783	983

Year.	Iron and Steel.							Other Mnfres.
	Pig.	Bars, Hoops, etc. Wrought Iron.	Anchors, Cables and Chains.	Rails.	Nails and Spikes.	Steel.	Sheets and Plates.	
1896.......	......	26,552	1,090	4,315	1,760	2,754	17,930	6,831
1897.......	21,606	29,038	1,367	7,637	2,097	4,350	23,350	10,695
1898.......	23,106	26,203	1,485	10,327	2,087	2,428	26,894	17,182
1899.......	21,445	25,379	1,394	8,137	1,529	2,652	32,192	21,400
1900.......	20,844	23,010	1,203	11,952	1,219	2,085	29,318	17,493
1901.......	19,112	20,672	1,708	22,959	1,808	1,905	31,184	18,372
1902.......	18,969	26,685	2,103	15,316	2,205	1,754	36,288	22,069
1903.......	20,652	21,977	1,807	4,631	1,261	1,958	42,098	18,855
1904.......	18,891	24,094	2,109	5,814	1,071	1,610	42,013	5,462
1905.......	20,828	27,740	2,224	6,566	1,222	1,436	42,203	44,414
1906.......	20,197	26,015	2,585	8,086	1,012	2,018	48,969	45,959

Year.	Lead in Pigs and Sheets.	Lead, White and Zinc Oxide.	Petroleum and Paraffin.	Potash.	Salt.	Salt-peter.	Soda.	Sulphur. (b)	Tin in Blocks, etc.	Zinc in Bars, Plates, etc.
1896...	653	1,192	35,823	945	117,920	308	5,156	9,347	142	1,101
1897...	848	1,119	39,810	919	164,572	277	5,492	10,701	236	1,102
1898...	732	1,491	36,504	754	127,341	477	4,823	9,589	257	1,370
1899...	869	1,296	42,182	802	134,583	278	4,555	10,734	546	1,509
1900...	670	1,216	39,657	638	143,365	356	4,576	14,827	149	1,254
1901...	590	1,321	47,011	518	127,607	208	5,220	11,149	141	1,027
1902...	(c)	(c)	(c)	(c)	141,415	315	(c)	(c)	(c)	1,104
1903...	311	(c)	58,822	457	143,110	245	4,200	8,829	106	1,015
1904...	498	1,898	50,543	477	153,699	321	3,197	12,181	176	940
1905...	448	1,309	43,860	393	137,800	1,048	3,704	10,240	134	967
1906...	459	1,149	41,546	396	167,300	776	4,334	11,465	135	1,087

MINERAL EXPORTS OF NORWAY. (a)

(In metric tons.)

Year.	Apatite.	Clay Products.		Copper.			Feldspar.
		Bricks. *Thousands.*	Earthen-ware.	Ore.	Ingot.	Scrap.	
1896	1,160	10,008	365	30,367	1,276	712	12,223
1897....................	872	11,711	260	15,111	1,222	670	17,392
1898	3,593	15,534	2	13,587	1,650	1,206	11,355
1899	1,500	11,949	2	7,198	1,785	1,038	19,260
1900	300	5,266	2	5,756	1,891	1,168	17,609
1901	738	12,103	7	6,041	1,465	774	(d)18,423
1902	2,295	(c)	....	4,848	1,913	(c)	(d)19,611
1903	1,795	37,972	5	3,448	1,930	888	(d)18,640
1904....................	1,456	29,706	11	2,673	1,124	785	20,835
1905....................	2,522	29,861	....	3,393	958	968	20,696
1906....................	3,482	22,831	....	84	875	964	19,669

Year.	Glass and Glassware.	Iodine. *Kg.*	Iron.				
			Ore.	Pig and Scrap.	Bars and Hoops.	Nails and Spikes.	Steel.
1896	1,231	1,959	2,051	5,493	12	10.664	132
1897..................	1,432	2,395	4,242	4,631	56	9,097	167
1898	841	5,474	4,601	3,844	25	7,270	158
1899	840	16,180	12,517	6,085	337	6,089	377
1900..................	1,531	11,210	27,158	8,141	135	5,643	220
1901..................	2,142	10,000	39,173	3,250	370	6,001	179
1902..................	(c)	....	48,775	7,359	166	6,431	240
1903..................	200	11,417	41,575	6,350	10	6,504	200
1904..................	219	9,414	45,434	10,152	13	7,477	167
1905..................	381	12,000	60,558	9,920	34	8,725	88
1906..................	404	13,000	81,398	7,362	8	6,786	21

Year.	Nickel Ore.	Pyrites.	Silver Ore.	Stone.			
				Ashlar.	Marble and Dolomite.	Quartz.	Whetstones.
1896	*Nil.*	41,562	174	66,233	5,421	3,178	205
1897	*Nil.*	70,552	119	74,492	3,111	5,608	112
1898	30	67,502	79	98,692	4,267	2,244	137
1899	63	83,912	14	105,591	2,814	3,291	170
1900	272	84,604	90	101,959	3,134	3,523	136
1901	55	104,151	6	121,362	2,052	3,512	181
1902	1	105,980	*Nil.*	144,691	2,341	3,428	138
1903	*Nil.*	118,148	*Nil.*	165,874	4,491	4,485	170
1904	30	116,550	*Nil.*	189,237	3,132	6,679	169
1905	220	147,155	*Nil.*	173,558	....	....	137
1906	*Nil.*	164,119	*Nil.*	158,949	....	....	152

(*a*) From *Tabeller vedkommende Norges Bergvaerksdrift und Tabeller vedkommende Norges Handel.* (*b*) Includes flowers of sulphur. (*c*) Returns not available. (*d*) Includes a small quantity of fluorspar.

PORTUGAL.

The subjoined table reports the mineral production of Portugal:

MINERAL PRODUCTION OF PORTUGAL. (*a*)
(In metric tons.)

Year.	Antimony Ore.	Arsenic.	Coal, (Anthracite) (*c*)	Coal Lignitic.	Ore.	Copper.	
						Cement.	Pyrite.
1895	753	(*b*)	8,787	10,309	202	5,055	195,304
1896	595	(*b*)	8,743	8,000	436	3,453	207,440
1897	418	524	7,996	9,342	241	3,304	276,738
1898	245	751	10,250	12,291	290	3,145	302,686
1899	59	1,083	11,930	10,269	408	2,521	347,234
1900	38	1,031	24,066	(*b*)	(*b*)	2,948	402,870
1901	(*b*)	527	16,000	(*b*)	(*b*)	2,061	443,397
1902	68	736	11,000	5,792	655	2,205	413,714
1903	83	698	8,063	(*b*)	527	2,448	376,177
1904	31	1,370	12,805	(*b*)	297	(*b*)	383,581
1905	84	1,562	11,449	(*b*)	210	2,148	352,479
1906	481	1,322	6,762	(*b*)	196	3,634	350,746

Year.	Gold Ore.	Iron Ore.	Lead Ore. (Galena)	Manganese Ore.	Tin Ore. and Metal	Tungsten Ore.
1895	222.0	(*b*)	1,346	1,240	3	12
1896	(*b*)	(*b*)	1,333	1,494	6	14
1897	17.0	(*b*)	2,180	1,652	9	29
1898	6.8	2,519	3,242	907	102	59
1899	13.0	15,078	3,468	2,949	30	55
1900	(*d*)2.6	19,803	3,620	1,970	81	49
1901	(*d*)2.0	21,599	445	904	31	90
1902	(*d*)2.0	19,914	1,651	(*b*)	24	234
1903	(*d*)1.3	15,200	830	30	(*b*)	228
1904	*Nil*	12,488	50	(*b*)	51	358
1905	*Nil*	3,200	291	(*b*)	20	290
1906	*Nil*	(*b*)	511	22	22	570

(*a*) From a report specially furnished THE MINERAL INDUSTRY by Señor Severiano Augusto da Fonseca Monteiro, Chief of the Department of Mines of the Ministerio das Obras Publicas except for 1904 and subsequent years, which are from official Government reports. (*b*) Not reported. (*c*) Consumed in the country. (*d*) *Kg.* fine metal.

RUSSIA.

The mineral and metallurgical production of Russia, according to official statistics especially reported to THE MINERAL INDUSTRY, is given in the subjoined tables. The latest available statistics are those for 1906;

MINERAL AND METALLURGICAL PRODUCTION OF RUSSIA. (a)
(In metric tons; one metric ton=61.05 poods.)

Year.	Asbestos.	Chrome Ore.	Coal.	Copper.	Gold (b).	Pig-iron.	Lead.	Manganese Ore.
1895..........	1,131	21,014	9,098,486	5,854	$24,198,383	1,452,338	411.9	203,081
1896..........	1,275	6,682	9,377,560	5,832	21,667,269	1,620,814	261.5	191,645
1897..........	1,016	13,433	11,202,750	6,940	22,194,664	1,880,130	450.1	263,115
1898..........	1,665	15,466	12,307,463	7,290	22,195,208	2,241,293	241.2	329,276
1899..........	2,693	19,146	13,974,376	7,533	22,396,315	2,708,752	321.8	659,302
1900..........	3,845	18,233	16,156,055	8,258	22,369,864	2,933,786	220.7	802,236
1901..........	4,398	22,169	16,526,652	8,467	22,763,967	2,866,779	156.0	522,395
1902..........	4,508	19,656	16,465,852	8,817	22,258,343	2,598,086	225.3	536,519
1903..........	5,264	16,421	17,868,515	9,232	24,147,222	2,487,783	106.3	414,334
1904..........	7,502	26,575	19,608,631	9,835	24,627,537	2,972,115	90.3	430,090
1905..........	5,896	27,051	18,727,766	8,515	20,521,587	2,628,101	700.2	508,635
1906..........	(c)	(c)	21,593,158	9,296	20,020,862	2,694,895	906.8	1,015,686

Year.	Petroleum.	Phosphate Rock.	Platinum. (Kg.)	Pyrites.	Quick-silver.	Salt.	Silver. (Kg.)	Sulphur.	Zinc.
1895..........	6,290,000	6,327	4,414	11,042	434	1,540,195	7,887	190	5,030
1896..........	6,371,826	3,776	4,930	11,550	491	1,346,118	7,808	437	6,257
1897..........	6,945,127	5,917	5,601	19,380	616	1,561,895	4,779	574	5,874
1898..........	8,009,828	1,867	6,016	24,570	362	1,505,602	5,143	1,018	5,664
1899..........	8,517,608	16,863	5,962	23,250	362	1,679,726	4,419	451	6,326
1900..........	9,844,390	25,663	5,089	23,154	141	1,968,007	2,293	1,587	5,963
1901..........	10,925,471	21,276	6,371	30,732	363	1,705,924	1,088	2,489	6,104
1902..........	10,445,536	13,709	6,135	26,465	416	1,847,021	1,200	1,800	8,264
1903..........	9,759,214	14,635	6,009	22,780	362	1,658,938	1,152	281	9,894
1904..........	10,058,968	20,282	5,016	31,667	332	1,908,275	726	16	10,612
1905..........	7,505,637	(c)	5,250	30,689	318	1,844,678	2,965	(c)	7,911
1906..........	8,167,934	(c)	5,776	(c)	210	1,730,934	430	(c)	9,602

(a) From.
(b) The value of gold is taken at $20.67 per ounce.
(c) Not reported.

SOUTH AMERICA

The following tables itemize the statistics of the production and the foreign commerce, or both, of mineral and metallurgical products of South American countries so far as available. No statistics later than those given in the tables have been published.

MINERAL AND METAL PRODUCTION OF BOLIVIA. (*a*)

(In metric tons.)

Year.	Antimony. Ore.	Bismuth.	Borate of Lime.	Cobalt. Ore.	Copper. (*c*)	Gold. (*b*)	Silver. (*d*)	Tin. Ore.	Tungsten. Ore.
1903......	59	288	1,197	3.8	4,093	$33,810	39,063	18,425	68
1904......	7	406	1,080	1.5	3,228	17,130	21,172	20,692	700
1905......	17	592	2,146	....	6,708	15,044	8,266	26,428	68
1906......	....	231	....	....	4,347	17,403	....	29,374	....

(*a*) From a British Consular report.
(*b*) Reduced to U. S. currency.
(*c*) Includes ingots, precipitate, matte and ore.
(*d*) Includes ingots, ore and sulphide.

MINERAL EXPORTS OF BRAZIL. (*a*)

(In metric tons or dollars.) (*d*)

Year.	Agate.	Carbonado.	Copper Ore.	Diamonds.	Gold.	Manganese Ore.	Mica and Talc.	Monazite	Platinum. (Grams)	Precious Stones. (*b*)	Rock Crystal.
1902	81	$49,611	234	$79,071	$636,739	157,295	11.0	1,205	....	$4,332	....
1903	74	66,888	316	62,248	684,389	161,926	7.0	3,299	1,315	8,247	....
1904	54	32,063	610	34,975	611,198	208,260	14.0	4,860	2,122	12,505	35
1905	83	113,157	658	142,459	647,581	224,377	1.0	4,437	72,000	88,463	23
1906	121	319,743	1,484	340,137	771,611	121,331	6.0	4,351	*Nil*	141,395	37
1907	(*c*)	111,157	(*c*)	33,713	603,640	236,778	6.5	4,438	*Nil*	33,335	37

(*a*) As reported by the *Brazilian Review*.
(*b*) Other than carbonado and diamonds.
(*c*) Statistics not available.
(*d*) The par exchange value of the *Mil Reis* in 1907 was $0.546 U. S. gold. Common exchange value was in 1902, $4.155; in 1903, $4.134; in 1904, $4.146; in 1905, $3.153; in 1906, $3.103; and in 1907, $3.301.

MINERAL PRODUCTION OF CHILE. (*a*)

(In metric tons.)

Year.	Borax.	Coal.	Cobalt Ore.	Copper.	Gold *Kg.*	Guano.	Iodine.	Salt.	Silver. *Kg.*	Sodium Nitrate.	Sulphur.
1896		(*c*)	(*d*)	23,649	1,634	(*f*)	(*e*)	2,434	150,480	1,158,088	940
1897		(*c*)	(*d*)	21,128	1,538	(*f*)	(*e*)	5,867	140,732	1,148,696	664
1898		(*c*)	(*d*)	26,331	2,037	(*f*)	(*e*)	6,684	131,995	1,283,563	1,256
1899	86,892 (*b*)	(*c*)	(*d*)	25,719	2,060	(*f*)	274	9,937	129,503	1,389,823	989
1900		(*c*)	(*d*)	25,715	1,975	(*f*)	302	9,879	73,071	1,460,100	2,472
1901		(*c*)	(*d*)	30,155	1,100	(*f*)	269	10,099	70,237	1,273,800	2,516
1902		(*c*)	(*d*)	27,066	1,286	(*f*)	242	9,532	57,418	1,400,408	2,636
1903	16,879	827,112	290	29,923	994	11,134	387	16,264	28,552	1,444,920	3,560
1904	16,733	751,628	125	31,025	1,135	2,669	461	17,674	28,501	1,487,598	3,594
1905	19,612	793,927	28	29,126	1,055	19,380	564	12,108	16,315	1,669,806	3,470

(*a*) From *Estadistica Minera de Chile.* (*b*) The combined output of the years 1894 to 1902 inclusive. (*c*) The combined output of Chile up to the end of 1902 is estimated at 20,650,000 tons. (*d*) The combined output of Chile up to the end of 1902 is estimated at 5941 tons. (*e*) Not reported (*f*) The combined output of Chile up to the end of 1902 is estimated at 163,704 tons, valued at 5,041,560 pesos ($1,840,169).

MINERAL EXPORTS OF CHILE. (*a*)
(In metric tons.)

Year.	Borate of Lime.	Coal.	Cobalt Ore.	Copper Matte.	Copper and Silver Matte.	Copper, Silver and Gold Matte.	Copper Ore.	Copper and Silver Ore.	Copper Silver and Gold Ore.	Copper Ingots.	Gold Bullion *Kg.*
1894	6,700	205,201	4.6	342	1,508	2.5	11,106	90.3	460	19,640	1,475.4
1895	4,425	195,115	13.4	417	664	15.3	6,963	84.4	2,012	20,042	1,184.5
1896	7,486	204,858	(*b*)	2,528	1,059	7.6	6,159	62.3	29,542	20,592	1,061.3
1897	3,154	243,968	6.0	2,519	904	(*b*)	3,396	161.8	(*b*)	19,011	1,131.7
1898	7,028	282,663	18.2	3,079	419	17.8	20,301	87.0	5,733	20,600	1,630.5
1899	14,951	241,995	55.	1,710	1,094	93.0	35,854	184.0	12,000	17,311	1,625.0
1900	13,177	325,042	26.8	4,838	1,918	241.8	20,213	238.5	360	20,340	1,871.1
1901	11,457	(*b*)	76.0	2,905	1,779	208.0	15,921	119.0	60	24,480	637.0
1902	14,327	(*b*)	464.0	2,094	(*b*)	220.0	22,622	133.0	2,000	21,197	762.0
1903	16,879	200,000	284.5	2,689	864	863.6	17,961	89.8	440	22,196	207.5
1904	16,733	213,262	125.0	472	460	1,092	21,899	88.3	1,152	26,442	397.0
1905	19,612	227,800	28.6	2,657	2,656	1,658	17,045	36.2	1,211	23,410	328.7

Year.	Gold Ore.	Iodine.	Lead and Silver in Bars.	Manganese Ore.	Mineral Specimens.	Silver Ore.	Silver and Gold Ore.	Silver Ingots. *Kg.*	Silver Lead Ore.	Silver-Sulp'de Ore.	Sodium Nitrate.
1894	192	323	87	47,994	$ 1,150	370	56	153,723	15	127	1,081,337
1895	270	144	93	24,075	2,800	2,137	113	148,747	21	99	1,220,427
1896	367	206	594	26,151	700	2,750	666	151,226	*Nil.*	160	1,111,757
1897	64	243	369	23,529	20,300	984	260	143,541	6	183	1,057,640
1898	8	235	13	20,851	1,400	284	269	139,756	12	290	1,294,227
1899	12	304	171	40,931	64,521	302	370	75,899	32	339	1,380,718
1900	129	318	14	25,715	3,550	225	217	45,623	1	172	1,465,935
1901	66	385	455	18,480	(*b*)	6,166	196	46,164	(*b*)	264	1,291,958
1902	115	244	99	12,990	(*b*)	114	610	31,812	161	176	1,330,598
1903	57	387	*Nil.*	17,110	292	55	1,216	(*c*) 10,857	102	17	1,443,286
1904	301	461	17	2,324	220	88	1,549	6,658	5	36	1,486,190
1905	103	564	0.2	1,323	18	10	1,052	2,348	0.6	31	1,668,976

(*a*) From *Estadistica Minera de Chile.* (*b*) Not reported. (*c*) Contains 500 kg. of gold. There is no real exportation of Chilean coal for foreign consumption; that supplied to steam vessels is considered as exported.

MINERAL AND METAL PRODUCTION OF PERU. (*a*)
(In metric tons.)

Year.	Bismuth.	Borate.	Coal.(*b*)	Copper.	Gold. *Kg.*	Lead.	Nickel. *Kg.*	Petroleum.	Quicksilver. *Kg.*	Silver. *Kg.*	Salt.	Sulphur.
1903...	......	2,466	36,920	9,497	1,078.3	1,302	....	37,079	....	170,800	17,637	....
1904...	......	2,675	59,920	9,504	601.4	2,209	....	38,683	....	145,165	18,545	....
1905...	12	1,954	75,338	12,213	776.6	1,476	1,778	49,700	1,554	191,476	21,039	....
1906...	......	2,598	79,969	13,474	1,247.0	2,568	....	70,832	2,304	230,300	20,226	1,830

(*a*) Reported by the Cuerpo de Ingenieros de Minas del Peru, in its *Boletin.* (*b*) Includes asphaltum and bituminous schist.

SPAIN.

The following tables record the mineral and metal production of Spain, as reported by official authorities:

MINERAL PRODUCTION OF SPAIN. (a)
(In metric tons.)

Year.	Aluminous Earths.	Antimony ore.	Arsenic	Asphaltum.	Asphalt Rock.	Barytes.	Cement, Hydraulic.	Coal.
								Anthracite.
1896	320	54	271	1,285	1,117	345	130,738	14,895
1897	409	354	244	1,878	1,656	429	159,439	8,758
1898	505	130	111	2,354	2,383	364	164,862	20,105
1899	685	50	101	2,646	2,542	887	165,645	34,842
1900	420	30	150	2,331	4,193	833	185,811	68,427
1901	305	10	120	4,182	3,956	1,067	189,909	85,266
1902	337	67	*Nil.*	6,034	6,301	642	201,856	109,298
1903	381	42	1,088	4,675	6,277	507	245,294	108,959
1904	925	245	400	3,463	3,761	453	286,737	163,275
1905	221	77	1,140	5,805	5,725	290	296,605	159,519
1906	386	180	1,114	6,229	7,794	330	299,294	113,747

Year.	Coal. (*Continued.*)			Coke.	Copper Ore.		Copper.		
	Bituminous.	Lignitic.	Briquettes.		Argentiferous.	Pyritic.	Fine.	Matte.	Precipitate.
1896	1,852,947	55,413	343,432	288,523	(c) 157,365	2,200,919	6	16,378	29,873
1897	2,010,960	54,232	332,272	755,394	(c) 18,488	2,161,182	7	16,120	29,652
1898	2,414,127	66,422	369,418	768,151	203	2,299,444	593	16,024	29,703
1899	2,565,437	70,901	348,838	341,443	1,103	2,443,044	4	15,755	41,927
1900	2,514,545	91,133	341,156	381,000	2,006	2,714,714	5	18,159	29,652
1901	2,566,591	95,867	338,684	455,586	(b)	2,672,365	79	15,634	28,433
1902	2,614,010	84,242	331,957	404,503	878	2,617,776	(b)	.....	36,045
1903	2,587,652	104,232	339,120	433,780	3,056	2,796,733	(b)	.....	27,448
1904	2,903,771	190,773	307,630	432,726	(b)	2,624,512	(b)	8,117	29,494
1905	2,912,466	168,994	290,830	448,073	(b)	2,621,054	(b)	8,243	17,988
1906	3,095,043	189,048	311,328	435,808	(b)	2,888,778	(b)	9,008	19,200

Year.	Fluor-spar.	Gold and Silver Ore.	Iron Ore.		Iron and Steel.			Kaolin. (China Clay.)
			Argentiferous.	Non-Argentiferous.	Pig.	Forged Iron.	Steel.	
1896	3	854	3,581	6,762,582	100,786	53,793	68,126	1,240
1897	2	2,456	5,559	7,419,768	146,940	80,894	66,007	6,294
1898	5	555	24,190	7,197,047	113,492	65,900	50,362	5,445
1899	310	(d) 1,110	17,139	9,397,733	113,071	40,332	112,982	2,790
1900	4	(d) 1,300	26,348	8,675,749	91,126	54,307	144,355	3,794
1901	(b)	(d) 1,595	27,726	7,906,517	135,600	47,085	121,023	2,220
1902	93	(d) 1,764	24,361	7,904,555	330,747	......	163,564	3,412
1903	4,000	(d) 2,681	90,996	8,304,153	380,284	......	199,642	2,578
1904	(b)	(b)	122,109	7,964,748	283,819	50,858	186,705	1,700
1905	(b)	(b)	152,027	9,007,245	305,462	11,366	223,545	720
1906	70	(b)	126,445	9,448,533	315,309	6,035	274,280	610

Year.	Lead. (Argentiferous.)		Lead. (Non-Argentiferous.)		Manganese Ore.	Mineral Paints. (Ocher.)	Phosphate Rock.	Pyrites. (Iron)
	Ore.	Metal.	Ore.	Metal.				
1896	182,565	84,802	104,160	82,215	38,265	212	770	100,000
1897	186,692	91,258	110,496	75,112	100,566	200	2,084	100,000
1898	244,068	88,981	150,472	78,370	102,228	200	4,500	70,265
1899	184,906	70,874	123,753	91,739	104,974	100	3,510	107,386
1900	182,016	74,341	131,437	98,189	112,897	58	4,170	34,638
1901	207,188	73,895	174,376	95,399	60,325	164	4,220	33,953
1902	227,645	74,370	100,403	103,190	46,069	(b)	1,150	145,173
1903	179,858	56,687	108,660	118,422	26,194	(b)	1,124	155,739
1904	177,104	57,956	93,230	127,906	18,732	(b)	3,305	161,841
1905	160,381	56,361	105,113	129,332	26,020	(b)	1,370	179,079
1906	158,425	53,856	105,095	131,614	62,822	164	1,300	189,243

Year.	Pyrites (arsenical.)	Quicksilver.		Salt.	Silver.		Soapstone.
		Ore.	Metal.		Ore.	Metal. *Kg.*	
1896	(b)	34,959	1,524	521,751	1,230	64,554	756
1897	(b)	32,378	1,728	508,606	982	71,168	3,601
1898	230	31,361	1,691	479,358	967	76,295	2,613
1899	(b)	32,144	1,361	598,108	764	88,409	4,844
1900	515	30,216	1,095	450,041	742	140,457	8,109
1901	1,328	23,367	754	345,063	391	94,977	4,880
1902	5,648	26,037	1,425	426,434	175	96,975	542
1903	7,996	30,370	968	427,394	231	112,978	3,725
1904	3,510	27,185	1,130	543,658	303	117,418	5,165
1905	4,790	26,485	853	493,451	540	123,607	4,364
1906	2,434	28,965	1,568	541,978	470	126,424	3,609

Year.	Sulphur.		Tin Ore (Dressed.)	Topaz. *Kg.*	Tungsten Ore.	Zinc.		
	Crude rock.	Refined.				Ore.	Spelter.	Sheets.
1896	26,204	1,800	(e)2,348	80	31	64,828	3,485	2,648
1897	18,805	3,500	(e)2,378	44	10	73,848	3,907	2,337
1898	34,943	3,100	4	90	37	99,836	4,300	1,731
1899	58,922	1,100	57	44	151	119,710	4,100	2,084
1900	64,364	750	47	95	1,958	86,158	2,855	2,756
1901	49,856	610	115	310	6	119,708	2,573	2,781
1902	15,442	450	12,762	*Nil.*	11	127,618	5,569	(b)
1903	38,573	1,680	330	90	*Nil.*	154,126	5,134	(b)
1904	40,389	605	229	....	60	156,329	5,887	2,913
1905	38,153	610	209	....	375	160,561	6,184	2,936
1906	28,965	700	86	171	430	170,384	6,209	2,639

(*a*) Figures are from *Estadistica Minera de España* except for 1896 and 1898 which were from the official *Reports of the Junta Superior Facultativa de Minas,* Madrid. (*b*) Not reported. (*c*) Represents non-argentiferous copper ore. (*d*) Gold ore only. (*e*) Undressed tin ore.

SWEDEN.

The official statistics of mineral production, imports and exports are summarized in the following tables, 1906 being the latest year for which the reports have been published:

MINERAL PRODUCTION OF SWEDEN. (a)
(In metric tons.)

Year.	Alum.	Fire Clay.	Coal.	Copper.			Feldspar.	Gold. *Kg.*
				Ore.	Ingot.	Sulphate.		
1897	131	112,283	224,343	25,207	289	1,315	19,298	113.3
1898	153	131,391	236,277	23,335	235	1,165	20,737	125.9
1899	164	129,875	239,344	22,334	179	1,287	16,017	106.2
1900	167	160,585	252,320	22,725	136	1,265	15,228	88.5
1901	121	175,876	271,509	23,660	137	1,224	13,502	62.7
1902	132	(b)	304,733	30,095	178	1,257	17,960	94.3
1903	140	172,718	320,390	36,687	776	1,171	19,392	50.6
1904	125	166,888	320,984	36,834	533	1,248	18,021	60.9
1905	139	119,947	322,384	39,255	1,385	1,029	19,224	55.0
1906	170	95,556	296,980	19,655	1,209	562	21,014	20.3

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate	Bessemer.	Basic.	Crucible.
1897	2,086,119	538,197	189,633	304,537	232	107,679	165,836	691
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434,606	497,727	195,331	328,999	105	91,898	179,357	1,225
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(b)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,878	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	252,832	1,162
1905	4,364,833	539,437	182,640	356,898	144	78,204	288,675	1,319
1906	4,501,656	604,789	178,298	381,118	170	84,633	311,435	1,457

Year.	Lead.		Manganese Ore.	Pyrites.	Silver-lead Ore.	Silver. *Kg.*	Sulphur.	Zinc Ore.
	Ore.	Pig.						
1897	99	1,480	2,749	517	10,068	2,218	(b)	56,636
1898	50	1,559	2,358	386	6,743	2,033	50	61,627
1899	35	1,606	2,622	150	5,730	2,290	(b)	65,159
1900	85	1,424	2,651	179	5,300	1,927	70	61,044
1901	56	988	2,271	*Nil.*	11,366	1,557	(b)	48,630
1902	63	843	2,850	*Nil.*	9,378	1,365	74	48,783
1903	25	678	2,244	7,793	9,792	1,005	(b)	62,927
1904	55	589	2,297	15,957	8,187	651	35	57,634
1905	40	576	1,992	20,762	8,397	606	....	56,885
1906	37	753	2,680	21,827	1,938	938	....	52,552

(a) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen.* (b) Not reported.

MINERAL IMPORTS OF SWEDEN. (a)

(In metric tons or dollars; 1 krone=27 cents.)

Year.	Alum.	Aluminum Sulphate.	Ammonium.					Anti-mony (crude).	Arseni-ous Acid.
			Carbonate.	Chloride.	Hydrate.	Nitrate.	Sulphate.		
1896	75	629	79	88	81	11	88	63	33
1897	103	733	109	110	59	42	67	58	33
1898	136	968	99	101	105	12	81	53	33
1899	158	866	89	112	110	12	181	59	12
1900	133	1,197	141	99	100	5	227	85	22
1901	346	1,192	131	113	92	1	285	50	12
1902	250	1,430	127	145	130	13	241	59	14
1903	302	1,082	113	133	150	47	197	54	21
1904	87	1,245	154	208	115	41	254	67	17
1905	106	1,127	142	170	70	26	189	67	16
1906	140	1,730	184	172	46	43	338	94	20

Year.	Asbestos. (c)	Asphalt.	Barytes	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement	Chalk. White, Unground. *Hectoliters.*
1896	116	4,092	298	128	73	4,334	2,901	6,148
1897	119	5,458	270	175	56	5,549	1,826	14,368
1898	112	5,409	299	196	75	5,401	1,656	7,016
1899	567	6,286	292	190	65	4,914	1,363	16,079
1900	763	5,676	411	194	66	6,084	1,941	12,059
1901	178	4,524	295	253	68	6,602	2,868	13,569
1902	213	5,779	....	242	71	7,278	9,822	11,583
1903	217	5,957	....	240	71	7,419	11,145	41,868
1904	356	6,243	....	299	77	10,128	10,526	10,115
1905	140	4,760	264	294	82	18,788	10,999	13,305
1906	287	7,134	559	321	79	9,908	13,136	10,777

Year.	Coal.	Copper, also Alloys of Copper.	Emery.	Gold.		Graphite.	Gypsum.	Iron (crude).
				Bars and Mfres. *Kg.*	Specie.			
1896	1,991,760	4,037	104	1,161	$ 608	135	4,940	34,549
1897	2,240,247	4,944	128	4,267	948	158	7,260	89,606
1898	2,392,451	5,227	131	3,998	2,396	167	7,979	76,832
1899	3,047,618	4,740	125	362	9,774	162	6,457	68,909
1900	3,033,885	4,745	136	3,365	98,905	213	6,794	82,957
1901	2,793,309	5,153	169	1,454	736,852	180	6,589	66,131
1902	2,911,286	6,890	147	945	*Nil.*	(b)	6,754	43,828
1903	3,192,990	6,109	132	89	965,346	(b)	8,795	49,411
1904	3,367,826	7,367	221	1,400	1,207,187	(b)	8,868	90,102
1905	3,297,485	6,481	271	1,765	750,009	(b)	11,270	87,843
1906	3,718,884	8,899	284	1,035	463,034	(b)	13,496	108,193

Year	Lead.	Lime *Hectoliters.*	Litharge.	Phosphorus. *Kg.*	Plati-num. *Kg.*	Porcelain.	Potassium.			
							Chloride.	Cyanide. *Kg.*	Hydrate.	Carbonate.
1896	1,911	7,768	150	52,482	34	327	241	2,122	285	1,933
1897	2,098	20,050	199	57,972	63	362	363	2,922	1,381	1,432
1898	2,139	23,079	160	66,466	49	298	259	2,604	1,451	1,112
1899	2,125	34,343	177	59,989	59	346	225	2,313	1,266	1,231
1900	2,067	25,047	148	67,557	99	382	364	2,221	1,915	1,257
1901	1,991	12,204	165	70,672	172	386	260	2,658	1,435	1,266
1902	2,509	6,579	172	68,441	130	....	222	2,950	1,720	1,238
1903	2,644	6,449	237	112,659	116	....	245	3,294	2,034	1,150
1904	2,849	13,388	213	47,421	84	396	214	3,237	2,234	1,184
1905	2,823	16,099	205	69,526	105	416	1,296	3,437	2,251	1,133
1906	3,457	10,578	255	79,048	133	486	1,986	4,106	2,486	1,082

Year.	Quick-silver. *Kg.*	Salt.		Silver.		Sodium.			
		Crude.	Refined.	Scrap and Mfres. *Kg.*	Specie.	Carbonate.	Hydrate.	Nitrate. (*d*)	Sulphate. (*e*)
1896........	5,194	84,629	3,673	7,375	$204,691	11,425	908	12,518	8,486
1897........	3,125	87,050	3,055	20,557	136,823	14,625	625	12,531	11,384
1898........	2,631	85,246	2,188	21,696	191,766	11,917	575	15,419	11,544
1899........	4,210	98,417	3,166	11,565	156,707	13,323	929	15,006	15,140
1900........	3,629	70,302	3,098	11,559	62,315	12,680	1,038	14,245	15,590
1901........	5,958	79,038	3,072	7,476	78,416	13,669	800	17,614	15,494
1902........	4,866	82,439	3,037	4,853	74,826	....	1,623	15,553	18,924
1903........	5,043	88,139	3,419	11,259	90,366	....	1,426	20,616	16,120
1904........	5,768	84,237	4,615	19,034	86,891	11,898	2,112	19,776	17,596
1905........	4,609	87,677	3,889	11.067	82,620	13,592	1,489	23,183	17,115
1906........	5,535	88,341	3,700	15,253	93,990	14,977	1,478	27,174	19,948

Year.	Sulphur.	Sulphuric Acid.	Tin.		Zinc.
			Salts–*Kg.*	Block.	
1896..	11,369	615	4,437	551	2,275
1897..	9,723	1,418	3,823	541	2,551
1898..	10,837	1,742	3,874	595	3,030
1899..	13,505	2,558	5,404	486	2,829
1900..	20,152	2,472	3,243	630	2,912
1901..	20,715	1,950	2,334	541	2,900
1902..	23,002	1,887	1,652	644	3,255
1903..	24,577	2,620	1,467	655	3,312
1904..	18,248	2,001	1,460	719	3,705
1905..	18,631	3,424	1.72	597	3.780
1906..	22,745	2,535	2,102	819	4,484

MINERAL EXPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1 krone=27 cents.)

Year.	Alum.	Ammo-nium Sulphate	Anti-mony, Crude-*Kg*	Asbestos. *Kg.*	Cement.	Coal.	Copper.		Graphite.
							Ore.	Copper & Alloys.	
1896................	40	100	800	2,040	22,991	141	1,094	1,911	3,500
1897................	54	180	800	1,348	27,112	74	(*b*)	933	7,215
1898................	32	36	4,700	1,055	28,676	496	1,102	1,346	9,108
1899................	26	2	2,600	2,812	31,101	762	315	1,230	16,664
1900................	24	2	4,600	2,436	42,564	1,108	448	2,012	17,719
1901................	56	156	1,800	2,179	17,794	716	602	1,243	16,761
1902................	20	174	4,090	1,864	19,499	866	845	1,516	5,420
1903................	22	*Nil*	3,473	15,357	21,319	509	1,555	1,858	8,744
1904................	9	219	3,810	16,339	27,509	605	749	1,396	(*b*)
1905................	12	445	3,147	2,386	38,504	425	2,137	2,654	(*b*)
1906................	11	30	4,584	1,510	45,960	1,352	1,841	2,602	(*b*)

Year.	Gypsum and Mfres.	Iron and Steel.		Lead and Mfres.	Lime. *Hectoliters.*	Peat.	Phos-phorus. *Kg.*	Potas-sium Chloride.
		Ore.	Unwrought.					
1896..................	9	1,150,695	304,138	1,182	102,787	1,452	1.510	254
1897..................	9	1,400,801	279,525	1,473	106,053	1,816	1,627	463
1898..................	27	1,439,860	301,192	570	123,939	1,616	4,085	506
1899..................	8	1,628,011	320,742	818	80,153	1,979	1,890	335
1900..................	10	1,619,902	304,175	1,209	84,242	3,843	879	931
1901..................	55	1,761,257	268,143	1,028	69,000	3,064	1,254	708
1902..................	117	1,729,000	(*f*) 73,403	546	81,796	3,620	1,290	1,114
1903..................	119	2,828,000	(*f*) 70,788	333	63,870	3,217	300	790
1904..................	162	3,065,522	(*f*) 88,124	275	56,417	4,212	1,994	1,266
1905..................	156	3,316,626	(*f*)120.987	512	113.702	5,157	34,388	1,499
1906..................	6	3,661,218	112,719	531	89,862	6,531	700	(*b*)

Year.	Salt, Refined. *Kg.*	Silver. *Kg.*		Soda.	Sulphur.	Tin.		Zinc.	
		Bullion.	Mfres.			Block.	Mfres-*Kg.*	Ore.	Crude and Mfrs.
1896	830	819	14	772	9	18.9	2,996	41,401	184
1897	1,424	329	119	686	11	25.6	7,113	44,425	135
1898	216	130	238	509	11	20.8	1,263	49,597	184
1899	110	367	258	227	68	8.8	1,033	45,634	157
1900	407	296	103	238	20	21.5	1,521	40,879	156
1901	1,556	179	9	237	12	20.4	8,110	41,248	101
1902	1,945	110	506	621	147	25.5	1,603	43,813	63
1903	*Nil*	484	195	10	217	43.3	3,893	45,389	351
1904	1,883	115	69	45	4	45.6	3,479	44,259	332
1905	*Nil*	10	77	403	4	33.9	654	51,765	295
1906	8,652	77	136	463	12	51.0	353	45,370	410

(*a*) From *Bidrag till Sveriges Officiella Statistik.* (*b*) Not reported. (*c*) Includes crude and manufactures. (*d*) Includes a small quantity of potassium nitrate. (*e*) Includes sodium bisulphate. (*f*) Includes only crude or ballast iron.

UNITED KINGDOM.

The statistics of the mineral production, imports and exports, according to official reports, are given in the subjoined tables.

MINERAL AND METALLURGICAL PRODUCTION OF THE UNITED KINGDOM. (a)
(In metric tons.)

Year.	Alum Shale.	Arsenious Acid.	Arsenical Pyrites.	Barytes.	Bauxite.	Chalk.	Clay. (e)	Coal.
1897..........	621	4,232	13,347	23,087	13,540	3,920,183	12,908,479	205,364,010
1898..........	13,835	4,241	11,272	22,581	12,600	4,366,782	14,974,290	205,287,388
1899..........	5,913	3,890	13,735	25,059	8,137	4,752,982	15,305,895	223,616,279
1900..........	1,329	4,146	9,727	29,937	5,871	4,444,765	14,279,181	228,772,886
1901..........	4,019	3,416	2,620	26,844	10,357	4,399,043	14,393,196	222,614,981
1902..........	5,755	2,165	842	23,986	9,192	4,466,004	15,549,002	230,728,562
1903..........	3,337	916	58	24,659	6,226	4,541,494	16,460,526	234,019,821
1904..........	6,636	992	44	26,748	8,839	4,509,768	16,210,734	236,130,373
1905..........	7,245	1,552	651	29,528	7,417	4,608,153	15,376,910	239,906,999
1906..........	9,605	1625	650	35,586	6,760	4,825,299	12,459,213	255,067,622
1907........	10,063	1,497	690	42,646	7,658	4,855,857	15,065,141	272,113,528

Year.	Copper.		Fluorspar.	Gold.		Gravel and Sand.	Gypsum.	Bog Ore (c)
	Ore and Precipitate.	Fine.		Ore.	Bullion. *Kg.*			
1897....................	7,470	526	302	4,589	63.2	1,378,496	184,287	7,238
1898....................	9,277	650	57	715	12.3	1,652,701	199,174	5,505
1899....................	8,452	647	796	3,096	103.5	1,800,208	215,974	4,390
1900....................	9,643	777	1,472	21,135	437.6	1,867,211	211,436	4,221
1901....................	6,903	541	4,232	16,641	194.5	1,990,926	204,045	2,649
1902....................	6,210	490	6,388	30,432	130.0	2,100,829	228,264	4,983
1903....................	6,977	545	12,102	29,057	171.0	2,281,689	223,426	4,156
1904....................	5,552	501	18,450	23,574	610.7	2,275,426	237,749	4,616
1905....................	7,267	727	38,606	16,237	169.0	2,277,486	259,596	3,256
1906....................	7,882	(b)	36,860	17,662	(b)	2,404,857	228,627	5,512
1907....................	6,867	(b)	40,873	13,186	(b)	2,438,798	247,537	6,391

Year.	Iron.		Lead.		Manganese Ore.	Mineral Paints.	Oil Shale.	Phosphate of Lime.
	Ore.	Pig.	Ore.	Pig.				
1897............	14,008,484	4,942,679	35,903	26,988	609	14,653	2,259,325	2,032
1898............	14,403,769	4,928,347	33,513	25,761	235	20,144	2,172,201	1,575
1899............	14,692,711	4,992,468	31,494	23,929	422	16,575	2,246,197	1,469
1900............	14,257,344	4,743,172	32,487	24,762	1,384	15,448	2,318,736	630
1901............	12,475,700	4,158,745	33,084	20,361	1,673	14,780	2,392,812	71
1902............	13,641,459	4,470,420	25,000	17,988	1,299	17,235	2,141,355	87
1903............	13,935,748	4,573,202	26,993	20,278	831	14,377	2,041,851	71
1904............	13,994,670	4,596,803	26,796	20,155	8,896	16,307	2,370,391	59
1905............	14,824,183	(f) 9,746,221	28,091	20,977	14,582	16,468	2,536,734	*Nil.*
1906............	15,748,412	(f) 9,999,211	30,710	(b)	23,126	14,437	2,586,851	*Nil.*
1907............	15,983,310	(f) 9,850,953	31,714	....	16,356	14,927	2,733,068	32

Year.	Pyrites.	Salt.	Silica. (chert and flint.)	Silver. *Kg.*	Stone.			
					Granite.	Limestone.(*d*)	Sandstone.	Slate.
1897	10,752	1,933,949	95,209	7,750	1,876,880	11,179,580	5,043,535	618,941
1898	12,302	1,908,723	83,370	6,575	1,905,830	12,172,267	5,325,988	679,461
1899	12,426	1,945,531	69,955	5,969	4,785,284	12,499,736	5,296,026	650,077
1900	12,484	1,873,601	78,971	5,964	4,709,997	12,099,940	5,101,868	595,428
1901	10,405	1,812,180	132,700	5,452	5,131,787	11,363,202	5,199,234	496,756
1902	9,315	1,893,881	100,938	4,560	5,554,696	12,368,196	5,571,121	525,665
1903	9,794	1,917,274	74,355	5,440	5,512,605	12,419,120	5,496,312	540,143
1904	10,452	1,921,899	66,300	4,967	6,084,642	12,235,825	5,391,265	572,181
1905	12,381	1,920,149	71,808	5,212	6,052,210	12,701,808	5,729,799	523,892
1906	11,318	1,996,593	69,300	....	6,264,402	12,962,725	5,345,328	500,546
1907	10,357	2,038,072	54,523	....	5,765,262	12,709,288	5,092,246	450,651

Year.	Strontium Sulphate	Tin.		Tungsten Ore.	Uranium Ore.	Zinc.	
		Ore, Dressed.	Block.			Ore.	Spelter.
1897	15,227	7,234	4,524	127	30	18,586	7,162
1898	13,148	7,498	4,722	331	26	23,929	8,711
1899	12,831	6,494	4,077	96	7	23,505	8,837
1900	9,270	6,911	4,337	9	42	25,070	9,214
1901	16,923	7,407	4,634	21	80	23,967	8,555
1902	32,799	7,681	4,462	9	53	25,462	9,275
1903	23,209	7,500	4,351	276	6	25,287	9,430
1904	18,460	6,849	4,198	164	*Nil.*	28,097	10,427
1905	14,523	7,316	4,540	174	105	24,025	9,023
1906	14,338	6,376	....	267	11	23,189	(*b*)
1907	10,917	6,184	....	327	72	20,264	....

(*a*) From *Mineral Statistics of the United Kingdom*. (*b*) Not reported. (*c*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*d*) Does not include chalk. (*e*) Includes China clay, potters' clay, and fuller's earth. (*f*) Includes production from imported ore.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Alkali.	Asphaltum.	Borax.	Brass and Bronze Mfres.	Clay Products, Porcelain and Earthenware.	Coal, Coke and Pat. Fuel.	Copper.		
							Ore.	Regulus and Precipitate.	Wrought, Unwrought and Old.
1897	11,557	44,541	(*b*)	2,129	17,847	9,605	83,916	90,008	62,055
1898	12,179	46,398	1,255	2,357	16,405	11,191	91,141	76,201	70,018
1899	12,078	59,073	3,076	1,988	18,341	1,777	130,611	84,015	60,502
1900	16,360	53,061	15,667	2,335	18,329	10,112	102,365	89,123	72,223
1901	(*c*)13,429	74,694	15,710	2,469	20,754	7,685	102,503	93,338	68,809
1902	(*c*)26,292	65,896	13,390	3,403	19,029	3,331	90,007	74,684	92,349
1903	(*c*)14,321	(*b*)	11,959	(*b*)	18,576	3,535	85,644	77,884	64,591
1904	(*c*)14,325	(*b*)	16,012	(*b*)	18,980	2,812	80,771	67,739	90,717
1905	(*c*)16,593	(*b*)	11,552	(*b*)	20,076	49,277	94,198	70.235	71,294
1906	(*c*)14,070	(*b*)	16,955	(*b*)	22,400	49,269	97,789	76,073	75,487
1907	(*c*)15,579	(*b*)	17,551	(*b*)	22,921	19,135	105,409	73.101	84,035

Year.	Diamonds.	Glass. All Kinds.	Gold.			Iron and Steel.		
			Ore.	Bullion and Specie.	Leaves. *Number.*	Iron Ore.	Pig Iron.	Scrap.
1897	$22,176,300	$15,034,555	$ 736,415	$154,044,290	68,173,400	6,064,179	(*f*)160,531	20,735
1898	22,619,075	16,423,915	1,393,175	218,614,800	63,632,700	5,555,889	(*f*)162,075	24,619
1899	20,597,910	16,044,400	761,755	162,667,485	49,108,570	7,168,061	(*f*)174,159	32,427
1900	17,168,180	15,997,555	1,432,970	130,954,365	54,346,025	6,398,639	178,199	31,687
1901	24,385,210	17,647,635	2,747,195	103,578,140	59,048,856	5,637,670	198,536	44,721
1902	26,901,950	18,484,960	2,688,170	108,145,245	59,334,023	6,542,793	226,708	39,584
1903	(*b*)	18,636,475	3,590,095	143,286,965	(*b*)	6,417,188	132,364	17,051
1904	(*b*)	16,901,500	4,259,360	169,382,940	(*b*)	6,198,368	132,494	19,326
1905	(*b*)	16,992,835	(*b*)	192,839,475	(*b*)	7,172,171	128,183	23,569
1906	(*b*)	16,346,330	(*b*)	230,062,950	(*b*)	7,634,839	90,674	36,559
1907	(*b*)	15,243,805	(*b*)	235,442,735	(*b*)	7,462,804	104,950	27,404

Year.	Iron and Steel. (*Continued.*)							
	Puddled and Wrought.	Sheets and Plates.	Rails.	Strips and Wire Rods.	Nails, Screws. Rivets, Bolts.	Steel Ingots, Blooms, Billets, etc.	Steel Bars, Shapes, Beams and Pillars	Mnfrs. Unenumerated (*k*)
1897	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,628	(*h*)	$27,894,295
1898	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	40,875	(*h*)	33,379,160
1899	(*g*)	(*g*)	(*h*)	(*g*)	(*g*)	78,257	(*h*)	39,527,075
1900	189,891	(*g*)	38,636	(*g*)	(*g*)	182,210	94,667	17,861,660
1901	102,811	(*g*)	55,809	(*g*)	(*g*)	185,810	124,648	18,252,435
1902	178,425	(*g*)	48,942	(*g*)	45,095	285,494	129,743	14,424,835
1903	196,084	73,079	74,939	35,574	51,888	278,441	343,259	9,007,990
1904	109,289	69,552	40,438	38,214	50,649	531,069	219,510	8,630,380
1905	110,576	69,831	34,439	60,318	55,331	613,612	148,995	7,783,895
1906	111,062	83,747	11,900	61,288	57,071	493,805	149,363	3,824,405
1907	83,145	57,280	19,337	56,110	51,863	332,442	90,327	3,731,275

Year.	Lead.		Manganese Ore.	Mica, Sheet.	Mica and Talc.	Paraffin.	Petroleum. *Liters.*	Phosphate Rock.
	Ore.	Pig and Sheet.						
1897	32,818	170,121	158,825	412	1,683	39,284	842,920,307	330,335
1898	44,457	197,591	156,390	517	1,398	48,104	829,995,751	334,884
1899	30,263	201,551	261,740	519	6,025	54,712	908,107,248	426,830
1900	21,566	198,416	270,098	469	7,952	50,033	965,167,850	361,309
1901	29,944	221,549	195,736	(*b*)	7,117	42,643	960,650,967	360,568
1902	25,838	235,522	237,066	1,078	6,127	52,023	1,078,095,152	370,697
1903	18,923	232,939	235,574	(*b*)	(*b*)	49,163	1,299,570,625	398,997
1904	8,748	250,452	208,458	(*b*)	(*b*)	42,882	1,373,488,176	425,978
1905	(*b*)	233,214	289,827	(*b*)	(*b*)	41,247	1,364,301,583	427,762
1906	(*b*)	211,577	314,016	(*b*)	(*b*)	44,673	1,130,667,737	450,058
1907	(*b*)	207,970	298,352	(*b*)	(*b*)	46,542	1,382,595,355	512,601

Year.	Paints, Mineral.	Platinum Wrought and Unwrought. *Kg.*	Potassium Nitrate.	Pyrites of Iron and Copper.	Quicksilver.	Silver Ore. (*e*)	Sodium Nitrate.
1897	$5,049,225	2,257	16,744	633,009	1,862	$7,149,210	107,525
1898	5,619,835	3,389	13,323	665,544	1,856	5,729,525	132,412
1899	5,842,063	5,404	12,635	712,393	1,759	5,162,750	163,387
1900	6,747,800	5,027	12,798	752,605	1,113	5,154,430	143,461
1901	6,489,710	4,917	12,115	664,041	1,202	5,309,920	108,822
1902	6,367,320	3,027	11,526	620,948	1,129	5,383,515	116,791
1903	7,025,050	(*b*)	9,425	747,714	1,187	6,596,045	118,582
1904	6,718,095	(*b*)	12,277	754,722	1,130	8,271,480	122,454
1905	7,168,725	(*b*)	8,260	709,926	1,158	10,426,570	106,107
1906	7,534,495	(*b*)	10,125	771,473	1,320	10,532,020	110,222
1907	8,257,915	(*b*)	10,719	781,447	1,341	11,224,650	115,716

Year.	Stone and Marble, Hewn or Mfr'd.	Sulphur.	Tin.		Zinc.		
			Ore.	Block, Ingot, Bars or Slabs.	Ore.	Spelter.	Mnfres.
1897	752,345	22,811	5,345	27,214	25,238	70,929	21,395
1898	883,699	19,642	5,710	20,665	53,945	78,761	21,613
1899	905,432	21,906	6,324	27,608	38,143	71,068	21,521
1900	961,492	22,993	7,449	33,648	42,755	61,504	21,751
1901	1,159,276	22,440	10,690	35,397	38,660	68,633	21,343
1902	1,192,023	23,863	12,255	35,713	45,312	89,688	21,717
1903	1,219,782	21,313	12,473	36,076	41,009	86,539	23,118
1904	1,329,502	17,629	15,734	39,932	54,438	90,088	22,788
1905	1,217,652	18,163	(*b*)	40,391	(*b*)	92,261	20,013
1906	1,245,174	22,704	21,003	44,306	(*b*)	95,203	19,664
1907	1,220,465	15,730	21,205	44,505	(*b*)	90,756	20,163

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported. (*c*) Classified as soda compounds since 1901. (*d*) Includes machinery and mill work. (*e*) Includes the value of silver in argentiferous ore and metal. (*f*) Includes puddled iron. (*g*) Not separately enumerated. (*h*) Former returns not available. (*k*) Prior to 1900 many manufactures were not reported separately.

MINERAL EXPORTS OF THE UNITED KINGDOM—DOMESTIC PRODUCTS. (a)
(In metric tons or dollars; £1=$5.)

Year.	Bleaching Materials.	Brass and Mfres.	Cement.	Clay. Unmanufactured.	Clay products. (c)	Coal.
1897	(b)	5,713	398,023	(b)	$8,629,425	35,919,965
1898	(b)	5,418	331,648	(b)	8,253,640	35,619,365
1899	(b)	5,797	359,273	(b)	9,314,255	41,839,217
1900	57,478	6,131	365,742	(b)	10,193,970	46,845,739
1901	46,912	5,905	318,216	(b)	9,963,985	42,547,114
1902	40,939	6,378	308,104	463,309	9,497,995	43,849,591
1903	49,415	7,646	406,388	490,157	10,879,940	45,669,258
1904	35,289	8,860	390,736	527,013	10,531,620	46,995,636
1905	42,526	12,526	463,863	573,706	10,487,585	48,236,334
1906	45,510	13,569	668,461	611,919	11,914,095	56,489,367
1907	48,856	11,691	777,741	665,224	13,249,345	64,618,562

Year.	Coke.	Patent Fuel.	Supplied to Steamers.	Coal Products (d)	Copper.			
					Ingot.	Mixed or Yellow Metal	Mfres.	Sulphate.
1897	993,980	(b)	10,623,050	$8,340,420	21,252	11,192	15,275	60,326
1898	782,053	(b)	11,444,431	7,624,740	27,102	10.452	13,765	52,573
1899	881,172	(b)	12,422,429	7,712,965	32,449	7,038	11,231	40,822
1900	1,001,131	(b)	11,940,353	9,058,220	18,300	8,940	10,765	43,601
1901	820,594	1,098,459	13,804,222	5,756,265	26,935	9,252	11,156	36,601
1902	669,664	1,067,060	15,390,485	5,991,025	21,658	13,314	14,075	43,995
1903	728,957	970,449	17,068,646	7,290,825	23,723	14,425	16,975	54,307
1904	779,060	1,257,589	17,465,954	6,879,400	14,791	16,704	18,467	71,367
1905	786,498	1,126,190	17,674,484	6,742,455	21,232	9.959	22,128	55,219
1906	828,266	1,399,244	18,887,656	7.226.790	19,778	7,149	16,195	43,670
1907	1,097,185	1,504,587	18,916,698	7,726,685	25,652	7,994	16,676	46,049

Year.	Fertilizers.	Glass. All kinds.	Iron.					
			Ore.	Pig.	Scrap.	Cast Iron and Mfres.	Wrought Iron, Shapes and Mfres.	Rails.
1897	(e)	4,359,535	(e)	(f)1,219,958	99,259	(b)	170.285	(g)795,983
1898	(e)	4,410,205	(e)	(f)1,058,973	86,602	(b)	152,911	(g)619,976
1899	(e)	4,578,415	(e)	(f)1,401,365	118,262	(b)	161,679	(g)601,266
1900	$12,041,450	5,172,910	(e)	(f)1,450,365	96,567	(b)	159.677	379,939
1901	11,987,555	5,285,775	(e)	(f) 852,609	86,559	(b)	119,962	474,073
1902	13,862,875	5,489,650	4,062	1,120,207	104,890	(b)	(b)	(b)
1903	14,254,340	5,512,470	4,534	1,082,426	143,929	62,249	217,139	613,741
1904	14,869,250	5,049,495	6,706	823,909	166,010	49,004	173,233	533,895
1905	14,425,085	5,558,820	14,664	997,601	151,619	49,193	186,340	555,390
1906	18,164,135	6,393,930	13,415	1,670,753	180,547	54,876	203,521	470,652
1907	20,036,390	7,002,355	15,538	1,978,350	162,295	43,218	215,159	440,576

Year.	Iron. (*Continued.*)								Lead. Pig and Mfres.
	Wire and Mfres. of.	Plates and Sheets.	Galvanized Sheets.	Black Plates for Tinning.	Tinned Plates.	Steel Ingots, Billets, Blooms, etc	Steel Shapes Beams and Pillars.	Total Iron and Steel and Mfres. of.	
1897	52,471	120,868	231,319	59,663	276,260	304,249	(b)	3,750,122	40,911
1898	44,954	102,638	230,219	59,289	255,797	290,182	(b)	3,299,326	38,684
1899	50,041	111,773	242,167	86,936	260,735	333,837	(b)	3,777,098	40,923
1900	39,104	39,157	251,203	66,810	278,338	313,383	(b)	3,602,083	36,576
1901	48,107	36,418	254,290	52,217	275,661	217,236	(b)	2,944,083	38,166
1902	(b)	(b)	336,572	58,245	317,201	306,152	(b)	3,529,223	33,537
1903	60,800	165,672	357,665	66,279	297,485	13,427	159,330	3,621,635	36,152
1904	61,894	154,774	391,608	63,467	365,262	4,324	176,232	3,315,047	35,600
1905	82,519	207,866	413,533	69,937	360,630	8,735	219,491	3,781,059	42,265
1906	96,641	279,459	450,221	66,749	381,421	11,924	311,231	4,763,868	45,612
1907	103,180	305,399	476,838	72,675	411,814	13,705	344,135	5,249,028	44,012

Year.	Salt.	Sodium.				Tin. Block.	Zinc.		
		Soda Ash.	Carbonate and Bicarbonate.	Hydrate.	Sulphate.		Ore.	Spelter.	Mfres.
1897	680,477	(h)252,736	(i)	(i)	(i)	5,050	6,072	6,951	1,047
1898	698,882	(h)191,578	(i)	(i)	(i)	5,557	6,483	7,577	1,227
1899	638,213	(h)193,492	(i)	(i)	(i)	4,785	8,171	5,492	1,249
1900	556,704	(h)185,783	(i)	(i)	(i)	5,713	13,913	7,136	1,159
1901	627,078	58,412	22,161	50,624	26,057	5,584	13,981	7,512	1,256
1902	624,752	59,894	24,654	61,658	35,672	6,210	16,717	6,756	1,345
1903	594,300	58,605	23,574	59,725	45,630	6,349	15,659	8,102	(k)
1904	632,605	61,327	25,252	61,985	40,324	5,953	14,606	7,993	(k)
1905	588,389	67,678	28,425	68,675	33,681	7,741	(b)	7,451	(k)
1906	629,658	86,232	26,970	72,218	44,448	8,631	(b)	7,962	(k)
1907	592,989	91,120	29,539	70,432	45,898	8,808	(b)	6,666	(k)

(a) From *Accounts Relating to Trade and Navigation of the United Kingdom*. (b) Not reported. (c) Comprises porcelain and earthenware. (d) Including naphtha, paraffin, paraffin oil and petroleum. (e) Previous reports not available. (f) Includes puddled iron. (g) Includes railroad material of all kinds. (h) Includes all soda compounds; not separately enumerated previous to 1901. (i) Included under soda ash. (k) Included under spelter.

UNITED STATES

Of the following three tables, the first records the imports of foreign mineral and metal products into the United States, whether dutiable or duty free; the second shows the exports of materials produced in the United States; and the third reports the re-exports of products of foreign origin. The statement of production in the United States is given on an early page in this volume. These statistics are as reported by the Bureau of Statistics of the Department of Commerce and Finance, and special acknowledgment is due to Hon. O. P. Austin, chief of the bureau, for "furnishing the figures for many substances which are not reported in the Monthly Summary."

IMPORTS. (a)

Year.	Aluminum.					Ammonium Sulphate.			
	Crude.				Mfrd.				
	Lb.	Kg.	Value.	Value per Lb.	Value.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	256,559	116,374	$44,455	$0.172	$3,111	24,024,188	10,897	$ 591,937	$0.025
1901	564,803	251,657	104,168	.186	5,580	31,711,085	14,384	728,085	.023
1902	745,217	338,028	215,032	.290	3,645	35,535,558	16,119	858,036	.024
1903	498,655	226,190	139,298	.279	4,273	29,104,817	13,199	765,230	.026
1904	515,416	234,293	128,350	.249	478	39,859,690	18,077	1,058,981	.027
1905	530,429	240,284	106,108	.200	33	15,512,399	7,038	416,048	.027
1906	770,713	349,195	154,292	.200	1,865	31,797,291	14,423	894,663	.028
1907	872,474	395,754	181,352	.208	1,124	73,735,054	33,438	1,915,908	.026

Year.	Antimony.				Antimony Ore.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	3,632,843	1,648	$ 285,749	$0.079	6,035,734	2,738	$ 78,581	$0.013
1901	3,674,923	1,667	255,346	.069	1,731,756	786	24,256	.014
1902	5,742,703	2,605	347,899	.061	1,639,043	743	29,476	.018
1903	5,125,515	2,325	279,957	.054	2,673,142	1,213	51,489	.019
1904	4,056,299	1,840	235,401	.058	2,487,602	1,129	50,362	.020
1905	5,737,891	2,603	431,774	.075	1,976,694	897	52,868	.027
1906	7,900,194	3,583	1,417,816	.179	2,247,131	1,019	128,347	.057
1907	8,662,683	3,928	1,423,276	.164	2,760,186	1,252	180,903	.066

Year.	Asbestos.			Asphaltum.			
	Crude Value.	Mfd. Value.	Total Value.	Long Tons.	MetricTons.	Value.	Value per L. T.
1900	$ 631,796	$ 24,155	$ 655,951	113,557	115,374	$404,921	$3.57
1901	667 087	24,741	691,828	132,079	134,192	516,515	3.85
1902	729,421	33,013	762,434	139,944	142,183	439,570	3.09
1903	657,269	32,058	689,327	167,554	170,235	514,051	3 06
1904	700,572	51,290	751,862	119,575	121,489	510,524	4.27
1905	776,362	70,117	846,479	86,748	88,136	382,667	4.41
1906	536,500	....	....	97,274	98,830	388,010	3.93
1907	1,104,109	200,371	1,304,480	127,902	129,948	518,074	4.05

Year.	Arsenic. (b)				Barytes.				Bauxite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.....	(f)	.....	.....	.....	(f)	.....	.....	.....	8,656	8,795	$32,967	$3.81
1901.....	(f)	.....	.....	.....	(f)	.....	.....	.....	17,866	18,153	66,107	3.70
1902.....	(f)	.....	.....	.....	(f)	.....	.....	.....	15,790	16,043	54,410	3.45
1903.....	7,391,566	3,241	$256,097	$0.036	6,344	6,446	$22,777	$3.59	14,889	15,127	49,684	3.34
1904.....	6,391,566	2,900	226,481	0.036	6,689	6,796	27,463	4.11	15,475	15,723	49,577	3.20
1905.....	6,444,083	2,924	219,198	0.034	7,879	8,005	36,796	4.67	11,726	11,914	46,517	3.96
1906.....	7,639,507	3,464	336,609	0.044	4,293	4,362	37,296	8.69	17,809	18,094	63,221	3.55
1097.....	9,922,870	4,500	553,440	0.056	28,350	28,804	174,225	6.15	25,065	25,466	93,208	3.72

Year.	Brass and Mfres. of, Value.	Chloride of Lime.				Cement.			
		Lb.	Metric Tons.	Value.	Value per Lb.	Barrels. (c)	Metric Tons.	Value.	Value per Bbl.
1900......	$ 20,113	132,520,478	60,111	$1,524,205	$0.012	2,386,684	433,037	$3,330,453	$1.40
1901......	35,976	120,611,346	54,709	1,673,190	.014	944,892	170,431	1,305,692	1.38
1902......	51,626	112,374,478	50,973	1,456,435	.013	1,994,790	361,932	2,581,883	1.29
1903......	206,905	113,285,240	51,586	912,843	.008	2,317,951	420,569	3,027,111	1.30
1904......	310,311	87,909,168	39,876	707,174	.008	1,046,404	189,910	1,382,913	1.32
1905......	1,169,320	104,919,462	47,604	843,285	.008	846,577	153,644	1,102,041	1.30
1906......	1,891,126	105,221,371	47,718	863,490	.008	2,205,712	400,115	2,950,268	1.33
1907......	1,344,753	112,090,783	50,833	939,248	.008	2,006,228	363,929	2,637,424	1.31

Year.	Chrome Ore.				Bismuth.				Clays and Earths.			
	Long Tons.	Metric Tons.	Value.	Value per L.T	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900........	17,572	17,823	$305,001	$17.38	....	....	....	.....	146,524	148,868	$966,579	$6.59
1901........	20,112	20,434	363,108	18.04	....	....	....	.....	181,013	183 589	1,176,633	6.51
1902........	39,570	40,203	582,597	14.72	....	....	....	.....	191,764	194,832	1,228,945	6.41
1903........	22,932	23,299	302,025	13.17	147,324	66,826	$235,199	$1.59	203,173	206,424	1,264,544	6.22
1904........	24,227	24,615	348,527	14.38	147,712	67,002	268,837	1.82	191,853	194,923	1,168,552	6.09
1905........	54,434	55,305	725,301	13.32	148,589	67,459	318,007	2.14	321,641	326,787	1,363,760	4.24
1906........	43,441	44,136	557,594	12.84	254,733	115,000	318,452	1.25	260,481	236,256	1,586,938	6.09
1907........	41,999	42,671	491,925	11.71	259,881	117,882	325,015	1.25	283,335	287,868	2,021,371	7.13

Year.	Coal, Anthracite.				Coal, Bituminous.				Total Coal.	
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Value.
1900.....	118	120	$ 649	$4.65	1,909,258	1,939,806	$5,019,553	$2.63	1,909,366	5,020,102
1901.....	286	291	1,844	6.45	1,919,962	1,950,681	5,291,429	2.75	1,920,248	5,293,273
1902.....	73,006	74,174	323,517	4.43	2,478,375	2,518,029	7,012,674	2.84	2,551,381	7,339,791
1903.....	151,023	153,439	675,623	4.47	3,295,379	3,348,105	9,329,221	2.83	3,446,402	10,004,844
1904.....	72,526	73,686	220,665	3.04	1,556,149	1,581,047	3,915,613	2.52	1,628,675	4,136,278
1905.....	34,262	34,810	107,394	3.13	1,618,581	1,644,478	3,908,877	2.42	1,652,843	4,016,271
1906.....	32,357	32,875	105,190	3.25	1,712,150	1,739,544	4,129,555	2.41	1,744,507	4,234,745
1907.....	9,896	10,054	40,966	4.14	2,116,122	2,149,980	5,398,167	2.55	2,126,018	5,439,133

Year.	Coke.				Cobalt Oxide.				Copper, Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value pr L.T.	Lb.	Kg.	Value.	Value pr Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900..........	103,175	104,826	$371,341	$3.60	54,073	24,527	$88,651	$1.64	54,329	55,201	$ 5,195,010	$ 92.23
1901..........	172,729	173,893	266,078	3.67	71,969	32,645	134,208	1.86	96,047	97,584	14,692,645	152.99
1902..........	107,437	109,156	423,774	4.05	79,984	36,281	151,115	1.89	181,566	184,470	8,695,780	47.89
1903..........	127,479	129,519	437,625	3.43	73,350	33,272	145,264	1.98	284,912	289,471	3,177,582	11.15
1904..........	161,476	164,060	648,520	4.01	42,352	19,211	86,925	2.05	268,234	272,527	4,308,410	16.06
1905..........	181,376	184,278	796,544	4.39	70,048	31,802	139,377	1.99	296,251	300,991	5,765,238	19.46
1906..........	114,703	116,538	558,419	4.87	41,084	18,652	83,167	2.02	208,702	212,041	6,796,696	32.56
1907..........	131,400	133,502	596,366	4.54	42,794	19,421	73,028	1.71	292.417	297,096	9,158,343	31.32

Year.	Copper, Ingots, Old, etc.				Copper Mfres.	Total Copper.	Cryolite.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	68,796,808	31,206	$10,557,870	$0.153	$ 23,390	$15,776,270	5,437	5,524	$ 72,763	$13.38
1901	73,826,406	33,488	11,812,216	.160	24,775	26,529,636	5,383	5,469	70,886	13.17
1902	103,129,568	46,778	13,051,159	.126	52,464	21,799,403	6,188	6,287	85,640	13.83
1903	136,707,995	62,011	17,262,148	.126	31,624	20,471,354	7,708	7,831	102,879	13.35
1904	142,344,433	64,567	18,374,959	.129	37,913	22,721,282	959	974	13,708	14.30
1905	160,619,385	72,876	22,103,741	.137	117,404	22,221,145	1,600	1,623	22,482	14.05
1906	176,558,390	80,069	30,416,578	.172	62,191	30,478,769	1,505	1,529	29,683	19.72
1907	192,901,267	87,481	38,658,754	.200	131,580	38,790,334	1,438	1,461	28,920	20.10

Year.	Emery Grains.				Emery Rock.				Emery Mfres.	Total Emery.
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.		
1900	661,482	300	$26,520	$0.040	11,392	11,574	$202,980	$17.82	10,006	$239,506
1901	1,116,729	506	43,207	.039	12,441	12,640	240,856	19.35	10,927	294,990
1902	1,665,737	756	60,079	.036	7,166	7,281	151,959	21.21	13,776	225,814
1903	3,595,239	1,630	109,272	.030	10,885	11,059	188,985	17.36	23,317	321,574
1904	2,281,193	1,035	109,772	.048	7,054	7,167	131,493	18.64	19,059	260,324
1905	3,209,915	1,456	143,729	.045	11,073	11,250	185,689	16.77	17,996	347,414
1906	4,655,168	2,113	215,357	.043	13,840	14,061	286,386	20.69	19,105	521,848
1907	4,282,228	1,942	186,156	.043	11,235	11,415	211,184	18,80	15,282	412,622

Year.	Fertilizers.									Fuller's Earth.	
	Guano.				Crude Phosphates.				All Other.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.		Long Tons.	Value.
1900	6,620	6,726	$67,413	$10.18	137,086	139,272	$791,189	$5.77	$1,400,336	....	....
1901	4,949	5,028	71,140	14.37	175,765	178,577	872,503	4.97	1,506,965	....	....
1902	8,407	8,542	164,783	19.60	137,386	139,584	646,264	4.70	1,725,333	....	....
1903	21,007	21,343	251,966	12.00	132,965	134,092	679,112	5.11	2,353,496	15,267	$120,671
1904	35,876	36,430	478,388	13.33	130,214	132,297	745,744	5.73	2,856,141	9,126	78,006
1905	25,651	26,061	365,823	14.26	56,021	56,917	273,289	4.88	4,051,003	13,001	105,997
1906	22,947	22,314	320,565	13.97	23,281	23,653	147,547	6.34	4,231,723	13,238	108,696
1907	29,141	29,607	365,357	12.54	25,876	26,290	163,944	6.34	4,994,346	14,648	122,221

Year.	Gold.		Iron Ore.				Pig Iron.			
	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	$45,703,256	$21,045,828	879,831	893,908	$1,303,196	$1.48	52,565	53,406	$1,907,361	$36.28
1901	33,237,629	21,524,251	966,950	982,421	1,659,273	1.72	62,930	63,937	1,792,014	28.48
1902	22,710,957	21,482,360	1,165,470	1,184,118	2,583,077	2.22	619,354	629,264	10,935,831	17.66
1903	44,054,902	21,212,794	980,440	996,127	2,261,008	2.31	599,574	609,167	11,173,302	18.64
1904	75,646,128	9,157,106	487,613	495,415	1,101,384	2.26	79,590	80,772	1,765,107	22.20
1905	38,564,328	11,729,077	845,651	859,181	2,062,161	2.44	212,465	215,864	5,185,784	24.41
1906	139,705,887	15,873,493	1,060,390	1,077,356	2,967,434	2.80	379,828	385,905	11,851,210	31.20
1907	130,605,413	12,792,659	1,229,168	1,248,835	3,937,483	3.20	489,440	497,271	13,417,276	27.41

Year.	Scrap, Iron and Steel.				Bar Iron.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	34,431	34,982	$663,231	$19.26	19,685	19,094	$1,058,761	$56.34
1901	20,130	20,452	339,827	16.88	20,792	21,126	1,093,736	52.60
1902	109,510	111,262	1,606,720	14.67	28,844	29,307	1,286,238	44.58
1903	82,921	84,248	1,273,941	15.36	43,392	44,090	1,904,469	43.89
1904	13,461	13,676	189,506	14.08	20,905	21,247	917,254	43.88
1905	23,731	24,111	370,328	15.61	37,294	37,891	1,522,434	40.82
1906	19,091	19,397	248,106	13.00	35,793	36,366	1,590,592	44.44
1907	27,687	28,130	370,553	13.38	39,746	40,382	1,774,441	44.64

Year.	Rails.				Hoop, Band or Scroll.				Ingots, Blooms, Slabs, Billets, etc.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	1,448	1,471	$56,129	$38.77	165	167	$12,409	$75.26	12,709	12,913	$1,332,896	$104.84
1901	1,905	1,935	67,052	35.19	2,974	3,021	116,841	39.29	8,164	8,295	2,340,112	164.15
1902	63,522	64,538	1,576,679	24.82	3,362	3,416	131,052	38.97	289,318	293,965	7,943,818	27.76
1903	95,555	97,083	2,159,273	22.59	1,525	1,550	74,898	49.11	261,570	265,932	7,331,299	28.03
1904	37,776	38,380	808,775	21.41	2,135	2,169	60,934	28.54	(m) 10,807	10,980	1,537,531	142.27
1905	17,278	17,554	409,807	23.72	4,772	4,848	137,612	28.84	(m) 14,641	14,875	2 072,606	141.56
1906	4,943	5,022	137,104	27.74	10,231	10,395	256,836	25.10	(m) 21,337	21,678	3,010,589	141.10
1907	3,752	3,812	104,958	27.97	1,508	1,532	82,706	54.84	(m) 19,334	19,643	3,004,178	155.38

Year.	Sheet, Plate and Taggers Iron or Steel.				Tin Plates, Terne Plates and Taggers Tin.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	5,143	5,226	$426,541	$82.93	60,386	61,356	$4,617,813	$76.46
1901	5,626	5,716	443,880	79.10	77,395	78,638	5,294,789	68.41
1902	7,156	7,270	545,739	76.26	60,115	61,080	4,023,421	66.93
1903	11,557	11,741	540,272	46.75	47,360	48,118	2,999,252	63.33
1904	4,165	4,232	302,500	72.63	70,652	71,782	4,354,761	61.63
1905	2,236	2,272	242,955	108.66	65,740	66,792	4,090,523	62.22
1906	3,231	3,283	325,276	67.35	56,982	57,894	3,883,225	68.15
1907	3,749	3,809	367,140	97.93	57,773	58,697	4,462,522	77.24

Year.	Wire-Rods.				Wire and Articles Made from.				Total Iron Imports. (e)
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	
1900	21,092	21,430	$1,212,594	$57.49	1,848	1,877	$409,087	$221.37	$20,443,911
1901	16,804	17,073	964,744	57.40	4,129	4,192	585,354	141.77	20,404,122
1902	21,382	21,725	1,033,074	48.31	3,469	3,525	606,724	174.90	41,468,826
1903	20,836	21,169	1,028,977	49.39	5,018	5,098	728,430	145.16	41,258,864
1904	15,313	15,558	707,779	46.22	3,956	4,019	624,892	157.96	21,621,970
1905	17,616	17,898	800,027	45.41	3,978	4,042	705,465	177.34	26,401,283
1906	17,799	18,084	876,270	49.23	6,610	6,716	1,079,868	163.37	34,827,132
1907	17,076	17,349	851,571	49.87	....	....	1,551,415	....	38,789,992

Year.	Lead in Ore and Base Bullion.				Lead in Pigs and Old.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Short Tons.	Metric Tons.	Value.	Value per Sh. T.
1900......	(g)114,397	103,780	$3,975,695	$34.60	....	....	....	....
1901......	111,867	101,486	4,807,762	47.37	604	548	$33,882	$56.10
1902......	105,186	95,425	4,424,511	46.37	2,529	2,294	132,500	52.40
1903......	103,384	93,790	3,596,635	38.35	3,023	2,742	164,528	54.42
1904......	104,127	94,464	3,517,691	37.24	8,724	7,914	461,316	52.88
1905......	92,657	84,081	3,565,282	38.48	5,720	5,812	367.106	64.18
1906......	72,371	65,640	3,490,750	48.23	11,763	10,669	910,417	77.39
1907......	70,538	64,019	3,579,990	50.72	9,277	8,414	846,166	91.21

Year.	Lead, Sheet, Pipe, Shot, Etc.				Other Lead Mfrs.	Total Lead.	White Lead.			
	Lb.	Metric Tons.	Value.	Value per Lb.			Lb.	Metric Tons.	Value.	Value per Lb.
1900.....	27,945	13	$1,393	$0.050	$5,854	$3,964,942	456,872	207	$28,366	$0.062
1901....	56,735	26	2,773	.048	4,654	4,849,071	384,673	174	21,226	.056
1902. ...	224,208	102	7,765	.034	18,918	4,533,694	506,423	230	25,320	.050
1903. ...	17,008	8	810	.048	8,071	3,770,044	453,284	206	24,595	.054
1904. ...	69,581	32	2,441	.035	7,755	3,989,203	587,338	266	33,788	.058
1905. ...	54,779	25	2,638	.048	4,580	3,939,606	597,510	271	34,722	.058
1906. ...	346,177	157	17,250	.050	20,681	4,439,098	647,636	294	41,233	.064
1907. ...	734,418	333	39,210	.053	12,736	4,478,102	584,309	265	37,482	.064

Yr.	Litharge.				Red Lead.				Orange Mineral.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	77,314	35	$2,852	$0.032	549,551	249	$25,532	$0.046	1,068,793	485	$61,885	$0.059
1901	49,306	22	1,873	.038	485,467	220	19,370	.040	977,644	443	52,409	.053
1902	88,115	40	2,908	.033	1,075,839	488	37,383	.035	997,494	452	49,060	.049
1903	42,756	19	1,464	.034	1,152,715	523	40,846	.035	756,742	343	36,407	.048
1904	44,541	20	1,500	.034	836,077	379	30,115	.036	766,469	348	37,178	.049
1905	117,757	53	4,139	.035	704,402	320	26,553	.038	628,003	285	31,106	.049
1906	87,230	40	3,737	.043	1,093,619	497	50,741	.046	770,342	350	42,519	.055
1907	90,475	41	4,356	.048	679,171	308	35,959	.053	615,015	279	37,793	.061

Year.	Magnesite.				Manganese Ore.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value	Value per L. T.
1900....................	....	....	....	....	256,252	260,352	$2,042,361	$7.97
1901....................	30,350	30,835	....	....	165,720	168,372	1,486,573	8.97
1902....................	45,157	45,880	$373,928	8.28	235,576	239,345	1,931,282	8.20
1903....................	49,684	50,479	461,399	9.29	146,056	148,393	1,278,108	8.75
1904....................	35,106	35,668	286,828	8.17	108,519	110,255	901,592	8.31
1905....................	66,405	67,566	638,619	9.46	257,033	261,146	1,952,407	7.60
1906....................	80,711	82,002	863,492	10.70	221,260	224,800	1,696,043	7.67
1907....................	88,400	89,814	875,359	9.90	208,321	211,654	1,793,143	8.61

Year.	Oil, Mineral.				Paints and Colors.	Platinum, Unmanufactured.				Platinum Mfres.
	Gal.	Liters.	Value.	Value per Gal.		Lb. Troy.	Kg.	Value.	Value per Lb. Troy.	
1900...	3,039,094	11,503,913	$274,766	$0.091	$1,491,902	9,246	3,450	$1,728,777	$187.00	$36,714
1901...	2,294,684	8,686,389	151,913	.066	1,524,125	7,496	2,797	1,673,713	223.30	24,482
1902...	3,578,393	13,545,646	207,310	.058	1,745,989	8,670	3,235	1,950,362	224.96	37,618
1903...	4,266,974	16,152,203	261,199	.061	1,811,902	9,540	3,561	1,921,772	201.44	1,727,830
1904...	4,846,681	18,344,650	277,399	.057	1,526,072	8,648	3,230	1,812,242	209.55	105,636
1905...	13,725,720	52,020,479	672,127	.049	1,570,839	8,681	3,240	1,985,107	228.67	188,156
1906...	21,045,316	79,654,921	1,061,076	.050	1,833,938	13,928	5,198	3,601,021	258.54	187,639
1907...	29,553,234	111,858,991	1,615,011	.055	2,107,122	7,515	2,805	2,509,926	333.99	175,651

Year.	Marble and Stone Mfres.		Metal Composition:		Mica.	Nickel. (*h*)	Nickel Ore and Matte.			
	Marble.	Stone. (*d*)	Bronze.	All Other.			Long Tons.	Metric Tons.	Value.	Value per L. T.
1900..	$945,705	$256,624	$791,306	$5,420,483	$ 319,560	(*f*)	(*f*)	......	....	....
1901..	1,226,524	237,191	945,702	5,162,392	335,054	....	(*f*)	......	....	....
1902..	1,435,457	222,435	816,668	5,942,453	466,332	....	(*f*)	......	....	....
1903..	1,502,111	258,968	790,639	6,402,215	317,969	$207,954	11,936	16,191	$1,285,935	$80.70
1904..	1,270,443	294,035	746,073	5,965,247	269,808	206,021	8,549	8,685	915,470	107.10
1905..	1,348,624	293,055	839,133	7,314,809	403,755	335,211	13,451	13,666	1,626,920	120.95
1906..	1,370,413	355,908	991,182	9,283,836	1,042,608	86,336	15,156	15,398	1,816,631	119.86
1907..	1,452,625	357,795	1,048,924	9,134,952	915,259	90,153	16,888	17,158	2,153,873	121.03

Year	Potassium Salts.											
	Chlorate.			Chloride.			Chromate and Bichromate.			Nitrate.		
	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.
		$	$		$	$		$	$		$	$
1900	1,243,612	68,772	0.055	130,175,481	1,976,604	0.015	111,761	7,758	0.069	10,545,392	276,664	0.026
1901	811,127	61,348	.076	148,189,337	2,316,577	.015	430,996	29,224	.068	9 656,393	253,286	.026
1902	1,209,148	60,429	.050	140,980,460	2,141,553	.015	231,009	15,161	.066	10,505,474	299,416	.028
1903	468,042	19,308	.041	169,337,673	2,550,478	.015	41,229	2,784	.067	13.835,668	367,721	.026
1904	95,889	4,209	.044	174,865,872	2,832,554	.016	26,053	1,817	.069	14,184,287	376,931	.027
1905	42,510	2,876	.067	214,207,064	3,326,748	.016	59,650	4,225	.070	9,911,534	304,596	.027
1906	45,873	3,103	.068	223,203,387	3,858,895	.017	30,098	2,102	.080	11,326,256	371,595	.033
1907	12,980	959	.074	252,303,441	4,175,353	.017	18,171	1,307	.072	20,419,770	612,894	.030

Year.	Potassium Salts. All Other.			Precious Stones.			Pyrites. (*i*)			
	Lb.	Value.	Value per Lb.	Uncut.	Cut, not Set.	Jewelry.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	54,904,088	$1,407,303	$0.025	$3,751,219	$9,612,127	(*f*)	332,517	337,837	$1,095,598	$3.30
1901...	72,489,913	1,636,856	.022	6,637,860	17,166,049	(*f*)	398,969	405,353	1,407,244	3.53
1902...	91,857,009	1,820,585	.020	8,282,760	18,494,288	(*f*)	437,319	444,316	1,623,430	3.71
1903...	70,205,850	1,593,380	.023	10,374,877	15,428,819	$954,456	427,319	434,156	1,636,450	3.83
1904...	74,720,241	1,678,699	.023	10,316,615	16,934,090	803,952	413,585	420,202	1,533,564	3.73
1905...	82,935,632	1,891,081	.023	10,206,350	26,699,670	801,566	515,722	520,926	1,780,800	3.47
1906...	30,302,735	763,513	.025	11,937,542	32,201,949	988,766	598,078	607,647	2,148,559	3.59
1907...	91,299,496	2,220,685	.024	8,892,643	23,706,975	1,069,373	627,985	638,033	2,581,787	4.11

Year.	Salt.				Silver.		Sodium Nitrate.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	207,933	188,636	$633,192	$3.05	$14,695,965	$25,404,378	182,108	185,022	$4,935,520	$27.12
1901	194,967	176,872	670,648	3.44	12,957,987	18,188,795	208,654	211,992	5,997,595	28.82
1902	188,775	167,481	654,990	3.47	8,502,614	17.900,321	205,245	208,529	5,996,205	29.21
1903	157,201	142,494	489,179	3.11	7,935,844	16,038,664	272,947	277,314	8,700,806	31.88
1904	167,295	151,810	515,822	3.08	11,865,805	14,221,237	228,012	231,660	9,333,613	32.41
1905	158,449	143,783	491,079	3.10	16,472,911	19,466,224	321,231	326,371	11,206,548	34.89
1906	170,505	154,648	502,583	2.95	20,402,738	23,825,103	372,222	378,178	14,115,206	37.92
1907	153,435	139,166	452,227	2.95	17,652,679	28,259,681	364,610	370,444	14,844,675	40.71

Year.	Sodium Hydroxide (Caustic).			Soda Ash and Carbonate.			All Other Sodium Salts.		
	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.
1900	8,403,749	$150,530	$0.018	73,815,425	$613,379	$0.008	20,484,938	$259,802	$.013
1901	3,812,847	94,303	.025	31,415,788	276,261	.009	14,491,559	189,543	.013
1902	3,334,697	77,482	.020	31,889,252	284,634	.009	17,151,682	283,745	.016
1903	2,970,426	73,647	.025	25,313,370	228,041	.009	14,272,646	268,738	.019
1904	2,570,984	64,405	.025	23,631,832	205,496	.009	10,399,711	281,527	.027
1905	2,245,789	56,515	.025	15,754,979	146,812	.009	11,257,629	247,413	.022
1906	1,209,053	35,262	.022	6,800,055	70,979	.010	....	....	....
1907	1,297,070	37,894	.029	6,198,136	66,521	.011	8,481,979	258,262	.030

Year.	Sulphur.											
	Crude.				Flowers.				Refined.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	166,457	169,120	$2,918,610	$17.53	628	638	$17,437	$27.77	243	247	$ 6,279	$25.84
1901	174,162	176,949	3,256,951	18.70	748	761	20,201	26.98	268	272	6,308	23.57
1902	176,951	179,782	3,360,562	19.00	738	750	19,954	27.04	14	15	369	24.99
1903	188,888	191,910	1,649,756	19.32	1,854	1,883	52,680	28.42	189	192	7,254	38.44
1904	128,885	130,947	2,463,779	19.12	1,332	1,353	39,133	29.38	204	207	9,776	47.92
1905	83,201	84,532	1,522,005	18.29	572	581	16,037	28.04	778	790	19,960	25.66
1906	72,404	73,562	1,282,873	17.72	1,100	1,118	29,565	26.87	709	720	17,928	25.29
1907	20,399	20,725	355,944	17.45	1,458	1,481	41,216	28.27	606	616	14,589	24.07

Year.	Talc.				Tin.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	79	72	$ 1,070	$13.54	69,989,502	31,747	$19,458,586	$0.278
1901	2,386	2,164	27,015	11.74	74,560,487	33,820	19,024,761	.255
1902	2,859	2,594	35,336	12.35	85,043,353	38,575	21,263,337	.250
1903	1,790	1,623	19,635	11.00	83,133,847	37,702	22,265,367	.268
1904	3,268	2,964	36,370	11.13	83,168,657	37,718	22,356,896	.270
1905	4,000	3,630	48,225	12.06	89,227,698	40,507	26,316,023	.294
1906	5,643	5,118	67,818	12.02	101,027,188	45,816	37,446,508	.371
1907	10,060	10,221	126,391	12.56	82,548,838	37,436	32,075,091	.389

Year.	Zinc. Blocks, Pigs and Old. Lb.	Metric Tons.	Value.	Value per Lb.	Oxide (j). Lb.	Value.	Sulphide. Lb.	Value.	Mfrcs. Value.
1900......	2,013,196	913	$97,772	$0.048	2,657,514	....	....	....	$36,836
1901......	775,881	352	30,920	.040	3,327,976	....	....	....	42,543
1902......	1,238,091	561	46,713	.038	3,434,466	....	1,247,936	$32,879	37,191
1903......	728,614	330	30,900	.042	3,653,076	$188,495	1,229,806	33,077	18,938
1904......	933,474	423	44,455	.048	2,809,905	165,110	1,228,875	31,382	11,918
1905......	1,042,081	473	51,052	.048	3,779,311	196,220	1,235,360	33,308	12,390
1906......	4,042,486	1,835	240,632	.060	4,494,014	288,065	1,286,469	40,112	17,385
1907......	3,418,704	1,550	220,474	.064	5,311,318	323,551	1,570,073	51,435	16,282

(a) From Summary of Commerce and Finance of the United States. (b) Includes arsenic sulphide. (c) Barrels of 400 lb. (d) Including slate. (e) Not including iron ore. (f) Not reported. (g) Includes pig and old. (h) Includes nickel oxide, alloys in which nickel is the principal constituent and manufactures of nickel. (i) Containing more than 25 per cent sulphur. (j) Includes white pigments containing zinc but not lead, dry and in oil. (m) Includes bar of steel and steel forms not elsewhere specified. The high value is due to the value of "high-speed" steel.

EXPORTS OF DOMESTIC PRODUCTS. (a)

Year.	Aluminum and Mfres. of.	Asbestos and Mfres. of.	Brass and Mfres. of.	Brick.	Cement. Bbl. (i)	Metric Tons.	Value.	Value per Bbl.	Chemicals and Drugs.
1900.....	$281,821	$124,971	$2,068,072	....	100,400	18,216	$ 225,306	$2.24	$13,771,682
1901.....	183,579	113,316	2,078,178	$541,589	373,934	67,393	679,296	1.82	14,267,110
1902.....	116,052	130,437	1,809,312	501,434	340,821	61,838	526,471	1.54	13,437,367
1903.....	157,187	158,360	2,063,569	439,277	285,463	51,748	433,984	1.52	14,276,465
1904.....	166,876	223,096	3,093,803	587,385	774,940	140,898	1,104,086	1.42	14,029,583
1905.....	290,777	....	3,055,189	799,878	1,016,236	185,345	1,387,706	1.37	17,126,967
1906.....	364,251	259,760	4,180,403	885,066	583,299	105,811	944,886	1.62	18,509,077
1907.....	304,938	200,371	4,069,965	816,971	900,550	163,360	1,450,841	1.61	20,963,865

Year.	Coal. Anthracite. Long Tons.	Metric Tons.	Value.	Value per L. T.	Bituminous. Long Tons.	Metric Tons.	Value.	Value per L. T.	Coke. Long Tons.	Value.
1900.	1,654,610	1,681,084	$7,092,489	$4.29	6,262,909	2,363,631	$14,431,590	$2.31	376,999	$1,358,968
1901.	1,993,307	2,025,200	8,937,147	4.48	5,390,086	3,476,327	13,085,763	2.53	384,330	1,516,898
1902.	907,977	922,505	4,301,946	4.73	5,218,969	5,302,472	13,927,063	2.66	392,491	1,785,188
1903.	2,008,857	2,040,999	9,780,044	4.86	6,303,241	6,404,093	17,410,385	2.76	416,385	2,091,875
1904.	2,228,392	2,264,046	11,077,570	4.97	6,345,126	6,446,648	17,160,538	2.74	523,090	2,311,401
1905.	2,229,983	2,265,663	11,104,654	4.98	6,959,265	7,070,613	17,867,964	2.56	599,054	2,243,010
1906.	2,216,969	2,252,441	10,896,200	4.91	(k) 7,704,850	7,828,128	(k)19,787,459	2.57	765,190	2,753,551
1907.	2,698,072	2,741,241	13,217,985	4.90	(m)10,448,676	10,615,855	(m)26,972,908	2.58	874,689	3,206,793

Year.	Copper. In Ore and Matte (b). Long Tons.	Metric Tons.	Value.	Value per L. T.	Ingots, Bars, Plates and Old. Lb.	Metric Tons.	Value.	Value per Lb.	Mfrcs. Value.	Total Except Ore.
1900...	10,007	10,168	$1,332,829	$133.18	337,973,751	153,304	$55,285,047	$0.164	$2,257,563	$57,542,610
1901...	19,613	19,924	2,536,549	129.40	194,249,828	88,111	31,692,563	.164	1,842,336	33,384,899
1902...	18,035	18,321	1,326,131	73.53	354,668,849	160,876	43,392,800	.122	2,092,798	45,485,598
1903	12,291	12,488	855,367	69.59	310,729,524	140,920	41,170,059	.132	2,339,729	43,509,788
1904...	18,927	19,230	1,202,537	63.54	554,550,030	251,497	71,488,116	.129	3,328,818	74,816,934
1905...	37,688	38,291	1,531,429	40.63	534,907,619	242,699	80,693,232	.151	4,184,070	84,890,302
1906...	47,619	48,380	1,760,140	36.96	454,752,018	206,239	84,728,400	.186	4,284,611	89,013,011
1907...	99,141	100,727	2,452,562	24.74	508,929,401	230,799	94,912,185	.186	5,888,170	100,800,355

Year	Earthen and China Ware.	Fertilizers. Crude Phosphates. Long Tons.	Metric Tons.	Value.	Value per L. T.	All Other. Long Tons.	Metric Tons.	Value.	Value per L. T.	Glassware.	Gold. In Coin and Bullion. (c)	In Ore. (d)
1900	$558,794	619,995	629,915	$5,217,560	$8.38	25,976	26,392	$537,908	$20.71	$2,042,633	$54,064,697	$69,926
1901	526,820	729,539	741,212	5,839,245	8.01	14,153	14,379	332,964	23.54	2,087,043	56,717,350	1,012,589
1902	604,646	802,086	814,919	6,193,372	7.73	16,451	16,714	383,438	23.31	2,094,701	35,722,835	307,756
1903	589,001	785,259	797,823	6,109,230	7.78	20,343	20,668	557,059	27.38	2,053,516	43,765,360	581,474
1904	791,739	842,484	855,964	6,521,555	7.74	25,549	25,958	714,367	27.96	2,130,297	120,226,424	985,403
1905	983,554	934,940	949,899	7,465,592	7.91	22 865	23,231	723,744	32.05	2,252 399	46,099,580	694,887
1906	1,118,450	904,214	918,681	7,373,945	8.16	31,999	32,511	1,088,004	34.00	2,534,271	46,068,451	640,707
1907	1,131,641	1,018,212	1,034,503	8,387,176	8.24	45,000	45,720	1,641,316	36.47	2,815,150	54,869,688	345,993

Year	Gunpowder and Other Explosives.	Iron. Ore. Long Tons.	Metric Tons.	Value.	Value per L. T.	Pig. Long Tons.	Metric Tons.	Value.	Value Per L. T.
1900........	....	51,460	52,283	$154,756	$3.01	286,687	291,404	$4,654,582	$16.23
1901........	$1,965,875	64,703	65,748	163,465	2.54	81,211	82,510	1,257,699	15.65
1902........	2,393,480	88,445	89,860	294,168	3.32	27,487	27,927	502,947	18.30
1903........	2,367,148	80,611	81,901	255,728	3.17	20,379	20,705	384,334	18.86
1904........	2,466,278	213,865	217,287	458,823	2.14	49,025	49,809	764,543	15.60
1905........	2,848,155	208,017	211,345	530,457	2.55	49,221	50,009	762,899	15.50
1906........	3,793,043	265,240	269,484	771,831	2.91	83,317	84,650	1,506,774	18.08
1907........	4,345,357	278,208	282,659	763,422	2.74	73,844	75,026	1,508,941	20.43

Year.	Iron, Bar. Long Tons.	Metric Tons.	Value.	Value per L. T.	Iron, Band, Hoop and Scroll. Long Tons.	Metric Tons.	Value.	Value per L. T.	Billets, Ingots and Blooms. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900...	13,298	13,512	$558,576	$42.04	2,976	3,024	$137,437	$46.20	107,385	109,103	$2,915,371	$27.15
1901...	17,708	17,993	674,671	38.16	1,561	1,586	74,056	47.44	28,614	29,072	708,887	24.78
1902...	22,249	22,605	869,519	39.08	1,674	1,701	82,322	49.18	2,409	2,447	74,938	31.11
1903...	19,380	19,690	796,631	41.11	1,241	2,175	101,839	47.56	5,445	5,532	141,924	26.07
1904...	29,582	30,055	1,133,128	34.93	3,435	3,489	162,039	47.18	314,324	319,353	6,150,035	19.56
1905...	32,025	32,537	1,255,418	39.20	4,426	4,497	182,431	41.22	237,738	241,542	4,701,909	19.79
1906...	56,024	56,920	2,575,905	45.98	5,405	5,491	242,776	44.92	192,616	195,698	4,094,659	21.26
1907...	24,190	24,577	1,092,631	45.17	8,587	8,724	395,758	46.09	79,991	81,271	1,983,319	24.79

Year.	Iron, Nails and Spikes, Cut. Lb.	Metric Tons.	Value.	Value per Lb.	Iron, Nails and Spikes, All Other. Lb.	Metric Tons.	Value.	Value per Lb.	Iron, Plates and Sheets. Long Tons.	Met. Tons.	Value.	Value per L. T.
1900	25,005,308	11,342	$626,497	$0.025	65,444,387	29,681	$1,816,813	$0.028	9,331	9,481	$600,600	$64.35
1901	20,835,944	9,452	450,331	.021	46,298,262	21,001	1,152 368	.025	6,909	7,020	452,695	65.52
1902	16,122,775	7,312	339,227	.021	64,565,650	29,287	1,456,768	.022	3,434	3,489	229,887	66.94
1903	19,912,563	9,031	424,985	.021	75,654,532	34,310	1,698,500	.024	4,782	4,858	273,618	57.22
1904	20,772,049	9,422	416,389	.020	80,279,746	36 403	1,949,908	.024	4,728	4,804	247,694	52.39
1905	17,674,099	8,019	352,405	.020	89,976,088	40,506	2,118,836	.024	8,004	8,132	460,995	57.60
1906	16,951,893	7,688	340,526	.020	116,310,428	52,747	2,731,021	.024	17,054	17,327	1,139,526	66.81
1907	15,621,208	7,071	354,802	.023	111,670,147	50,642	3,014,863	.027	40,651	41,301	2.902.025	71.39

Year.	Steel, Sheets and Plates. Long Tons.	Metric Tons.	Value.	Value per L. T.	Iron Rails. Long Tons.	Metric Tons.	Value.	Value per L. T.	Steel Rails. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.	45,534	46,264	$1,638,478	$35.98	5,374	5,460	$119,206	$22.18	356,445	361,945	$10,895,416	$30.58
1901.	23,923	24,303	959,471	40.11	901	915	32,357	35.93	318,055	323,044	8,628,781	27.14
1902.	14,866	15,104	725,547	48.80	211	214	4,639	22.02	67,455	68,534	1,902,396	28.09
1903.	13,312	13,525	657,713	49.47	181	184	8,808	48.67	30,656	31,146	937,779	30.59
1904.	50,477	51,278	2,064,241	40.89	1,405	1,427	23,870	17.00	414,845	421,482	10,661,222	25.72
1905.	67,093	68,166	2,889,084	43.06	*Nil.*	....	....	....	295,023	299,473	7,310,029	24.78
1906.	93,601	95,099	4,081,915	43.61	*Nil.*	....	....	....	328,036	333,285	8,903,411	27.14
1907.	82,045	83,358	4,262,582	51.95	*Nil.*	....	....	....	338,906	344,328	10,411,072	30.72

Year	Structural Iron and Steel. Long Tons.	Metric Tons.	Value.	Value per L. T.	Wire. Long Tons.	Metric Tons.	Value.	Value per L. T.	Steel Wire Rods. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.	67,714	68,797	$3,570,769	$52.73	78,014	79,262	$4,604,047	$59.77	10,652	10,822	$505,529	$47.37
1901.	54,005	54,869	3,031,861	56.10	88,238	89,650	4,805,608	54.36	8,165	8,296	271,552	33.26
1902.	53,859	54,721	2,828,460	52.52	97,843	99,414	5,140,702	52.54	24,613	25,007	831,067	33.76
1903.	30,641	31,131	1,788,556	58.37	108,521	110,258	5,528,726	50.94	22,360	22,718	713,718	31.92
1904.	55,514	56 402	2,777,768	50.04	118,581	120,478	5,935,093	50.05	20,073	20.394	695,448	34.64
1905.	84,234	85,582	4,357,186	51.73	142,601	144,883	7,061,442	49.52	6,514	6,618	277,651	42.62
1906.	112,555	114,356	6,140,861	54.56	174.014	176,798	8,770,042	50.40	5,896	5,990	221,679	37.60
1907.	138,442	140,657	7,784,618	56.23	161,228	163,808	9,164,829	56.84	10,653	10,823	465,757	43.72

Year.	Lead and Mfres. of.	Marble, Stone.	Nickel. (*e*)	Petroleum products. (In Thousands of Units.)* Crude. M Gals.	M Liters.	M Value.	Value per Gal.	Naphtha. M Gals.	M Liters.	M Value.	Value per Gal.
1900.........	$459,574	$1,556,981	$1,382,727	138,161	523,000	$7,341	$0.053	18,570	70,295	$1,681	$0.081
1901.........	625,234	1,785,515	1,521,291	127,008	480,781	6,038	.050	21,685	82,087	1,742	.079
1902.........	696,010	1,587,957	924,579	145 234	549,775	6,331	.042	19,683	74,509	1,393	.071
1903.........	491,362	1,688,316	703,550	126,512	478,847	6,782	.054	12,973	49,103	1,519	.118
1904.........	616,126	1,337,754	2,130,933	111,176	420,801	6,351	.057	24,989	94,583	2,322	.093
1905.........	511,699	1,384,208	2,894.700	126,185	447,610	6,086	.048	28,420	107,570	2,215	.078
1906.........	600.057	1,391,667	3,493,643	148,045	560,350	7.731	.052	27,545	104.258	2.488	.090
1907.........	[illegible],096	1,309,986	2,845,663	126,302	478,053	6,334	.050	34,635	131,093	3,676	.106

Year.	Petroleum Products. (In Thousands of Units.)* Illuminating Oil. M Gals.	M Liters.	M Value.	Value per Gal.	Lubricating Oil. M Gals.	M Liters.	M Value.	Value per Gal.	Residue, Etc. (*g*) M Gals.	M Liters.	M Value.	Value per Gal.
1900...	739,163	3,104,593	$54,693	$0.074	71,211	269,540	$9,933	$0.139	19,750	74,760	$845	$0.042
1901...	827,479	3,131,399	53,491	.065	75,306	285,010	10,260	.136	27,596	104,463	1,255	.046
1902...	778,801	2,947,762	49,079	.063	82,201	311,163	10,872	.133	38,316	145,043	922	.024
1903...	691,837	2,618,603	51,356	.074	95,622	361,929	12,690	.133	9,753	36,916	282	.029
1904...	761,358	2,881,740	58,384	.077	89,738	339,469	12,389	.138	34,904	132,072	1,174	.034
1905...	881,450	3.336,288	54,901	.062	113,730	430,468	14,312	.126	70,728	267,705	2,128	.030
1906...	878,284	3.324,305	54,858	.063	151,269	572,553	18,690	.124	64,645	244,681	1,971	.030
1907...	905,924	3,428,922	59,635	.066	152,028	575,426	19,210	.126	75,775	286,808	2,528	.033

Year.	Petroleum Products.* Paraffin.				Quicksilver.				Silver.	
	M Lb.	M Metric Tons.	M Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	In Coin and Bullion (*c*)	In Ore (*d*)
1900........	157,108	71.2	$8,186	$0.052	778,191	353	$425,812	$0.547	$65,705,909	$515,755
1901........	151,694	68.8	7,960	.052	843,938	383	475,609	.563	55,526,975	111,383
1902........	175,269	79.5	8,398	.048	1,013,434	459	575,099	.568	49,228,303	44,651
1903........	204,120	92.6	9,596	.047	1,344,615	610	719,119	.535	40,531,095	79,247
1904........	174,582	79.2	8,273	.047	1,611,365	731	847,108	.526	49,975,370	159,875
1905........	160,836	73.0	7,873	.049	1,009,446	458	497,470	.493	54,133,721	3,379,381
1906........	173,504	72.9	8,463	.049	484,151	219	243,914	.504	57,012,104	266,674
1907........	207,504	94.1	10,209	.049	384,913	174	192,094	.500	61,202,024	423,842

Year.	Tin Mfres.	Zinc Ore.				Zinc Pigs, Bars, Plates and Sheet.			
		Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900............	$ 467,354	37,555	38,158	$1,133,633	$30.19	44,802,577	20,322	$2,217,693	$0.050
1901............	495,435	39,425	40,056	1,167,684	29.62	6,780,221	3,071	288,906	.043
1902............	529,061	49,762	50,558	1,449,104	29.12	6,473,135	2,936	300,557	.046
1903............	777,917	35,188	35,751	987,000	28.05	3,041,911	1,380	163,379	.053
1904............	701,625	32,063	32,576	905,782	28.25	20,145,942	9,204	1,094,490	.053
1905............	930,844	27,630	28,072	848,451	30.71	11,031,815	5,005	682,254	.062
1906............	1,120,410	24,750	25,146	733,300	29.63	9,340,455	4.236	583,526	.062
1907............	1,079,291	18,171	18,462	579,490	31.89	1,126,753	511	75,194	.067

Year.	Zinc Oxide.				Zinc Mfres.	Total (Except Ore).
	Lb.	Metric Tons.	Value.	Value per Lb.		
1900...............	11,391,666	5,167	$496,380	$.044	$99,288	$2,813,361
1901...............	9,122,283	4,138	393,259	.043	82,046	764,211
1902...............	10,716,364	4,861	433,722	.040	114,197	848,476
1903...............	14,429,885	6 544	578,215	.041	71,354	812,948
1904...............	16,313,826	7,399	628,494	.039	117,957	1,840,941
1905...............	22,559,625	10,236	810,203	.036	159,995	1,652,452
1906...............	31,156,616	14,129	1,149,297	.037	204,269	1,937,092
1907...............	26,512,920	12,023	1,069,924	.040	186,203	1,331,321

RE-EXPORTS OF FOREIGN PRODUCTS. (*a*)

Year.	Antimony.				Antimony Ore.			
	Lb.	Metric Tons.	Value.	Value per lb.	Short Tons.	Metric Tons.	Value.	Value per Sh. T
1900..................................	23,520	10.7	$2,352	$0.100	*Nil.*	....	....	....
1901..................................	*Nil.*	....	....	....	25	22.1	$1,536	$63.05
1902..................................	37,184	16.9	2,710	.073	104	94.6	4,602	44.13
1903..................................	79,917	36.0	4,478	.056	*Nil.*	....	....	....
1904..................................	31,077	14.0	1,734	.056	214	194.0	10,775	50.35
1905..................................	*Nil.*	....	....	....	*Nil.*	....	....	....
1906..................................	24,892	11.2	4,939	.118	*Nil.*	....	....	....
1907..................................	47,999	21.8	9,064	.189	6	5	273	45.50

Year.	Asphaltum, Crude.				Brass and Mfres.	Cement.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.		Bbl. (*i*)	Metric Tons.	Value.	Value per Bbl.
1900..................	629	639	$10,044	$15.98	$2,155	39,540	7,174	$63,880	$1.62
1901..................	2,209	2,244	18,078	8.19	813	43,691	7,927	72,761	1.67
1902..................	2,930	2,977	23,564	8.11	938	32,594	5,913	48,797	1.50
1903..................	1,605	1,631	13,894	8.66	7,576	25,362	4,601	32,156	1.27
1904..................	1,887	1,917	26,272	13.92	2,517	39,711	7,186	54,486	1.37
1905..................	1.081	1,098	18,190	16.83	1,228	31,874	5,782	40,583	1.27
1906..................	1,765	1,793	22,324	12.64	....	16,216	2,941	19,487	1.20
1907..................	8,288	8,421	31,749	3.83	....	20,697	3,754	30,435	1.47

Year.	Chemicals.											
	Salts of Potassium. (*f*)				Chloride of Lime.				Nitrate of Sodium.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900...	808,701	366,824	$43,524	$0.054	148,116	67,185	$1,987	$0.014	3,089	3,139	$112,550	$36.43
1901...	633,100	287,182	43,446	.068	13,916	6,312	312	.023	2,482	2,519	101,489	40.90
1902...	1,266,145	574,323	59,789	.048	198,794	90,172	2,997	.014	3,675	3,734	144,650	39.36
1903...	1,299,905	589,637	33,264	.026	836,411	379,696	7,609	.009	4,417	4,488	184,657	41.81
1904...	1,262,222	572,544	33,358	.027	1,434	650	13	.009	6,076	6,173	279,864	46.06
1905...	3,053,191	1,386,149	83,652	.027	100	102	3	.003	8,991	9,135	420,613	46.78
1906...	2,264,175	1,027,935	77,043	.034	*Nil.*	....	....	....	6,660	6,767	324,915	48.78
1907...	2,675,248	1,285,892	75,470	.028	*Nil.*	....	....	....	7,159	7,274	370,048	51.69

Year.	Chemicals. (*Continued.*)											
	Caustic Soda.				Soda Ash and Carbonate.				Sodium Salts, All Other.			
	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb.	Lb.	Kg.	Value.	Value per Lb
1900......	1,139,954	517,080	$24,228	$0.023	78,017	35,388	$1,126	$0.014	270,307	122,610	$2,788	$0.009
1901......	1,001,940	452,482	21,511	.023	369,521	167,614	5,184	.014	133,400	60,510	3,398	.023
1902......	1,343,132	609,246	28,704	.023	62,653	28,419	931	.014	115,491	52,386	1,626	.014
1903......	1,116,354	506,378	23,227	.028	30,030	13,622	464	.015	42,540	19,294	437	.010
1904......	1,115,600	506,036	23,608	.021	40,351	18,303	593	.014	1,778,616	806,780	25,312	.014
1905......	1,087,772	493,848	22,728	.021	32,221	14,628	473	.015	16,748	7,604	177	.011
1906......	(*l*)	....	....	....	2,486	1,128	41	.016	1,032,372	468,180	21,624	.021
1907......	(*l*)	....	....	....	3,100	1,406	53	.017	742,201	336,662	16,099	.022

Year.	Clay or Earths.				Coal, Bituminous.				Copper.			
									Ore and Matte.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900........	78	79	$572	$7.34	6,740	6,848	$19,740	$2.93	964	979	$170,191	$176.53
1901........	80	81	825	10.34	3,796	4,403	10,627	2.45	9,891	10,050	1,406,648	142.19
1902........	123	125	1,284	10.43	7,559	7,680	22,153	2.93	14,446	14,657	2,229,912	154.57
1903........	88	89	621	7.06	88,468	89,883	453,613	5.13	5,750	5,232	852,726	165.58
1904........	210	214	2,466	11.74	7,250	7,366	21,910	3.02	*Nil.*	....	....	....
1905........	102	104	954	9.35	3,945	4,008	10,974	2.78	*Nil.*	....	....	....
1906........	140	142	1,291	9.22	2,541	2,582	13,062	5.14	71	72	29,791	419.59
1907........	122	124	1,305	10.70	1,947	1,978	12,199	6.26	....	....	....	....

Year.	Copper. (*Continued.*)					Earthen, Stone and China Ware	Fertilizers. Total Value.	Glass and Glassware.	Graphite.	
	Pigs, Bars, Ingots, Old and All Unmanufactured.				Mfres.					
	Lb.	Metric Tons.	Value.	Value per Lb.					Long Tons.	Value.
1900...	1,281,782	581	$212,264	$0.166	$21,032	$38,008	$32,102	$14,614	3	$115
1901...	12,888,083	5,846	2,145,468	.166	9,462	24,080	2,833	16,749	*Nil.*	....
1902...	11,629,877	5,275	1,604,522	.138	10,939	18,989	31,476	34,236	12	834
1903...	2,093,103	949	261,413	.125	13,027	19,411	3,281	19,116	63	4,223
1904...	1,088,672	494	140,695	.129	19,461	32,640	139,363	20,522	8	455
1905...	1,718,584	780	272,945	.159	12,621	30,455	8,984	34,552	5	91
1906...	1,567,782	711	309,605	.197	16,589	41,151	15,568	19,965	3	362
1907..	995,555	451	199,828	.201	1,188	36,538	14,799	15,064	1	41

Year.	Iron and Steel.											
	Pig Iron.				Scrap.				Bar Iron.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	151	153	$6,579	$43.69	9,079	9,224	$131,241	$14.46	48	49	$2,447	$51.00
1901....	189	191	6,148	32.70	3,331	3,384	51,663	18.51	67	68	7,569	113.00
1902....	250	254	6,286	25.14	1,542	1,567	25,020	a 16.23	22	22	1,875	86.56
1903....	1,863	1,893	33,996	18.25	262	266	2,862	10.92	16	16	2,108	130.93
1904....	1,646	1,672	25,910	15.74	190	193	2,367	12.46	7	7	765	102.55
1905....	1,010	1,026	29,047	28.76	4,270	4,338	80,623	18.88	22	22	2,556	118.18
1906.. .	6,750	6,858	236,957	35.10	5,111	5,193	101,886	19.93	61	62	7,207	118.15
1907....	2,921	2,968	86,420	29.59	157	160	3,378	21.52	38	39	3,959	104.18

Year.	Iron and Steel. (*Continued.*)											
	Rails.				Steel, Ingots, Blooms, Etc.				Sheets, Plates, Rods, Wire.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
9000 ...	*Nil.*	....	....	....	2	2	$1,342	$559.33	209	213	11,599	$55.41
1901....	*Nil.*	....	....	....	2	2	1,059	504.30	190	193	17,272	90.81
1902....	297	302	$7,184	$24.20	106	108	6,774	64.09	236	240	14,221	60.26
1903....	739	751	17,560	23.76	60	61	5,316	83.01	55	56	5,532	100.04
1904 ..	96	98	2,305	24.00	40	41	6,208	154.81	108	110	6,482	60.01
1905....	31	31	1,132	36.52	86	87	15,570	181.05	161	164	8,019	49.81
1906....	Nil	....	....	....	196	199	14,104	71.96	318	323	27,631	86.89
1907....	Nil	....	....	....	292	297	25,974	88.95	73	13	1,220	93.84

Year.	Iron and Steel. (*Concluded.*)					Lead and Mfres.	Marble and Stone Mfres (*h*)	Metal Composition.
	Tin and Terne Plates, Taggers Tin.				Mfres.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.				
1900..	464	470	$37,395	$80.60	$328,704	$3,843,881	$5,721	$79,218
1901..	118	120	8,519	72.20	149,771	4,190,525	17,063	35,438
1902..	98	100	7,471	76.24	242,225	3,553,144	11,210	108,575
1903..	2	2	184	118.71	399,147	2,917,957	7,389	54,490
1904..	81	82	5,306	65.86	518,564	2,880,907	21,055	42,730
1905..	26	26	3,014	115.92	544,321	2,441,166	13,653	92,227
1906..	0.4	0.4	28	70.00	200,489	2,307,345	20,966	85,955
1907..	42.2	42.9	1,813	42.96	176,987	2,421,284	17,542	66,068

Year.	Paints and Colors.	Salt.				Sulphur—Crude.			
		Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900.....	$13,814	3,548,724	1,610	$3,907	$0.0011	590	599	$13,495	$22.89
1901.....	17,923	3,699,411	1,678	7,155	.0019	207	210	5,086	24.60
1902.....	14,217	2,310,759	1,048	4,544	.0020	1,253	1,273	28,024	22.37
1903.....	13,467	7,804,215	3,585	26,636	.0034	967	982	22,658	23.43
1904.....	11,888	2,089,234	948	2,814	.0013	2,493	2,533	58,887	23.62
1905.....	14,227	611,912	278	893	.0015	1,713	1,741	36,858	21.52
1906.....	12,775	1,462,413	663	1,129	.0008	403	409	8,475	21.03
1907.....	11,050	1,166,049	529	1,686	.0014	301	306	5,759	19.13

Year.	Tin in Blocks, Pig and Granulated.				Zinc and Mfres.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	
1900	495	503	$335,377	$677.96	$3,048
1901	939	954	562,350	598.89	1,641
1902	479	486	286,897	598.95	765
1903	512	520	317,805	620.47	2,362
1904	519	527	322,234	620.87	1,236
1905	557	567	375,763	674.62	1,831
1906	807	820	650,411	805.96	2,214
1907	562	571	492,415	876.18	2,527

* For convenience in tabulating, the quantities of all petroleum products and their gross values have been divided by 1,000

(*a*) From Summary of Commerce and Finance of the United States.

(*c*) Total exports of coin and bullion; that is, includes both foreign and domestic.

(*d*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte, which is classified as copper ore, and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin.

(*e*) Includes nickel oxide and nickel matte.

(*f*) Includes chlorate, chloride, nitrate and all other salts of potassium.

(*g*) Reported in barrels, but calculated to gallons, on a basis of 42 gallons to the barrel.

(*h*) Includes slate.

(*i*) Barrel of 400 lb.

(*k*) Does not include 5,125,103 tons of coal valued at $16,202,150, used for fuel on vessels for foreign trade.

(*l*) Included in all other salts of soda.

(*m*) Does not include 5,754,176 tons of coal valued at $18,523,594 used for fuel on vessels for foreign trade.

INDEX.

Hydrometallurgy of Silver

By Ottokar Hofmann

336 *pages, 6x9, illustrated,* \$4.00 (17*s.*) *postpaid.*

(*Published March,* 1907.)

Chloridizing roasting of silver ores, hyposulphite lixiviation in its various forms, the extraction of silver from cupriferous materials, the manufacture of copper sulphate, and the cyanide process as applied to silver ores, are treated here.

The portion devoted to the cyanide process is alone worth the price of the book to anybody interested in the metallurgy of silver. Mr. Hofmann has made a long study of this subject, is an eminent metallurgist and a recognized authority.

Tulley's Handbook on Engineering

Steam and Electrical

A Big Reference Library in a One Thousand Page Pocket Book and all for \$3.50 (15*s.*) *postpaid.*

(*Revised to* 1907.)

It gives the art of erecting engines and setting them in line so as to work smoothly and correctly.

The setting of valves on single and double eccentric long-range Corliss Engines, and setting the valves on all kinds of engines.

The operation, care and management of steam pumps; how to set the valves on the different kinds of steam pumps, also the care and handling of injectors and inspirators.

Direct currents for lighting, alternating currents for arc-lighting, two and three-phase currents fully illustrated and explained with about fifty illustrations and seventy-five pages.

Care and management of steam engines and boilers. Art of erecting shafting and pulleys. Application of electricity to purposes of lighting and motive power, etc., etc.

Engine Room Chemistry

By Prof. August H. Gill

Of the Massachusetts Institute of Technology.

230 *pages,* $4\frac{1}{2}$*x*$6\frac{1}{2}$, *fully illustrated,* \$1.00 (4/6) *postpaid.*

(*Published December,* 1907.)

A practical handbook describing the apparatus and tests in the engine-room. No knowledge of chemistry is assumed yet the necessary test for the analysis of fuels, flue gases, lubricants and for the neutralization of water and for efficient combustion are given.

Hill Publishing Company 505 PEARL ST., NEW YORK
6 Bouverie St., London, E. C.

The Engineering and Mining Journal *Power* *American Machinist*

Standard Mining and Metallurgical Books

The Nature of Ore Deposits
Dr. Richard Beck

Professor of Geology and Economic Geology, Mining Academy of Freiberg. Translated and revised by Walter Harvey Weed, U. S. Geological Survey.

Two volumes, 6x9, illustrated and with maps, price per set, $8.00 postpaid.

Practical Notes on the Cyanide Process
Francis L. Bosqui, Ph.B.

Superintendent of the Standard Consolidated Mining Co.'s Cyanide Works, Bodie, Cal.

201 pages, $5\frac{3}{4}$x9, illus., $2.50 postpaid.

Winding Plants for Great Depths
Hans C. Behr

$5\frac{1}{8}$x$8\frac{1}{2}$, with many plans and working drawings, $12.50 postpaid.

Report Book for Mining Engineers
A. G. Charleton

Instructions, together with a series of blanks for the reports of mining engineers, made in pocket size, bound in flexible red Morocco, $2.00 postpaid.

The Chemistry of Cyanide Solutions
Resulting from the Treatment of Ores
J. C. Clennell

164 pages, 6x9, cloth, illus., $2.50 postpaid.

Gold Mines of the World
J. H. Curle

$7\frac{1}{4}$x$9\frac{5}{8}$, cloth, fully revised and illustrated, $5.00 postpaid.

Industrial Furnaces
Prof. Emilio Damour

Translated and augmented by A. L. Queneau, Consulting Engineer of the N. J. Zinc Co.

317 pages, 6x9, illustrated, $4.00.

The Elements of Mining and Quarrying
Sir C. LeNeve Foster, D.Sc., F.R.S.

$5\frac{1}{4}$x$7\frac{5}{8}$, cloth, $2.50 postpaid.

Notes on Lead and Copper Smelting and Copper Converting
Hiram W. Hixon

$6\frac{1}{8}$x$9\frac{1}{8}$, $3.00 postpaid.

The Metallurgy of Lead
H. O. Hofman

Prof. of Metallurgy, Mass. Inst. of Technology.
600 pages, 6x9, $6.00 postpaid.

Hill Publishing Company
505 PEARL ST., NEW YORK
6 Bouverie St., London, E. C.

The Engineering and Mining Journal *Power* *American Machinist*